Earl R. Pilgrim.

GENERAL METALLURGY

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BY

H. O. HOFMAN, E. M., Met. E., Ph. D.
PROFESSOR OF METALLURGY, MASSACHUSETTS INSTITUTE OF TECHNOLOGY

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THIS BOOK IS DEDICATED

TO

R. H. RICHARDS, S. B., LL. D.

TEACHER, INVESTIGATOR, FRIEND

PREFACE

Metallurgical literature contains many treatises and monographs dealing with the metals proper and parts of metallurgical activity, but not a single work upon general metallurgy which meets present requirements. Of the modern books on general metallurgy, the earliest is that of Grüner (1875, 1878) which is a model of clear exposition. In Kerl's condensed encyclopedia (1879) the reader will find references to practically everything of value that had appeared in print up to 1879. Schnabel's treatise (1903) assembles to some extent the work of his predecessors, and adds valuable chapters upon electro-chemical processes, but passes over lightly the properties of metals and alloys, and does not interpret metallurgical phenomena from a modern point of view. The last treatise is that of Babu (1904, 1906) which unfortunately was not finished by the author. The works of Roberts-Austen and Harbord (1910) and of Fulton (1910) do not aim to cover the whole ground. They deal in a masterly fashion with many important phases of metallurgy, laying stress upon physical and physico-chemical phenomena.

In the present treatise the aim has been to cover the field of General Metallurgy as a whole. The good of the older endeavor has been combined with that of modern research, and the whole has been treated from the point of view of the metallurgist who has a leaning toward physical chemistry. Mechanical processes have received more consideration than has been customary, as the tendency of present metallurgical practice is in this direction. Technical literature has been drawn upon freely. It is believed that reference to most books and papers of importance will be found in the foot-notes, at the proper places, even if the contents of the publications have not been used in the text.

In a treatise like the present, the author has to cover a large field of technical endeavor with which he cannot well be as familiar as he would like to be. He has to seek assistance, if he is to succeed at all. For this reason, the thermo-chemical data have been taken bodily from Professor J. W. Richards' standard "Metallurgical Calculations," and many illustrations borrowed from Professor R. H. Richards' great treatise on "Ore Dressing." In dealing with some of the several subjects, I have been helped by my colleagues at the Institute of Technology; thus in mechanics by Professor C. E. Fuller, in pumps by Professor E. F. Miller, in electro-chemistry by Professor M. DeK. Thompson, and in the constitution of alloys as well as in thermo-chemical calculations by Professor C. R. Hayward. I want to express here my thanks to these gentlemen.

It has taken a great deal more time to write this book than was anticipated. The large amount of labor involved in such an undertaking may be the reason why we have no modern general metallurgy. The future will show whether the right selection has been made between practice and theory.

H. O. Hofman.

Massachusetts Institute of Technology,
Boston, Mass.,
July, 1913.

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GENERAL METALLURGY.

CHAPTER I

INTRODUCTION

1. **Definition and Scope.**—Metals form a group of elementary substances which possess a certain kind of luster, opacity, high conductivity for heat and electricity, and more or less plasticity. Metallurgy is the art of extracting metals from their ores and refining and fashioning them for use in metal industries. The subject is divided into three parts:

GENERAL METALLURGY, which discusses in a general way the history of metals, the physical and chemical properties of metals, alloys and metallic compounds, the metallurgical processes, ores, fluxes, apparatus, and metallurgical products, and in detail, fuels and refractory materials.

NON-FERROUS METALLURGY, which discusses in detail the treatment of ores of copper, lead, silver, gold, platinum, quicksilver, zinc and cadmium; tin, arsenic, antimony, bismuth, nickel and cobalt; aluminum, uranium, tungsten, manganese, chromium, molybdenum, titanium, vanadium, and other rare metals.

FERROUS METALLURGY, which discusses in detail the manufacture of iron and steel.

It is essential for the successful study of metallurgy that the student be well versed in mathematics, physics, chemistry and mineralogy, and that he should have a general grounding in mechanical and electrical engineering and some knowledge of business affairs.

2. **Historical Notice.**[1]—The degree of civilization is closely connected with the use of metals, as the more extensively a race employs metals, the greater will be their importance. It is therefore to be expected that when a race had sufficiently grown away from savagery to leave a record of its existence we should find some mention of the metals which it was using. The remains of prehistoric races show that the people were acquainted with gold, silver, iron, copper, tin and lead. The order in which these metals came into use must have varied with different peoples. It is now generally believed that gold was the metal first known, as it occurs in the metallic state, is bright and heavy and easily worked. In a few regions of the world, copper occurs in the native state,

[1] Karsten, C. J. B. "System der Metallurgie," Berlin, 1831, Vol. 1.
Hoefer, F. "Histoire de la Chimie," Paris, 1842.
Kopp, H. "Geschichte der Chemie," Brunswick, 1843.
Zippe, F. H. M. "Geschichte der Metalle," Vienna, 1857.
Rössing, A. "Geschichte der Metalle," Berlin, 1901.
Neumann, B. "Die Metalle, Geschichte, Vorkommen und Gewinnung, nebst ausführlicher Productions- und Preiss-Statistik," Berlin, 1904.

requiring only forging to be converted into a useful tool. Meteoric iron may have been the first source of iron implements. The former assumption, however, that a bronze-age, following the stone-age, always preceded an iron-age has been given up by archæologists and ethnologists. That iron should as a rule have been used earlier than copper or bronze may be inferred from the infrequent occurrence of copper and tin in comparison with iron, and from the metallurgical fact that iron oxides are readily reduced to forgeable metallic iron, while copper oxides after reduction have to be fused to furnish a serviceable metal. The Greeks and Romans knew quicksilver in addition to the above-mentioned metals. These seven metals remained the only ones known until the fifteenth and sixteenth century when antimony, bismuth and zinc were recognized; at the close of the seventeenth century arsenic was added to the list, and in the eighteenth century, nickel cobalt, manganese and platinum; the other metals belong to the nineteenth century.

The history of metallurgy may be divided into four great periods. The first extends from the oldest times to the first century, A. D., when Pliny the Elder brought together in his "Naturalis Historia" everything known at his time about metals. The second period extends from the first to the beginning of the sixteenth century. From Agricola's "De Re Metallica"[1] (1530) we see that many of our present methods of ore-treatment are only mechanical improvements on those in use at his time. The third period, from the beginning of the sixteenth to that of the nineteenth century, shows little progress over that of the second. During the first quarter of the nineteenth century we enter upon the fourth period, that of our present metallurgy, with the application of the sciences to the art, and the evolution of a science of its own. The works of Schlüter (1738), Calvör (1763), Lampadius (1830) and Karsten (1831) are the leading records of the progress made. Since Karsten's masterly treatises were published the metallurgical literature has become too extensive to be quoted in this general summary.

3. **Bibliography.**—The following are the leading modern treatises on general metallurgy.

Grüner, L. Traité de Métallurgie, Dunod, Paris, Vol. I, 1875, Vol. II, 1878.

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Schnabel, C. Lehrbuch der Allgemeinen Hüttenkunde, Springer, Berlin, 1903.

Babu, L. Traité Théoretique et Pratique de Métallurgie Générale, Béranger, Paris, Vol. I, 1904, Vol. II, 1906.

Roberts-Austen W., and Harbord, F. W. An Introduction to the Study of Metallurgy, Griffin & Co., London, and Lippincott Co., Philadelphia, 1910.

Fulton, C. H. Principles of Metallurgy, McGraw-Hill Book Co., New York, 1910.

[1] Translated by C. and L. H. Hoover, The Technical Book-Shop, Salisbury House, London, E. C., 1913.

CHAPTER II

PROPERTIES OF METALS

4. **Density.**—A characteristic of the industrial metals is their high specific gravity. With the exception of Al and Ti their specific gravities are higher than 6; the lightest metal is Li (0.59), the heaviest Os (22.48). Variations in the specific gravity of an industrial metal are caused by its degree of purity and the compactness of its structure, the latter being modified by the manner of casting, the rate of cooling, and mechanical treatment. It is difficult to produce a metal which is absolutely pure; the specific gravity of a commercial metal will differ from that which is chemically pure, as it always contains some impurity. In casting a metal under a head, the molecules will be the more compacted the greater the pressure, and this increases the specific gravity. The density of tin depends on the rate of cooling; quick cooling gives a grouping of the particles and density different from that given by slow cooling.

Table 1.—Specific Gravities of Some Metals

Os	22.48	Cu, deposited	8.92	Sn, disintegrated by cold	5.87
Ir	22.42	Cu, molten	8.22	Zn, mean	7.15
Pt, cast	21.50	Cu, cast	8.80	Zn, molten	6.48
Pt, sheet, wire	21.2–21.7	Cu, wire	8.94	Zn, cast, slowly cooled	7.13
W	19.10	Ni	8.80	Zn, cast, quickly cooled	7.09
Au, cast	19.25	Cd	8.60	Zn, rolled	7.19
Au, hammered	19.35	Cd, molten	7.99	Cr	6.52–6.73
Hg, liquid	13.59	Cd, hammered	8.67	Sb	6.71
Hg, solid	14.19	Mo	8.60	Te	6.25
Rh	12.10	Co	8.50–8.80	As, molten under pres.	5.71
Pb	11.37	Mn	7.50	As, crystallized	5.73
Pb, molten	10.64	Fe	7.86	As, amorphous	4.71
Pb, cast	11.35	Fe, molten	6.88	Va	5.50
Pb, rolled	11.38	Fe, wrought iron	7.80–7.90	Ti	5.30
Pd	11.40	Fe, steel	7.40–7.80	Al	2.60
Ag, deposited	10.53	Fe, cast white	7.58–7.73	Al, cast	2.56
Ag, molten	9.51	Fe, cast gray	7.03–7.13	Al, rolled	2.66
Ag, cast	10.75	Sn, cast	7.29	Al, commercial	2.58
Ag, compressed	10.56	Sn, molten	7.09	Mg	1.74
Bi	9.82	Sn, hammered	7.30	Li	0.59
Bi, molten	10.04	Sn, crystal	7.07		

Hammering, rolling and wire-drawing were once supposed to increase the specific gravity by forcing the molecules closer together. Up to a certain limit the application of pressure does increase the specific gravity of a metal by closing up cavities. Thus Spring[1] found that with Pb, Sn, Bi, Sb, Cd, Al and Zn a pres-

[1] *Bull. Soc. Chim.*, 1883, VI, 515.

sure of 20,000 atmospheres continued for several days was sufficient to close the pores. Further compression can have no effect upon the specific gravity unless it causes a molecular re-arrangement of particles. When the form of a metal is changed by mechanical treatment, the specific gravity of the metal is actually diminished.[1] Annealing more or less obliterates the effect. Merely stretching a metal (wire) may diminish the specific gravity by tending to develop cavities.[2] In the liquid state all metals are lighter than when in the solid; the only exception is Bi,[3] which attains its highest specific gravity just before it solidifies.

The specific gravity of some molten metals has been investigated by Mallet,[4] Roberts-Austen,[5] Wrightson,[6] Roberts-Austen and Wrightson,[7] and Nies and Winkelmann.[8]

5. Luster and Transparency.—Most metals have a high luster which is very characteristic and has been termed metallic. This power to reflect light[9] in great perfection is seen especially on burnished or polished surfaces of the harder metals, or metals in the liquid state. Rough surfaces or fractures show luster only when the grains composing them have large crystal facets; finely divided metal is dull.

Under ordinary conditions metals are opaque. Faraday observed in 1857[10] that gold leaf ($\frac{1}{300000}$ in. thick) supported on a glass plate transmitted green light, and that silver resembled gold in these respects.[11] When heated to about 250° C. the gold leaf becomes transparent and transmits white light instead of green. This is due to the aggregating of particles of gold into globules between which the light passes. Silver begins to show a similar behavior with a film $\frac{1}{800000}$ in. when heated in air to 335° C., the change being assisted by a momentary oxidation. Films of copper heated to a suitable temperature become transparent and transmit an emerald green light which becomes darker as the oxidation progresses until the brilliant color has been changed into a black cloud.

6. Color and Odor.—The color of finished metals is of less importance to the metallurgist than to the metal manufacturer. In the compact state the colors are white (Ag); silvery (Pt, Al, Sn, Ni, Hg, Cd); bluish (Sb, Zn, Pb); pinkish (Bi); gray (Fe, As); yellow (Au), and red (Cu). In a finely-divided

[1] Kahlbaum-Roth-Siedler, *Zt. anorg. Chem.*, 1902, XXII, 277.
Kahlbaum-Sturm, *op. cit.*, 1905, XLVII, 217.
Hibbard, *Tr. A. I. M. E.*, 1906, XXXVII, 371.

[2] Gray, *Proc. Roy. Soc.*, Ser. A, 1893, LIV, 284.

[3] Lüdeking, *Ann. Phys. Chem.*, 1888, XXXIV, 21.

[4] *Proc. Roy. Soc.*, Ser. A, 1873–74, XXII, 366; 1874–75, XXIII, 209.

[5] *Proc. Roy. Soc.*, Ser. A, 1874–75, XXIII, 481.

[6] *J. I. and St. I.*, 1879, II, 418; 1880, I, 11.

[7] *Phil. Mag.*, 1881, XI, 295; 1882, XIII, 360.

[8] Sitzungsberichte der k.b. Akademie der Wissenschaften zu München, 1881, XI, 63.

[9] Coblentz, The reflecting power of various metals, *J. Frankl. Inst.*, 1910, CLXX, 169.

[10] *Phil. Trans.*, 1897, CXLVII, 145.

[11] Beilby, *Proc. Roy. Soc.*, 1905, LXXVI, Ser. A, 462; 1907, LXXIX, Ser. A, 463; Turner, *op. cit.*, 1908, LXXXI, Ser. A, 301; Horton, *Cass. Mag.*, 1908–09, XXXV, 735; *Met. Ind.*, 1909, VII, 97.

state, *e.g.*, when precipitated from solutions, the color is liable to be different from that of the compact metal (the gray of Ag, the brown of Au). In the fluid state it is different from that in the solid (the green of Cu and of Au).

Odor is usually not associated with metals, but Cu and Sn rubbed with the hands give off characteristic disagreeable odors.

7. **Crystallization, Cleavage.**[1]—All metals tend to assume crystalline forms when they change from the state of vapor to that of solid, from the liquid to the solid state, and from one solid form to another, *i.e.*, they tend toward a state in which the molecules have definite geometrical positions. Microscopic research has proved that metals solidified from the molten condition do not exist in the non-crystalline amorphous state, and that they retain their crystalline structure when they are severely strained.

Crystallization produces regularity of internal structure, and this may cause a metal to separate along certain crystallographic planes instead of fracturing irregularly. These cleavage planes represent directions of weakness in the crystal. Crystallization in the change from the state of vapor to that of solid is well exemplified by the sublimation of Mg and As which collect in rhombohedral crystals in the neck of the retort; the crystals of Mg are especially well developed, while As generally forms a finely crystalline aggregate. In precipitating Ag from a sulphate solution by means of Cu or Fe, the Ag comes down in the form of crystals, the sizes of which are governed by the degree of concentration and the temperature of the solution. In the electrolytic parting of doré silver with $AgNO_3$ as electrolyte the Ag is plated out at the cathode with a current of high density in the form of loose crystals, while with a weak current the deposit forms a crystalline coating. In dissolving Ir or Os in melted tin and removing, after cooling, the tin with HCl, Ir or Os will remain behind in the form of a powder consisting of well-developed crystals. Cr dissolved in molten Zn will be found in the form of small pointed rhombohedrons after the metallic solvent has been removed with dilute HNO_3.

The change of a metal from the molten to the solid state[2] is one of considerable importance on account of the effect upon its physical properties. The crystallization of a molten metal follows in general the same laws that govern the crystallization of a salt from its solution. It begins as a rule on the surface of the solidifying metal and progresses rapidly along the angles and edges of the primary crystals; the slower and the less interrupted the cooling, the more

[1] Lehmann, O., "Molecularphysik," Engelmann, Leipsic, 1888, I, 291.

Behrens, H., "Die mikroskopischen Gefüge der Metalle und Legirungen, Hamburg-Leipsic, 1894, p. I.

Ewing and Rosenhain, *Phil. Trans. Roy. Soc.*, Ser. A, 1900, CXCIII, 353; 1901, CXCV, 279; *Metallog.*, 1900, III, 94; 1902, V, 81.

Wingham, *J. I. and S. I.*, 1901, II, 272.

Beilby, *Rep. Brit. Assoc.*, 1901, 604; *J. Soc. Chem. Ind.*, 1903, XXII, 1166; *Proc. Roy. Soc.*, 1904, LXXII, 218, 1905, LXXVI, 462.

[2] Osmond-Cartaud, *Rev. Mét.*, 1907, IV, 819.

Beilby, *J. Inst. Met.*, 1912, VIII, 186.

perfect will be the development of the crystals. The crystals first formed usually lie parallel with or at right angles to the cooling surface. When once started, crystallization proceeds very rapidly, as the molecules of the crystallizing metal lie very near to one another and furnish the necessary material so rapidly that there is not sufficient time for particles to arrange themselves evenly and normally on the growing crystal. The forms are generally small and not well developed, and the faces are curved. The convection currents set up by the rapid movement of material toward the crystal have a decided effect in determining the final shape of the crystal. In the case of simple crystals, such as the skeleton cubes of Bi, the edges receive the material so rapidly that hopper-shaped crystals result. With complex forms, the rapid movement of the particles interferes with their normal orientation, so they drop into the next available crystallographic position, that of a twin crystal. Repeated parallel growths, accompanied by repeated twinning,[1] and aided by facial distortion and other irregularities incident to rapid growth are the causes which give rise to the greater part of the branching, reticulated, dendritic forms characteristic for metals.

Bars of refined Pb and Sn upon solidifying show characteristic forms darting out from the surfaces; the star of Sb forms under a cover of slag if the pure metal is poured and cooled in the right way. Crystal forms of Cd, Al, Sb, Ag, Cu, Pb, Zn, Bi and Sn are shown by Campbell.[2] The fracture of a cake of Zn shows how the crystals have grown perpendicularly to the surface toward the center; polished and etched surfaces of cast rods of Pb, Sn and Sb show a radial structure.

In some instances, as, *e.g.*, is often the case with Cu, crystallization starts from several centers in the liquid metal at the same time as at the surface, or it starts even before the surface begins to solidify. In such a case, there will be formed groups of crystalline aggregates separated by narrow, apparently amorphous zones in which the crystallographic forces have been nearly overcome by mutual interference. Crystallization may be further interfered with by the presence of impurities which being thrown out of solution remain irregularly disseminated through the mass or deposited upon the crystal faces in the form of cementing films.

The molecular changes in the solid metal (allotropic changes) are well shown in the behavior of Fe, Ni, Zn, Sn, Pb, Ag, Sb, As, Se, Te, Ir, C, P, S, etc. Melted Fe[3] upon solidifying assumes the form of γ-iron, and upon further cooling changes at 860° C. to β-iron and at 750° C. to α-iron, each showing characteristic crystal forms; Ni undergoes a molecular change at 340° C.; Zn at 360° C.; Sn, when solidifying in the ordinary way forms tetragonal crystals; when it is cooled very slowly so as to become undercooled, it assumes orthorhombic forms; when it is cooled to −40° C. it loses its luster, becomes gray and the

[1] Edwards, *Int. Zt. Metallographie*, 1912, III, 179.

[2] *J. Frankl. Inst.*, 1902, CLIV, p. 136, Plate III, Figs. 9 to 15; *Proc. Am. Soc. Test. Mat.*, 1904, IV, 382; Appendix IV, *Sixth Report Alloys Research Committee*, 1904.

[3] Osmond-Cartaud, *J. I. and St. I.*, 1906, III, 444.

metal crumbles to a fine powder having a specific gravity of 5.87 while ordinary Sn has a specific gravity[1] of 7.07–7.29.

Isometric crystal forms are found in Pt, Pd, Au, Ag, Cu, Pb, Fe, Ni, Cr, Zn, Hg, Ir, Os, Ti, C and Si; tetragonal and orthorhombic forms with Sn; hexagonal forms with Sb, As, Te, Bi; Zn, Cd; Sn and Zn are dimorphous. Metals which crystallize readily are usually weak.

Cleavage is found with a few metals only, as in Sb,[2] Bi, Te, Zn.

8. Structure.[3]—The structure of a metal is studied by examining both the fracture (macrostructure) of a specimen with the naked eye, with a magnifying glass, and the polished and etched or heat-tinted surface (microstructure) with a microscope. In breaking[4] a piece of metal, the rupture will follow the lines of least resistance. These may lie wholly along the faces of the crystalline grains, or wholly through the grains, or in part along, in part through, the grains. The fracture may be columnar, fibrous, hackly, splintery, radiated, lamellar, granular, crystalline, conchoidal, porcelanic, etc. The fracture varies, however, not only with the kind of metal, but with the manner of rupturing and with the thermal and mechanical treatment a metal has undergone.[5]

When a bar of wrought iron is nicked on one side only and then broken by bending over with successive blows, the fracture will be fibrous; when it is nicked on four sides and then struck with a sharp blow, the fracture will be crystalline to granular. That the rate of cooling should have an effect upon the form, size and strength of a grain and upon the adhesion of the single grains is clear. While fracture gives only an indication of the structure of a metal, it has great practical value in assisting to observe the changes a metal is undergoing or has undergone in its metallurgical treatment. The study of fracture may be assisted by etching.[6]

The ultimate structure[7] of a metal depends upon the crystallinity (degree of crystallization), the relative size and the fabric (orientation, general arrangement, net-work) of the single grains. It is revealed by the microscopic examination of polished and otherwise suitably prepared surfaces which are illuminated by reflected light. The study originated with Sorby of Sheffield in 1864; in 1878 Martens of Berlin investigated the microscopic structure of iron and steel; in France, Osmond and Werth were the pioneers in 1885.

[1] Allotropic changes of Tin: Cohen-van Eyck, *Zt. phys. Chem.*, 1899, XXX, 601.
Cohen, *op. cit.*. 1900, XXXIII, 57; 1900, XXXV, 588; 1901, XXXVI, 513.

[2] Campbell, *Proc. Am. Soc. Test. Mat.*, 1904, IV, 382; *Proc. Engl. Inst. Mech. Eng.*, June, 1904.

[3] Ewing, *Rep. Brit. Assoc.*, 1906, 657; *Eng. Rev.*, 1906, XV, 205; *Phil. Trans.* Roy. Soc., 1900, CICIII, 353; 1901, CICV, 279; *J. Inst. Met.*, 1912, VIII, 4; *Engineering*, 1912, XCIII, 651.

[4] Martens, A., and Henning, Gus. C., "Handbook of Testing Materials," New York, 1899, p. 101.

[5] Osmond-Frémont-Cartaud. *Rev. Mét.*, 1904, I, 11. Freminville, *op. cit.*, 1907, IV, 833.

[6] Heyn, *Metallurgie*, 1907, IV, 119.
Ast., *op. cit.* 123.
Frémont, *Rev. Mét.*, 1908, V, 649.
Heyn, *Stahl u. Eisen*, 1908, XXVIII, 1827.
Kruft, *op. cit.*, 1909, XXIX, 517.

[7] Le Chatelier, *Rev. Mét.*, 1906, III, 493.

The microscopic examination of metals and alloys has become an independent art and science, and forms an important branch of metallography, the study of metals and alloys. Its technic is discussed in a number of papers and treatises.[1]

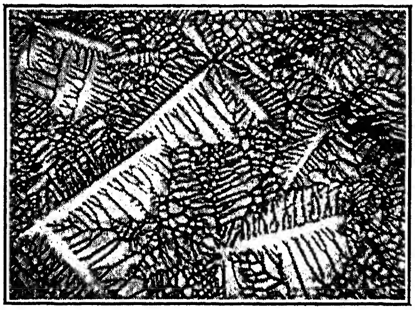


FIG. 1.—Surface of cast of aluminum, 30 d.

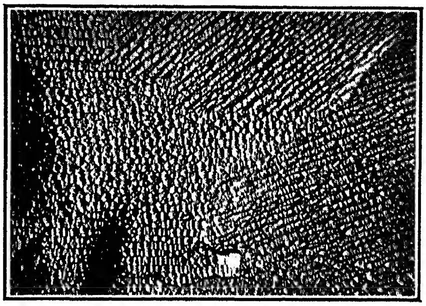


FIG. 2.—Surface of ingot of silver, 30 d.

FIG. 3.—Surface of cast tin dendrite and ground mass, 30 d.

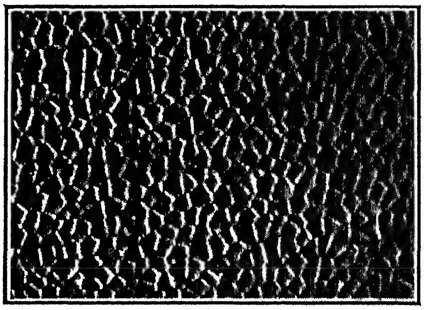


FIG. 4.—Surface of cast tin ground mass, 75 d.

Figs.[2] 1 to 5 show some characteristic structural forms of metals. Fig. 1 represents the surface of aluminum cast upon stone. There are seen dendrites

[1] Osmond, F., and Stead, J. E., "Microscopical Analysis of Metals"; Griffin & Co., London, Lippincott Co., Philadelphia, 1904.

Le Chatelier, H., "La technique de la métallographie microscopique" in "Contributions à l'étude des alliages," Commission des Alliages, Paris, 1896–1890, p. 421 (translation: *Metallographist*, 1901, IV, 1); continuation in *Rev. Mét.*, 1905, II, 528.

Stead, J. E., "Practical Metallography," Cleveland (England) Institute of Engineers, 1900, p. 97 (reprint, *Metallographist*, 1900, III, 220).

Campbell, W., "Notes on Metallography," *Sch. Mines Quart.*, 1904, XXV, 390.

Osmond, F., and Cartaud, G., "Les enseignements scientifiques du polissage," *Rev. Générale des Sciences*, 1905, XVI, 51.

Goerens, P., and Ibbotson, F., "Introduction to Metallography," Longmans, Green & Co., London, 1908.

Desch, C. H., "Metallography," Longmans, Green & Co., London, 1910.

[2] Figs. 1 to 4, Campbell, Appendix VI, *Report Alloys Research Committee*, 1901. Fig. 5, Sorby, *J. I. and St. I.*, 1887, I, 255.

composed of two axes at right angles; only one quadrant of a dendrite is well developed.

Fig. 2 gives the surface structure of an ingot of silver; three or more primary crystals or grains are built up of numerous secondary crystals possessing distinct orientation.

Figs. 3 and 4. In Fig. 3 is seen on the surface of cast tin a small dendrite, with two main axes at right angles, imbedded in a ground mass made up of small secondary crystals shown under larger magnification in Fig. 4.

Fig. 5 shows the polished and etched surface of a longitudinal section through a bar of wrought iron. It is made up of crystals of practically pure iron (ferrite) and a few small black streaks of slag.

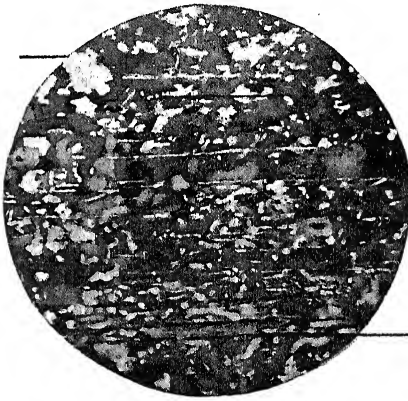


FIG. 5.—Structure of wrought iron (Sorby).

Beilby[1] has shown, that in polishing a metal, a surface layer or film is likely to be produced where structure and appearance differ from the undisturbed substance below, and that etching the surface removes the layer which had been spread over the surface by polishing, and reveals the true structure.

9. Hardness.[2]—The word *hardness* of a metal is used to express five ideas: the resistance it offers, to scratching (scratch hardness), to identation (identation hardness), to elastic impact (rebounding hardness), to cutting (cutting hardness), to permanent deformation (tensile hardness).

1. SCRATCH HARDNESS.—This is the hardness of the mineralogist. In the Mohs scale the following ten minerals are arranged in the order of hardness: Talc, 1; gypsum, 2; calcite, 3; fluorspar, 4; apatite, 5; feldspar,6; quartz, 7;

[1] *Proc. Roy. Soc.*, 1903, LXXII, 218.

[2] Calvert-Johnson, *Phil. Mag.*, and *J. Sc.*, 1859, XVII, 114; *Dingl. Pol. J.*, 1859, LII, 129; Commission des Methodes d'Essai des Matériaux de construction, Rothschild, Paris, 1895, III, 2, 61.

Martens and Henning, "Handbook of Testing Materials," Wiley, New York, 1899, p. 288.

Springer, *Cass. Mag.*, 1908, XXXIV, 387.

Turner, *J. I. and St. I.*, 1909, I, 426; Devries, *Proc. Am. Soc. Test. Mat.*, 1911, XI, 709; *American Machinist*, 1911, XXXV, 711.

topaz, 8; corundum, 9; and diamond, 10. A mineral will scratch all those below it in the series, and will be scratched by those above.

As the range of hardness between the several numbers of the series is uneven, Behrens[1] used the following set of seventeen needle-points which he graded according to the Mohs scale: Lead, 1; tin, 1.7; iron-bearing tin, 2; hard lead, 1.5–2.2; zinc, 2.5; copper, 3; brass wire, 3.1; gun metal, 3.3; bronze with 12 per cent. Sn, 3.5; the same with 18 per cent. Sn, 3.7; iron wire 3.7 to 3.9; sewing needles 5 to 5.5; the same tempered to yellow, 4; to blue, 5; steel auger tempered to yellow, 6; chrome steel, 6.2 to 6.5; ferrochrome with 50 per cent. Cr, 7 to 7.3.

In 1886 Turner[2] employed a diamond point, set in a vertical pencil and carried by a balanced arm, for scratching a metal moved slowly underneath the point. The point was pressed downward by a sliding weight until a normal scratch (one that is just visible as a dark line on a bright surface) had been made. The load in grams formed the measure of hardness or hardness number. Martens[3] modified the mode of operating in his sclerometer, by making the load on the pencil constant (20 grams) and measuring the width of the scratch with a micrometer-microscope. The hardness number is 1 divided by width of scratch in millimeters. A few of his numbers with the figures of the Mohs scale in parentheses are subjoined: Pb 16.8 (1.5), Sn 23.4 to 28.2 (2 to 3), Cu 34.3 to 39.8 (3), Zn 42.6, Ni 55.7. Kurnakoff-Schemtschuschny[4] photograph the Martens scratch under a known magnification and measure it.

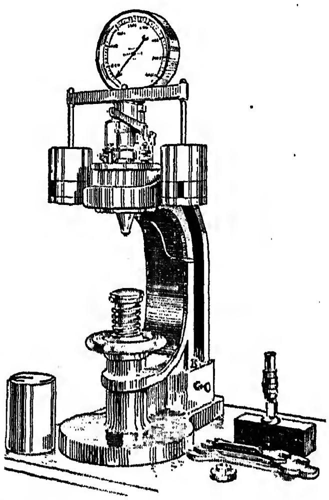


FIG. 6.—Brinell hardness testing machine.

2. INDENTATION HARDNESS.—The leading exponent of the method of producing a permanent indentation for measuring hardness is Brinell.[5] His method consists in forcing a hardened steel ball, usually 10 mm. in diameter, into the smooth surface of the metal to be tested and measuring the depth of the indentation. The hardness number is the quotient obtained by dividing the pressure in kilograms (3,000 kg. for iron and steel, 500 kg. for softer metals) by the spherical area of the cavity formed in square millimeters. Thus, with a 500-kg. load the hardness numbers of some non-

[1] Behrens, H., "Das Mikroscopische Gefüge der Metalle und Legirungen," Voss, Hamburg-Leipsic, 1898, p. 21.

[2] *Proc. Birmingham Philos. Soc.*, 1886, V, 291.

[3] Martens-Henning, *op. cit.*, p. 299; *Verh. Ver. Bef. Gewerbefleiss.*, 1888, LXVII, 40. Mitt. Kgl. techn. Versuchsanstalt, 1890, VIII, 236.

[4] *Zt. anorg. chem.*, 1908, LX, 1.

[5] Wahlberg, *J. I. and St. I.*, 1901, I, 243; Guillery, *Rev. Mét.*, 1904, I, 405; Revillon, *op. cit.*, 1908, V, 270; Grenet, *op. cit.*, 1908, V. 928; Schneider, *op. cit.*, *extr.*, 1912, IX, 569. Robin, *I. and St. I.*, Carnegie Mem., 1911, III, 236. Ammon, *Tr. A. I. M. E*, 1912, XLIV, 356.

ferrous metals are: Cu 74.0; Ag 59.0; Sb 55.0; Au 48.0; Zn 46.0; Al 38.0; Sn 14.5; with a 3,000-kg. load those for Swedish charcoal iron: gray 179; mottled 444; white 460.

Various apparatus have been constructed to carry out this test. The one shown in Fig. 6 is the latest form. It consists of a heavy cast-iron frame carrying an adjustable table for the support of the test-piece and a hydraulic press, the piston of which holds at the lower end the steel ball. The pressure is applied by means of a small hand-pump, and is read in kg. on the dial. In order to guard against error, a controlling apparatus is provided which limits the pressure to the amount for which the apparatus is set. It consists of a small pressure-cylinder communicating directly with the main cylinder, and contains a frictionless piston with cross-bar provided with hangers for weights. As soon as the pressure in the main cylinder has reached the amount desired for a test, the auxiliary piston with its controlling weights is raised; the pressure in the main cylinder remains constant as long as the weights of the control are held in suspension. In testing specimens of iron and steel, the pressure is maintained for 15 sec., in the case of softer metals 30 sec. is required.

Brinell[1] determined the relation that exists between his hardness number and the yield-point, ultimate strength and elongtaion of a metal. Karmarsch[2] pointed out years ago that, with the exception of Zn and Al, the harder a metal the greater its tensile strength. Kurnakoff-Schemtschuschny[3] found that the Brinell hardness number forms a true measure of the plasticity of metals.

A modification of the Brinell method is that of Ludwik.[4] The results obtained by Brinell are governed in part by the amount of pressure and the size of ball used. Ludwik does away with this inconvenience by employing a hard steel cone with a top angle of 90°, the impressions of which are uniform regardless of the pressures used. His hardness numbers, of course, differ from those of Brinell.

An indirect method has been devised by Ballantine.[5]

3. Rebounding Hardness.—This method, devised by Shore, is based on measuring the rebound of a hard body from the metal that is to be tested. The leading parts of the Shore scleroscope,[6] shown in Fig. 7, are a glass tube 11 in. long, 1/2 in. outside, and a little less than 1/4 in. inside diameter, graduated into 140 divisions; a cylindrical drop hammer pointed at the lower end; a rubber bulb, closing the top of the glass tube, connected with tubing to another near

[1] Wahlberg, *loc. cit.*

[2] *Dingl, Polyt. J.*, 1859, CLIII, 415.

[3] *Zt. anorg. Chem.*, 1909, LXIV, 149.

[4] Ludwik, P., "Die Kugelprobe," Springer, Berlin, 1908.

[5] *J. Frankl. Inst.*, 1908, CLXVI, 447.

[6] Shore, *Am. Machinist*, 1907, XXX, 747.

Springer, *Cass. Mag.*, 1908, XXXIV, 388; *IronAge*, 1908, LXXXII, 555.

Fréminville, *Rev. Mét.*, 1908, V, 329.

Shore, *Met. Ind.*, 1910, VIII, 332; *Proc. Am. Inst. Test. Mat.*, 1910, X, 490; *Iron Age*, 1910, LXXXVI, 490.

the base; a frame with plumb-rod and screws to level the base which carries the support of the pinion by means of which the tube is raised and lowered through a rack; a rod with sliding holder carrying a pointer and a magnifying glass, and two adjusting screws.

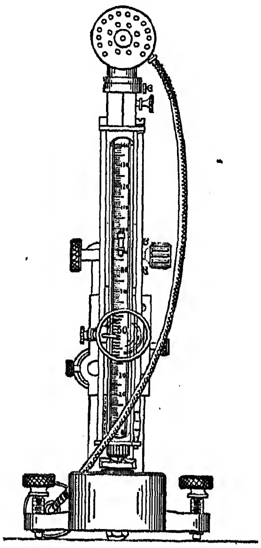


FIG. 7.—Shore scleroscope.

In making a test, the specimen is so placed that the smooth surface to be struck by the hammer is level, and the tube lowered; the upper rubber bulb is compressed and suddenly released when the suction draws up the drop-hammer, which is held at the top of the tube by a suitable device; a valve is opened to connect the tube with the air and the lower rubber bulb then squeezed to cause the hammer to fall. Upon striking the test-piece, the hammer rebounds; the approximate height of rebound is found by a preliminary trial, the pointer is adjusted, and the exact height read with the lens on the scale; this height forms the hardness number. With a large mass of hard metal the rebound is 95 per cent. of the fall. Table 2[1] shows how closely the results of the three methods discussed agree. The scleroscope gives a greater hardness to cold-worked metal than the other two methods. Turner and Shore agree on soft copper, as do Shore and Brinell on hard-drawn copper. The investigations of Maurer[2] upon the hardening and annealing of steels give some comparisons of methods for determining hardness.

TABLE 2.—COMPARISON OF HARDNESS NUMBERS[3]

Metal	Turner sclerometer	Shore scleroscope	Brinell ball test No. 6
Lead	1.0	1.0	1.0
Tin	2.5	3.0	2.5
Zinc	6.0	7.0	7.5
Copper, soft	8.0	8.0	....
Copper, hard	....	12.0	12.0
Softest iron	15.0	....	14.5
Mild steel	21.0	22.0	16–24
Soft cast iron	21–24	24.0	24.0
Rail steel	24.0	27.0	26–35
Hard cast iron	36.0	40.0	35.0
Hard white iron	72.2	70.0	75.0
Hardened steel	....	95.0	93.0

[1] Turner, *J. I. and St. I.*, 1909, I, 434; *Engineering*, 1911, XCII, 183, 246, 305

[2] *Metallurgie*, 1909, VI, 33.

[3] Other Comparative Data: Ballantine, *J. Frankl. Inst.*, 1912, CLXXIV, 583.

An instrument, similar to that of Shore, is the Rebondimeter of Tinberg,[1] another the Bernard Normameter.[2]

4. Cutting Hardness.[3]—In this method, a drill, with a given weight acting upon the point, is made to cut a hole to a definite depth; the number of revolutions required to do this gives the means of comparison. Keep's hardness testing machine[4] is used to determine the hardness (workability) of cast iron. A 3/8-in. straight-fluted drill of standard hardness bores upward into the test-piece with a speed of 200 r.p.m. The table to which the test-piece is clamped is weighted so that the combined weight of piece and table is 150 lb.; its downward motion is recorded autographically on a diagram in which dead-soft material gives a horizontal line, and material that the drill cannot penetrate a vertical line. A straight diagonal shows that the resistance of the metal to penetration is uniform, a deflection upward reveals a hard spot, one downward a spongy part.

A drill with diamond point intended for mineralogical purposes is the Jaggar microsclerometer.[5] It has been used by Boynton[6] to determine the hardnesses of the constituents of steel.

5. Tensile Hardness.—In general it holds true that the greater the tensile strength and the lower the elastic limit of a metal, the greater the hardness. This question has been recently investigated by Kürth,[7] Devries and Shore[8] and by Guillery.[9] In determining tensile hardness, it is essential that only thoroughly annealed metal be used, as mechanical treatment (hammering, rolling, drawing) greatly raises the yield-point while it does not affect the surface hardness; hard-drawn copper, *e.g.*, will show the same surface hardness as cast copper.

6. Calculation of Hardness.—This is based on the fact that soft metals have great atomic volumes (atom. wt.: spec. gr.) and hard ones small. Bottone[10] gives the following values, with diamond as 3010 at the top of the series: Mn 1456, Co 1450, Ni 1410, Fe 1375, Cu 1360, Pd 1200, Pt 1107, Zn 1077, Ag 990, In 984, Au 979, Al 821, Cd 760, Mg 726, Sn 651, Pb 570, Tl 565, Ca 405, Na 400, K 230. Benedicks[11] has elaborated this idea.

7. General.—The hardness of metal is greatly affected by the presence of very small amounts of other elements. Thus, Fe by C, P, Si, Mn, W, Cr, etc.; Cu by P, As, Sb, Sn, Mn, Ni; Ag by Au, Cu; Sn and Pb by As and Sb; and so on.

[1] Cook, *Foundry*, 1910, XXXVI, 121.

[2] *Rev. Mét.*, 1912, IX, 570.

[3] Kirsch, *Mitth.* k. k. *Technolog. Gewerbemuseum*, 1891, XI, 79; *Oest. Zt. Berg-Hüttenw.*, 1896, XLIV, 91.

[4] Keep, W. J., "Cast Iron," Wiley, New York 1902, p. 187.

[5] Jaggar *Am. J. Sc.*, 1897, IV, 399; Boynton, *J. I. and St. I.*, 1906, II, 291.

[6] *Op. cit.*, 1906, II, 287, 1908, II, 133.

[7] *Zt. Ver. deutsch. Ing.*, 1908, LII, 1560, 1608.

[8] *Proc. Am. Soc. Test. Mat.*, 1911, XI, 726, 733.

[9] *Proc. Int. Assoc. Test. Mat.*, New York, 1912, III[5].

[10] *Am. J. Sc.*, 1873, CL, 644; *Pogg. Ann.*, 1873, L, 640; *Chem. News*, 1873, XXVII, 215.

[11] *Zt. phys. Chem.*, 1901, XXXVI, 529.

The temperature of a metal greatly affects the hardness. Kürth[1] found that with the non-ferrous metals Cu, Ag, Ni, Al, Zn and Sn the hardness decreased as the temperature rose, and that with annealed steels containing C 0.20 to 0.75 per cent., the hardness diminished as the temperature rose from 20° to 150° C., increased slightly from 150° to 250° C., and then diminished rapidly from 250° to 500° C.

Hard metals are more or less sonorous. According to Robin,[2] this property is weaker with metals when pure than when alloyed with a small amount of another metal; it decreases with the rise of temperature and disappears at a red heat.

10. Strength of Metal.—The strength of a metal is shown by the resistance it opposes to forces which tend to change its shape. These may lead to rupture, crushing, bending, twisting and shearing of the metal and call forth corresponding tensile, crushing, bending, torsional and shearing resistances.

Small amounts of foreign elements present in a metal or purposely added to it, greatly modify the strength of a metal. Roberts-Austen[3] has tried to trace a connection between the atomic volumes (at. wt. : spec. gr.) of some elements and the effects they produce when added to metals with which they form solid solutions. The strength of a metal is further altered by mechanical treatment and annealing, by the temperature, and by the so-called fatigue. Hammering, rolling or drawing increases the strength; annealing again decreases it; raising the temperature diminishes it,[4] while lowering the temperature[5] has the opposite effect and increases the hardness.

FATIGUE[6] of a metal has been called the diminution of strength due to frequent applications and reversions of stresses. It has been believed that these alternations (vibratory stresses) caused rearrangement of molecules (crystallization) and thus a change in strength; it has also been held that such deterioration was due to microscopic flaws which in time became planes of rupture. Ewing and Humphrey[7] found by subjecting Swedish iron to alternating stresses inside

[1] *Zt. Ver. deutsch. Ing.*, 1909, LIII, 85, 209.

[2] *J. I. and St. I.*, Carnegie Schol. Mem., 1911, III, 125; *Rev. Mét.*, 1912, IX, 618.

[3] *Proc. Roy. Soc.*, 1888, XLIII, 425; *Phil. Trans. Roy. Soc.*, Ser. A, 1888, CLXXIX, 339; "Introduction to the Study of Metallurgy," 1910, p. 104.

[4] Baudrimont, *Ann. chim. phys.*, 1850, XXX, 304; *Pogg. Ann.*, 1851, XXII, 156; Le Chatelier, *Compt. Rend.*, 1889, CIX, 24; *Berg. Hütt. Z.*, 1890, XLIX, 73; Kürth, Zt. Ver. deutsch. Ing., 1909, LIII, 85, 209.

[5] Dewar, *Proc. Roy. Soc.*, 1893–95, XIV, 1; *Chem. News*, 1895, LXXI, 200; Dewar-Hadfield, *Proc., Roy. Soc.*, 1904, XLVII, 326; *Chem. News*, 1905, XCI, 13; Hadfield, *J. I. and St. I.*, 1905, I, 147; Baumann, R., "Die Festigheitseigenschaften der Metalle in der Wärme und Kälte," Kröner, Stuttgart, 1907.

[6] Stanton-Bairstow, *Proc. Inst. Civ. Eng.*, 1905–06, CLXVI, pt. IV, p. 78; Arnold, *Engineer*, 1905, C, 158; *Eng. Min. J.*, 1905, LXXX, 1105; Gardner, *J. I. and St. I.*, 1905, I, 481; Stanton, *op. cit.*, 1908, I, 54; Souther, *Proc. Am. Soc. Test. Mat.*, 1908, VIII, 379; Le Chatelier, *Rev. Mét.*, 1909, VI, 1156; Smith, *Iron, Coal Trades Rev.*, Sept. 30, 1910, p. 532; *Rev. Mét.*, 1911, VIII, 377; Smith, *Engineering*, 1909, LXXXVIII, 105; *Rev. Mét.*, 1910, VII, 217; Frémont, *Génie Civil*, 1910–11, LVIII, 53, 74. Wille, *Proc. Am. Soc. Test. Mat.*, 1904, IV, 321.

[7] *Phil. Trans. Roy. Soc.*, Ser. A, 1903, CC, 241; Metallographist, 1903, VI, 96.

the elastic limit and examining microscopically a polished part from time to time that there appeared on some of the crystals, probably more stressed than others, microscopic slip-lines similar to those observed with simple tensile stresses outside of the elastic limit. These lines showed a tendency to increase in number and to broaden, changed into slip-bands which grouping together covered the surface with dark markings. At this stage cracks appeared in some of the crystals, formed by the crystals yielding along the slip-bands, and joining formed a fissure which quickly caused the specimen to break. Thus a microscopic crack can be formed by alternating stresses in an otherwise perfectly sound metal, and cause final breaking.

TENSILE STRENGTH is the greatest stress a metal can bear without rupture. When a piece of metal is pulled in a testing machine, it is stretched uniformly. If the force be released, the metal will resume its original form on account of its *elasticity* up to the *elastic limit*. If the force be increased, the metal will undergo a permanent *elongation* and be finally ruptured with a decided *reduction of area* of cross-section. The ultimate tensile strength and the (commercial) elastic limit are measured in pounds per square inch, the elongation in percentage of the original length (=8 in. of the uniform cross-section to which the test-piece was reduced), and the reduction of cross-section by the percentage of difference between original and broken areas, and original area.

TABLE 3.—TENSILE STRENGTH OF SOME METALS AT ORDINARY TEMPERATURE

Metal	Tensile strength, lbs. per sq. in.	Metal	Tensile strength, lbs. per sq. in.
Co	75,000	Au, wire, hard drawn	37,000
Ni	54,000	Au, wire, soft drawn	24,000
Fe, rolled	55,000	Al, cast	12,590
Fe, cast	48,000	Al, rolled	19,290
Pd	50,000	Al, hammered	22,575
Pt, cast	45,000	Al, drawn	17,007
Pt, wire, unannealed	56,000	Te, cast	8,500
Pt, wire, annealed	32,000	Zn	5,000
Ag, cast	41,000	Sn, cast	4,600
Cu, cast	24,000	Sn, drawn	5,800
Cu, sheet	30,000	Bi, cast	3,000
Cu, bolts	34,000	Pb, cast	2,050
Cu, wire, hard drawn	60,000	Pb, pipe	1,650
Cu, wire, soft drawn	35,500	Pb, sheet	1,720
Au, cast	20,000	Sb, cast	1,000

CRUSHING STRENGTH.—There is no ultimate crushing strength with metals under compressive stress, as when the load on a metal is increased beyond a certain point it simply spreads. Inside this point, the (apparent) elastic limit,

a metal resumes its original form after removal of the load. With iron and steel the compression elastic limit is the same as with tension so that compression tests are rarely made.

For *Bending*, *Twisting* and *Shearing* strengths the treatise of Martens-Henning quoted, or other similar works, must be consulted.

11. **Brittleness[1] and Toughness.**—A metal is said to be brittle when a slight mechanical deformation or a sudden change of temperature produces rupture. Brittleness must not be confounded with hardness; sealing-wax, *e.g.*, is brittle, but not hard. All metals are brittle at temperatures approaching their melting points; if held at these temperatures for some time, they assume a coarse, fissured structure, and are said to be burnt. Good examples of this are furnished by the metals Zn, Cd, Sn and Cu, and the alloys brass and steel. The normal states of a metal can often be restored by slow heating to a suitable temperature followed by quick cooling. Heating or cooling a metal too quickly is likely to cause brittleness on account of the uneven expansion or contraction of the crystals. Brittleness is usually increased by the presence of a foreign substance, also by a low temperature (see § 10).

The reverse of brittleness is toughness. Besides the purity, the temperature of a metal has much influence upon its toughness. Martens[2] has arranged the following metals in the order of their toughness according to

$$\text{Toughness of test-length} = \frac{\text{Ultimate Strength}}{\text{Yield Point}} \times \frac{\text{Per cent. elongation in test-length}}{100}$$

as follows: Pb, Pt, Fe, Al, Ni, Zn, Sn, Cu, Au and Ag.

12. **Plasticity,** or the property of permitting large permanent deformation without rupture, is characteristic for metals which are soft, inelastic and tough. The more prominent these properties in a metal, the easier will it be to roll, forge, draw or spin it. Ewing and Rosenhain[3] have shown that the plastic movement in a metal is due to slips in a crystal in certain directions along so-called cleavage or gliding planes. When the plastic deformation of a metal begins, the faces of a crystal grain show under the microscope fine lines which grow in number as the strain increases. At first they are approximately at right angles to the direction of the pull; as the strain increases lines of a different direction appear on other grains, and later several systems of intersecting lines on a single grain may be seen. These slip-lines or slip-bands are not cracks, but steps in the surface, as the strained metal recovers its original elasticity after it has been allowed to rest or has been annealed. The same authors have also found that the crystalline structure of metals is not destroyed by strain, but that certain metals, especially Cu, Au, Ag, Pb, Cd, Sn, Zn and Ni are often likely to form twin crystals; twinning does not occur in Fe. Plastic yielding may there-

[1] Charpy, "Rapport sur les essais des métaux par choc," at Copenhagen, *Rev. Mét.*, 1909, VI, 1229.

[2] *Op. cit.*, p. 307.

[3] *Philos. Trans. Roy. Soc.*, Ser. A, 1899, CXCIII, 353; 1901, CXCV, 279; *Metallographist*, 1900, III, 94; 1902, V, 81; *Proc. Roy. Soc.*, 1905, LXXIV, 557; *J. I. and St. I.*, 1906, II, 189.

fore be due in part also to the formation of twin crystals. Humphrey[1] has carried further the work of the above investigators.

According to the formula: Plasticity $= \frac{\text{Toughness}}{\text{Yield Point}} \times 1000$, Martens[2] has placed the following metals in the given order: Fe, Pt, Ni, Al, Zn, Cu, Ag, Au, Pb and Sn. Coarsely crystalline metals, such as As, Sb and Bi, being rather inelastic and brittle, are not easily deformed without rupture. Kurnakoff-Schemtschuschny[3] give the following order: K, Na, Pb, Tl, Sn, Bi, Cd, Zn, Sb, which supplements the one of Martens; they found that the plasticity of a metal is proportional to its Brinell hardness number.

As the crystallinity of most metals becomes much developed when they are subjected to mechanical treatment, the plasticity becomes impaired; it is restored by annealing and then allowing them to cool slowly. These temperatures vary with the metals. Some metals can be permanently deformed at all temperatures; others only at a red heat, they crack at ordinary temperatures (are *cold-short*); again metals can be readily rolled, hammered, drawn at ordinary temperature, but not at a red heat (are *red-short*); finally, some metals can be worked only within a small range of temperature; thus, Zn is brittle up to and at 100° C., and above 200° C., but readily malleable at 150° C. Plasticity is very much influenced by the casting temperature, the preceding mechanical and thermal treatment and the purity of a metal, and further by the rate at which the deforming force is applied.

The three means usually employed for permanently deforming a metal are by rolling, forging and drawing; others are spinning, pressing and punching.

MALLEABILITY is the property of metals of permanently extending in all directions without cracking when rolled and forged. The following order of malleability: Au, Ag, Al, Cu, Sn, Pt, Pb, Zn, Fe, Ni gives an approximate idea of the behavior of metals in rolling.

DUCTILITY is the property of metals of permanently extending by traction, or the property that enables them to be drawn into wire. Ductility is closely related to malleability, but not identical with it as seen by the following order: Au, Ag, Pt, Fe, Ni, Cu, Al, Zn, Sn, Sb.

FLOW OF METALS.[4]—The permanent deformation of a metal caused by the application of pressure is due to the movement of the molecules after the yield-

[1] *Philos. Trans. Roy. Soc.*, Ser. A, 1903, CC, 225; *Proc. Roy. Soc.*, 1902, LXX, 462.

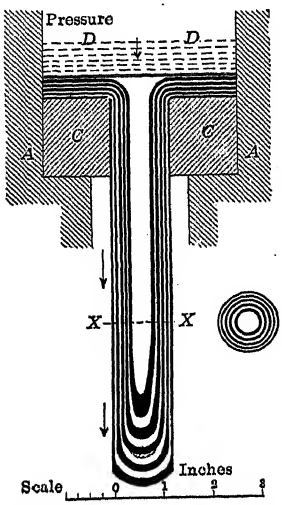
See also Osmond-Frémont-Cartaud, *Rev. Mét.*, 1904, I, 11; Rogers, *op. cit.*, 1906, III, 518; Gulliver, *Proc. Inst. Mech. Eng.*, 1905, 141; Mallock, *Proc. Roy. Soc.*, 1909, LXXXII, A, 26; Campbell, *Proc. Ind. Mech. Eng.*, 1901, 859; Frémont, *Rev. Mét.*, 1905, II, 801; Osmond-Cartaud, *op. cit.*, 1906, III, 653; *J. I. and St. I.*, 1906, III, 144; Tammann, *Zt. Electrochem.*, 1912, XVIII, 584.

[2] *Op. cit.*

[3] *Zt. anorg. Chem.*, 1909, LXIV, 149.

[4] Tresca, "Sur l'écoulement," Académie des Sciences, Mémoirs des Savants Étrangers, 1868, XVIII, 733; 1872, XX, 75, 281; *Compt. Rendus* 1864, XLIX, 754; 1865, LX, 398; 1867, LXIV, 809; 1868, LXVI, 263; *Rev. Un. des Mines*, 1866, XX, 188; *J. Frankl. Inst.*, 1866, LI, 271. Laminage et Forgeage: *Mémoirs Savants, Étr.*, 1872, XX, 137; *Compt. Rend.*, 1883, XCVI, 1821, XCVII, 222, 510, 928. Poinconnage: *Mém. Savants, Étr.*, 1872, XX, 617; *Compt. Rend.*,

point has been reached. It has always been held that metals flowed like viscous fluids, but Ewing and Rosenhain[1] have shown that the flow is not homogeneous throughout the mass, but is due to slips occurring in a limited number of places.



FIGS. 8 AND 9.—Flow of solid metal under pressure.

The form and rapidity of flow depend upon the structure and temperature of the metal, the nature of the pressure, and the duration of pressure. Tresca, in 1865, was the first to study the flow of metals under pressure. In one of his experiments, Figs. 8 and 9, he used a cast-iron cylinder *A*, 3.94 in. in diam., with shoulder carrying a die *C* having an opening 1.18 in. in diam. In the cylinder were placed 10 discs of sheet lead, 0.12 in. thick, as indicated by the dotted lines *D*., and then hydraulic pressure was applied to a piston closely fitting the cylinder. The lead flowed through the orifice in the form of a jet, 1.18 in. in diam. and 7.87 in. long, rounded at the bottom and consisting of as many concentric tubes as there were sheets of lead. The flow of metal in punching, as shown by Frémont,[2] is represented in Fig. 10. The block to be punched, 4.72 in. sq. and 2.52 in. thick, was made up of sixteen separate sheets of lead, 0.16 in. thick; the punch was 0.79 in. in diam. The disc cut out was found to be only 1.22 in. thick instead of the original 2.52 in. This proved that the metal had

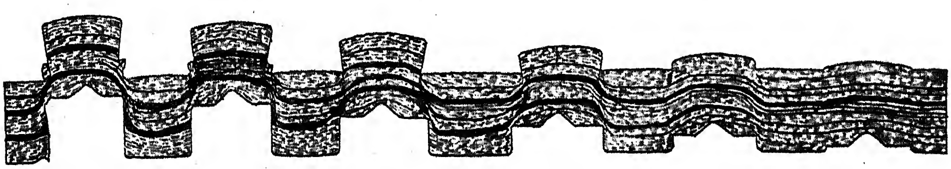


FIG. 10.—Flow of solid metal in punching.

1869, LXVIII, 1197; 1870, LXX, 27, 288, 368. Rabotage: *Compt. Rend.*, 1871, LXXIII, 1307; 1873, XX, 584, 655; 1874, I, 503; *Proc. Engl. Inst. Mech. Eng.*, 1867, CXIV, 114.

Martens-Henning, "Handbook of Testing Materials," Wiley, New York, 1899, p. 86.

Hartman, L., "Distribution des déformations des métaux soumis à des efforts," Paris, 1896.

Fredmond, *Bull. Soc. d'Enc.*, 1896, 1218.

Tissot, *Bull. Soc. Ind. Min.*, 1900, XIV, 3, 1660.

Osmond-Frémont-Cartaud, *Rev. Mét.*, 1904, I, 11; 1905, II, 311.

Obermayer-Tammann-Werigin-Lewkojew, *Beibl. Ann. Phys.*, 1905, XXXIX, 6.

Hort, *Berichte der deutsch. phys. Gesell.*, 1907, V, 54.

Frémont, *Rev. Mét.*, 1908, V, 678, *et seq.*

Gulliver, *Proc. Roy. Soc. Edinburgh*, 1909, XXX, 38; *Rev. Mét.*, *Extr.* 1910, VII, 443.

Jänecke, *Metallurgie*, 1911, VIII, 68; Extrusion (squirting) of Metal, *Iron Age*, 1910. LXXXV, 1507; *Eng. Rev.*, 1910, XXXIII, 310.

[1] *Loc. cit.*

[2] *Rev. Mét.*, 1908, V, 684; see also Townsend, *J. Frankl. Inst.*, 1878, CV, 145; Kreuzpointer, *Iron Age*, 1890, XLV, 948.

flowed away from underneath the punch until it had grown so thin that the resistance to shearing had become less than the pressure on the punch.

In making tensile tests[1] with a plastic metal, the metal stretches uniformly until the yield-point is reached and then begins to flow from the ends toward the center. When the flows have met, the metal having become perfectly plastic will be stretched again uniformly throughout its length. Now a local contraction (necking down) begins, because the metal yields more rapidly at one cross-section than at another, and elongation occurs only in the immediate vicinity of the contraction; the cross-section becomes smaller until finally rupture occurs with a reduction of area of cross-section of 50 or more per cent. In testing an iron rod as it comes from the rolls, the first superficial change is a peeling-off of the mill-scale; if the rod has been polished, the surface will become dull after the yield-point has been passed, then crinkled, more or less scarred, grooves will form with soft metals; and finally the break will show cup-shapes ranging from a perfect cone and cup, through a truncated cone to an oblique fracture. In rolling or hammering, metals assume their shapes by virtue of the flow of molecules.

The stretching and spreading effect of rolls is discussed in §289.

13. Welding Power[2] is the property which some metals possess of forming a solid union when extraneous pressure is applied. Pt, Fe and Ni show this property to a marked degree. If two metals are to be welded, the parts must be brought into such intimate contact that they may be joined by the cohesion of the molecules. Ordinarily this contact is effected by bringing the metals to the temperatures at which they become pasty, when the motion of the molecules is so accelerated that they interpenetrate or diffuse into one another. In heating readily oxidizable metals, it is essential to exclude air in order to retain clean surfaces, or to use fluxes which will slag any oxides produced by heating and form an impervious coating. Ewing and Rosenhain[3] have welded at ordinary temperature two cast discs of lead, 1 1/2 in. diam., suitably cleaned, by subjecting them to a pressure of 5 tons, thus showing that pressure to a certain extent can replace heat. Spring[4] welded filings or powders of a metal to a solid block by placing them in a steel cylinder and subjecting them, by means of a steel piston, to great pressures. He proved experimentally[5] that the heat evolved in his extremely slow compression was insufficient to fuse any metal In this way filings of lead under a pressure of 13 tons per sq. in. were compressed to a block in which no filings could be detected with a microscope. The lead thus obtained had a specific gravity of 11.5, while that of the lead after simple fusion was 11.3. Sn required 19 tons; Cu, 33; Zn, Sb, Al, Bi, each 38 tons. If

[1] Kirsch, Mittheilungen der kgl. Versuchsanstalten zu Berlin, 1887, V, 69; 1888, VI, 37; 1889, VII, 9.

Kirkaldy, "Strength and Properties of Materials," London-New York, 1891, Plates 24 and 25.

[2] Law-Merrett-Digby, *J. I. and St. I.*, 1911, I, 103.

[3] *Phil., Trans. Roy. Soc.*, Ser. A, 1901, CXCV, 279; *Metallographist*, 1902, V, 106.

[4] *Ann. Chim. Phys.*, 1881, XXII, 170; *Bull. Soc. Chim.*, 1883, XL, 520.

[5] *Bull. Soc. chim.*, 1884, XLI, 488.

the pressure is increased above that required to form a block, it will be converted into heat; thus, if with lead the pressure is increased to 33 tons, the lead begins to flow, while Sn requires 47 tons pressure to make it flow.

14. Diffusion of Metals.[1]—When samples are taken from a bath of silver-bearing copper in a reverberatory furnace and assayed, the results will show that the silver is uniformly distributed through the copper. This uniform spreading of one metal in another is due to diffusion or molecular mixing. The experiments of Roberts-Austen have shown that solid metals diffuse in liquid metals as do salts in water, and that even solid metals diffuse in one another at ordinary temperature. As the rate of diffusion is greatly increased by a rise of temperature, the diffusion of molten metals is almost instantaneous, that of solid and molten metal a great deal slower, and that of two solid metals extremely slow. Thus Roberts-Austen has shown that "the amount of gold which would diffuse in solid lead at ordinary temperature in 1,000 years is almost the same as that which would diffuse in molten lead in one day". Le Gris[2] gives a number of excellent photomicrographs which clearly show the progress of diffusion between superimposed layers of liquefied metals.

The diffusion of C in Fe is the basis of two important metallurgical operations —the carburization of iron in the cementation process, and the decarburization in the manufacture of malleable castings. Diffusion of O in Cu[3] causes copper tubes to become brittle when exposed to oxidizing influences. The peculiar phenomenon in the operation of kernel-roasting of Cu that travels toward the center and of Ag in the opposite direction has been attributed by Roberts-Austen in part at least to diffusion.

15. Occlusion and Diffusion of Gases.—Some metals have the power to dissolve gases; others are sufficiently permeable for the penetration of gases; again gases form chemical compounds with metals, such as nitrides (§ 63), hydrides (§ 64), etc. The recent researches of Sievert and associates[4] cover the ground almost wholly. They found that N and H are insoluble in Cd, Tl, Zn, Pb, Bi, Sn, Sb, Al, Ag and Au; that N is insoluble in Cu, Ni, Pd, and that CO and CO_2 are insoluble in Cu. The statement of the insolubility of CO in Cu will be doubted by copper refiners. As to solubility the results show that, under certain conditions, H is soluble in Pd[5], Fe, Ni, Co, Cu (older statements give also Au, Tl, Al), O in Ag,[6] N in γ-iron, SO_2 in Cu.

Some of their results are represented graphically in Fig. 11. The straight-

[1] Roberts-Austen, *Philos. Trans. Roy. Soc.*, Ser. A, 1896, CLXXXVII, 383; *Proc. Roy. Soc.*, 1900, LXVII, 101.

Bruni-Meneghini, *Internat. J. Metallography*, 1911, II, 26.

[2] *Rev. Mét.*, 1911, VIII, 613.

[3] Heyn, *Mitth. kgl. Versuchsanst.*, Berlin, 1900, XVIII, 315.

[4] *Zt. phys. Chem.*, 1907, LX, 129; 1909, LXVIII, 115; *Ber. deutsch. chem. Ges.*, 1909, XLII, 338; 1910, XLIII, 893; *Zt. anorg. Chem.*, 1909, XLIV, 29; *Zt. Electrochem.*, 1910, XVI, 707; *Stahl u. Eisen*, 1909, XXIX, 1248; 1910, XXX, 1531; *Rev. Mét. Ext.*, 1908, V, 10; 1909, VI, 664; 1910, VII, 743; *Ber. deutsch. chem. Ges.*, 1912, XLV, 221.

E. Bergner, Dissertation, Leipsic, 1912.

[5] Valentiner, *Ber deutsch. chem. Ges.* 1911, XIII, 1003; *J. Inst. Met.*, 1912, VII. 274.

[6] Donnan-Shaw, *J. Soc. Chem. Ind.*, 1910, XXIX, 989.

line curves show that the solubility of H_2 and SO_2 in the solid metals is proportional to the temperature; that at each melting-point there is a break when the solubility increases with rise of temperature more rapidly than in the solid metal. This is especially noticeable with Cu, which in the solid state dissolves

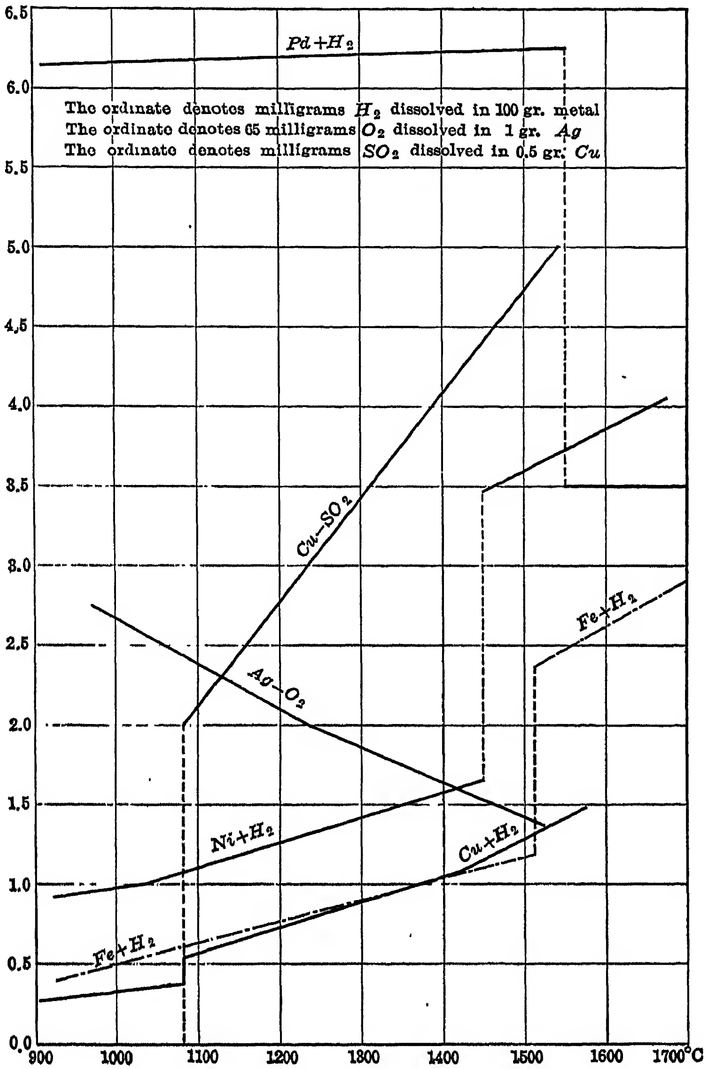


FIG. 11.—Solubility of gases in metals.

hardly any SO_2, at least at atmospheric pressure.[1] The solubility of O in Ag at ordinary temperature is zero, it reaches its maximum at the melting-point, and decreases gradually with the rise of temperature of the liquid metal. Precipitated Pt and Pt-metals readily absorb O_2, H_2, CO_2 and H_2O which when driven

[1] Schenck-Hempelmann, *Metall-Erz*, 1913, x, 283.

off by heat are not reabsorbed. The absorbing power of Pd for H forms an exception to the general absorption law expressed by $K = \frac{\sqrt{p}}{m}$, in that the solubility for H_2 decreases rapidly from 100° to 250° C., more slowly from 250° to 650° and changes very little to 1540° (see Fig. 11), the melting-point of Pd.

As to the diffusion of gases, Sievert and associates show that H_2 readily diffuses through Fe, Pd, Pt, Cu, Ni, as does O through Ag[1] above 770° C.; that H_2 does not penetrate Al and Ag; that CO does not diffuse through Cu, Fe, Ni, nor SO_2 through Cu. In general, the diffusing power increases with the temperature; it usually is paralleled by the dissolving power, the leading exception being Pt through which H_2 diffuses at a low temperature, but by which it is not dissolved.

The absorbing powers of industrial forms of iron, for gases, have been studied by Boudouard[2] and Goerens.[3] The mechanical properties of some metals are strongly affected by the presence of gases as, *e.g.*, that of steel by H.[4]

16. Fusibility.—At ordinary temperature all metals are solid excepting Hg, but most metals can be fused if exposed to a heat of sufficient intensity. A few metals which remain solid when heated with the O–H blowpipe (2,000° C.) have been liquefied at the temperature of the electric arc (3,500° C.).

Table 4 by Burgess[5] contains the latest data of the melting-points of metals.

TABLE 4.—MELTING-POINTS OF METALS

Metal	Deg. C.	Metal	Deg. C.	Metal	Deg. C.
Hg	−38.7	Mg	651	Pd	1,549
K	62.3	Al	658.7	Th	>1,700<Pt
Na	97.5	Ca	810	Va	1,730?
S	113.5–119.2	Sr	>Ca, <Ba?	Pt	1,755
Se	217–220	Ba	850	Ti	1,900?
Sn	231.9	Ag	960.5	Rh	1,940
Bi	271	Au	1,063	Ru	>1,950
Tl	302	Cu	1,083	Ir	2,300?
Cd	320.9	Mn	1,225	Ur	near Mo
Pb	327.4	Si	1,420	Mo	2,500?
Zn	419.4	Ni	1,452	Os	2,700?
Te	452	Co	1,490	Ta	2,850
As	500?, 850	Cr	1,510	W	3,000
Sb	630	Fe	1,520,	C	>3,600
Ce	640				

[1] Troost, *Compt. Rend.*, 1884, XCVIII, 1427.
[2] *Rev. Mét.*, 1908, V, 69; *Metallurgie*, 1908, V, 277.
[3] *Metallurgie*, 1910, VII, 305 (with bibliography).
[4] Heyn, *Stahl. u. Eisen*, 1900, XX, 837; Wedding-Fischer, *op. cit.*, 1903, XXXIII, 1268. *I. and S. Inst.*, Carnegie Mem. 1911, III, 236, (Andrew), 249, (Baker).
[5] J. Wash. Acad. Sc. 1911, I, 16; *Circular* No. 35, Bureau of Standards, Washington, D. C., June 15, 1912.

Arsenic sublimes at 450° C. without fusing; it becomes liquid under the pressure of its own vapor at 850° C.

If the junction of a thermo-electric pyrometer[1] is placed in a fused metal, and this is allowed to cool, and if, further, simultaneous readings of temperature and time are taken and plotted by coordinates, a *cooling* or *time-temperature curve* like *A–B*, Fig. 12, will result. As the metal cools, the temperature sinks (branch *A–a*) until solidification begins when the temperature remains constant (branch *a–b*), on account of the evolution of heat caused by crystallization. Only when the metal has become completely solid will the curve resume its downward course (branch *b–B*).

In heating a metal above its melting-point the temperature will rise from *B* to *b*, when, on account of the heat absorbed by the fusion of the metal, the temperature will remain constant (branch *b–a*) until the metal is completely fused, then the temperature will rise again (branch *a–A*). The slope of branch *A–a* gradually decreases because the difference in temperature between metal and

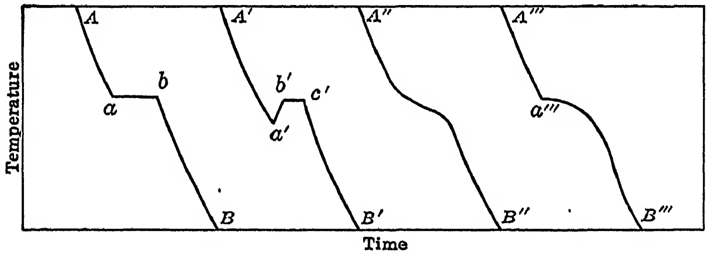


Fig. 12.—Cooling or time-temperature curves of pure metal.

atmosphere grows smaller and the heat lost diminishes. The slope of branch *b–B* is steeper than that of *A–a*, as the specific heat of the liquid metal is greater than that of the solid. The length of branch *a–b* depends upon the weight of the metal, its specific heat, latent heat of solidification and the rate of cooling. The purer the metal, the more sharply defined will be the branch *a–b*.

The change of a solid cooling metal into an allotropic form causes a similar retardation.

In cooling a liquid metal slowly, it may happen that the temperature sinks below the true freezing-point, curve *A'B'*, Fig. 12, to *a'* without the metal solidifying. This is called *undercooling* or *surfusion*.[2] When the undercooled metal begins to solidify, the liberation of the latent heat of fusion reheats the metal to the true freezing-point; and now the cooling proceeds through *b'c'B'* as in curve *A–B*. Undercooling is more common with metals than with alloys. Of all the metals, Sn[3] has shown the greatest undercooling, viz., 20° C., while

[1] Burgess, *Electrochem. Met. Ind.*, 1908, VI, 366, 403; *Bull.* 99, U.S. Bureau of Standards, Washington, 1908, V, No. 2.

[2] Riemsdijk, *Ann. chim. phys.*, 1880, XX, 66; *Chem. News*, 1880, XLI, 126, 266. Roberts-Austen, *Proc. Roy. Soc.*, 1898, LXIII, 447.

[3] *Reports, Alloys Research Committee*, I, 1891, 543; II, 1893, 102; III, 1895, 238; IV, 1897, 31; V, 1899, 35; VI, 1904, 7, 859, 1319; VII, 1905, 857; VIII, 1907, 57.

with Au it reaches only 2.5° and with Cu about 7° C. Undercooling may be minimized by stirring the metal while it is cooling, or by inoculating it; that is, dropping into the vessel a small solid particle of the cooling metal. It is not known that a metal can be heated above its melting-point without becoming liquid.

If a metal contains a small amount of another element which remains free, *i.e.*, unalloyed, the freezing-point becomes indistinct and may even be obliterated, curve *A'' B''*, Fig. 12. If the other element is alloyed with the metal, forming a solid solution (§29), the direction of the curve is suddenly changed from the normal, as shown at *a'''* in curve *A''' B'''*, Fig. 12.

In Table 5 are given the *latent heats* of *fusion* of some metals; they represent the number of degrees centigrade through which 1 kg. H_2O would be raised by 1 kg. of each of the several metals in passing from the liquid to the solid state.

Richards[1] found that, with the exception of Al, Sn and Bi, the latent heat of fusion of 1 kg. of metal is about one-third the quantity of heat required to raise it from absolute zero (−273° C.) to its melting-point, or equal 2.1 times its melting-point (degrees centigrade absolute temperature) divided by its atomic weight. The high latent heat of fusion of Al may explain the fact that the metal passes through a mushy stage before it becomes thoroughly fluid.

TABLE 5.—LATENT HEATS OF FUSION OF SOME METALS

Al.......	100.0	(H_2O....	80.03)	Fe.......	69.0	Co......	68.0	Ni........	68.0
Cu.......	43.0	Pd......	36.3	Pt.......	27.2	Ag......	24.35	Zn........	22.6
Te.......	19.0	Au......	16.3	Sb.......	40.2	Sn......	13.82	Cd........	13.1
Bi.......	12.0	Pb......	4.0	Hg.......	2.8				

17. Volatility.—Most metals require very high temperatures to be converted into vapor by means of heat. Table 6 gives the boiling-points of metals published by Watts[2] and of Greenwood.[3] The latter have received the preference, as they represent actual determinations while the figures of Watts are approximate numbers for the metals arranged in their order of volatility by Moissan[4] in his experiments in an electric furnace.

Two metals, As and Os, are changed into vapor without melting.

Krafft and Bergfeld[5] give the data regarding the volatilization of metals *in vacuo* shown in Table 8.

[1] *J. Frankl. Inst.*, 1893, CXXXVI, 118; 1897, CXLIII, 382.

[2] *Tr. Am. Electrochem. Soc.*, 1907, XII, 141; *Electrochem. Met. Ind.*, 1907, V, 450.

[3] *Proc. Roy. Soc.*, 1909, A, LXXXII, 396; *Tr. Faraday Soc.*, 1912, VII, 145; *Electrochem. Met. Ind.*, 1909, VII, 408; *Eng. Min. J.*, 1911, XCII, 3; *Zt. Electrochem.*, 1912, XVIII, 319; *Rev. Mét. Extr.*, 1912, IX, 520.

[4] Moissan, H.-Mouilpied, A. T. de, "The Electric Furnace," Arnold, London, 1904, p. 29, or tr. by Lehner, V., Chemical Publishing Co., Easton, Pa., 1905, p. 30; *Compt. Rend.*, 1906, CXLII, 425; *Eng. Min. J.*, 1906, LXXXI, 842; *VI Internat. Congress Appl. Chem.*, 1906, I, 61.

[5] *Ber. deutsch. chem, Ges.*, 1905, XXXVIII, 254; *Eng. Min. J.*, 1905, LXXIX, 767.

TABLE 6.—BOILING-POINTS OF SOME METALS AT ATMOSPHERIC PRESSURE

Metal	Deg. C.	Metal	Deg. C.	Metal	Deg. C.
Zn	940 W	Au	2,200	Rh	2,750 W
Cd	1,025 W	Cr	2,200 G	Ru	2,780 W
Mg	1,120 G	Sn	2,270 G	Pd	2,820 W
Bi	1,420 G	Cu	2,310 G	Ir	2,850 W
Sb	1,440 G	Ni	2,450 W	Os	2,950 W
Pb	1,525 G	Fe	2,450 G	Ur	3,100 W
Al	1,800 G	Pt	2,650 W	Mo	3,350 W
Ag	1,955 G	Ti	2,700 W	W	3,700 W

TABLE 7.—BOILING-POINTS OF SOME METALS *in Vacuo*

Metal	Beginning of evaporation *in vacuo*	Boiling-point *in vacuo*	Boiling-point at 760 mm.
	Deg. C.	Deg. C.	Deg. C.
Hg	−40	155	357
Cd	156	450	749
Zn	184	550	920
K	63	365	667
Na	98	418	742
Bi	270	993	1,700
Ag	680	1,360	2,070

The data agree with those of Demarcay.[1] Boudouard[2] found that Fe began to be volatilized *in vacuo* at 900° C. and was decidedly volatile at 1,100° C. Hughes[3] found that Cu was rendered volatile *in vacuo* at 700° C. Other data are those by Kraft.[4]

In 1912 Turner[5] separated Zn from brass by heating *in vacuo* to 1,200 ° C.; in an impure sample of brass, he removed with the Zn, all the Pb and As present. He also succeeded in recovering Zn from hard head (Zn charged with Fe).

In the smelting of Pb, Ag, and Au ores, some metal is lost by volatilization, although the boiling-points of those metals at atmospheric pressure are never reached. Thus men working over a kettle filled with molten lead below a red heat are likely to feel the poisonous effect of lead fumes; in cupelling Pb at a red

[1] *Compt. Rend*, 1882, XCV, 183.
[2] *Rev. Mét.*, 1908, V, 74.
[3] *J. Inst. Met.*, 1912, VII, 116.
[4] *Ber. deutsch. chem. Ges.*, 1903, XXXVI, 1690; 1905, XXXVIII, 262; Kraft-Kuch-Haagn, *op.cit.* 1913, XXVI, 1690; Kraft-Lehmann, *op. cit.*, 1905, XXXVIII, 242; Kraft-Merz, *op. cit.*, 1903, XXXVI, 4344; Groves-Turner, J. Soc. Chem. Ind., 1912, XCI, 585.
[5] *J. Inst. Met.*, 1912, VII, 105.

heat, the fume is readily seen. Richards[1] shows that Ag and Au are volatilized when melted in a reverberatory furnace with a current of air passing over them Kohlschütter-Ehlers,[2] studying the condensation of vapors of As, Fe, Cd and Zn *in vacuo*, in the presence and absence of indifferent gases, found that the condensed metal was the more compact the smaller the amount of indifferent gas that was present, and the lighter the gas.

18. **Expansion and Contraction.**—All metals expand when they are heated and shrink when they are cooled. The expansion of metals for the same rise of temperature is smaller when they are solid than when they have been liquefied. The coefficient of linear expansion of a metal, *i.e.*, the elongation of its unit of length when the temperature rises from zero to 1° C., is one-third that of the cubical expansion. Table 8 gives the coefficients of linear expansion of the leading metals for 1° C. between 0 and 100° C.

TABLE 8.—COEFFICIENTS OF LINEAR EXPANSION FOR 1° C. BETWEEN 0 AND 100° C.

Metal	Coefficient	Metal	Coefficient
Ag	0.000019	Na	0.000072
Al	0.000023	Ni	0.000013
As	0.000006	Os	0.000007
Au	0.000014	Pb	0.000029
Bi	0.000013	Pd	0.000012
Cd	0.000030	Pt	0.000009
Co	0.000012	Rh	0.000009
Cu	0.000017	Sb	0.000017
Fe	0.000012	Sn	0.000023
Hg	0.000181	Te	0.000017
Ir	0.000007	Tl	0.000031
K	0.000083	Zn	0.000029
Mg	0.000027		

With the exception of Sb, Bi and Sn, the order of expansion is similar to the order of fusibility, or metals with a low melting-point have a high coefficient of expansion and *vice versa*. The coefficient of expansion increases with the temperature—near the melting-point of a metal the increase is very decided; it varies also with the physical condition of the metal (cast, rolled, hammered, hardened, annealed)—thus an operation which increases the density will also increase the rate of expansion; with metals crystallizing in the isometric system, the expansion will be the same for the three dimensions, unless it be hampered by the metal being strained in some direction; with metals crystallizing in other systems, the expansion will be unequal.

SHRINKAGE is the reverse of expansion, hence the coefficient of expansion is a true measure for shrinkage. But the coefficient increases with the tempera-

[1] "Metallurgical Calculations," McGraw Publishing Co., New York, 1908, III, 588.
[2] *Zt. Electrochem.*, 1912, XVIII, 3731.

ture and is suddenly changed when a metal changes from the solid to the liquid state. Wüst[1] has investigated the shrinkages of a number of metals and alloys. The results with metals are given in Tables 9 and 10. The figures show the shrinkage during freezing and the total shrinkage, both expressed in per cent.

TABLE 9.—ANALYSES OF METALS

Metal	Pb	Sn	Zn	Fe	Al	Cu	Bi	Sb
Pb	98.2	1.27						
Zn			97.3	2 67				
Sn (Banca)		100						
Sn		99 8						
Al				0.33	99 16			
Cu				0 35		99.16		
Bi	0.12						99.80	
Sb		0.34		1.30		0.56		97.09

TABLE 10.—SHRINKAGE OF METALS

Metal	Casting temperature, deg. C.	Freezing-point, deg. C.	Shrinkage during freezing, per cent.	Total shrinkage, per cent.
Pb	500	326	0.065	0.82
Pb	600	326	0.085	0.83
Zn	650	416	0.08	1.40
Zn	700	416	0.08	1.40
Zn	750	416	0 08	1.40
Sn (Banca)	550	225	0.1 to 0 15	0.44
Sn	500	225	0.1 to 0.15	0.55
Al	800	683		1.78
Al	850	683		1.78
Cu	1,250	1,060	Expansion	1.42
Bi	500	261		0.29
Sb	710	621		0.29
Sb	750	621		0.63
Sb	800	621		0.29
Sb	1,050	621		0.66

The expansion of copper is to be attributed to the setting-free of dissolved gas.

19. Specific Heat.[2]—The specific heat of a metal, or the ratio between the amount of heat necessary to increase its temperature 1° C. and the amount neces-

[1] *Metallurgie*, 1909, VI, 769, includes bibliography; abstract in *Iron Age*, 1910, LXXXV, 790.

[2] Richards, *J. Frankl. Inst.*, 1893, CXXXVI, 37, 116, 178.

sary to increase the temperature of an equal weight of water 1° C., increases with the temperature, being especially high near the melting-point, decreases as the density increases, and varies with the purity. Table 11 shows that the specific heats of metals are low; they are in fact lower than those of other solids; it gives the specific heats of some metals at about 15° C. and in the neighborhood of their melting-points.

TABLE 11.—SPECIFIC HEATS OF SOME METALS

Metal	Specific heat: At about 15° C.	Specific heat: At about melting-point	Metal	Specific heat: At about 15° C.	Specific heat: At about melting-point
Ag	0.055	0.076	Mn	0.122	....
Al.	0.167	0.308	Mo	0.066	....
As	0.076	....	Na.	0 293	....
Au	0.030	....	Ni	0.109	0.161
Bi	0.030	0.030	Os	0.031	....
Ca	0.180	....	Pb	0.030	0.034
Cd	0.054	0.062	Pd	0.059	....
Co	0.106	0.204	Pt	0.032	0.046
Cr	0.104	....	Rh	0.058	....
Cu	0.086	0.118	Sb	0.048	0.054
Fe	0.116	0.162	Sn	0.055	0.059
Hg	0.033	0.032	Ti	0.112	....
Ir	0.030	0.040	Va	0.115	....
K	0.166	....	W	0.034	....
Li	0.941	....	Zn	0.093	0.122
Mg	0.246	....			

20. **Thermal conductivity,** or the power of transmitting heat, is characteristic for metals, all of which are good conductors. The conductivity depends upon the structure, the purity and the temperature. Metals that have been rolled or hammered are better conductors than when cast; small amounts of foreign matter reduce the conductivity; with increase of temperature the conductivity decreases. The relative conducting powers are as follows:

Ag 100, Cu 73.6, Au 53.2, Al 31.33, Cd 20.06, Sn 15.2, Pb 8.5, Pt 8.4, Sb 4.03, Bi 1.08, Hg 1.3.

The absolute heat conductivity K of some metals is given in Table 13. The letter K gives the value for the unit capacity for conducting heat, or that in which "a cube 1 cm. on a side transmits 1 gr.-cal. of heat per sec., with a drop of temperature from one surface to the other of 1° C."[1] In fact, the ratio of the absolute heat conductivity K and the electric conductivity σ for any given temperature is a constant.[2]

[1] Richards, J. W., "Metallurgical Calculations," McGraw Publishing Co., 1906, 1, 172.
[2] Schenck, R., "Physikalische Chemie der Metalle," Knapp, Halle, 1909, p. 20.

TABLE 12.—ABSOLUTE HEAT CONDUCTIVITY OF SOME METALS

Metal	K	Metal	K	Metal	K
Ag	1.096	Fe	0.17	Pt	0.17
Al	0.35	Hg	0.02	Sb	0.04
Au	0.70	Mg	0.38	Sn	0.14
Bi	0.02	Ni	0.14	Zn	0.26
Cd	0.21	Pb	0.08		
Cu	0.72	Pd	0.17		

21. Electric conductivity, or the power of transmitting electricity, is as characteristic for metals as is their thermal conductivity. It depends upon the length and thickness of the conductor, its molecular condition, purity and temperature.[1] The resistance, or inverse of conductivity, increases with the length and decreases with the cross-section of the conductor. Metals that have been rolled, hammered or drawn do not conduct as well as when they have been annealed; in fact the Brinell hardness number (§ 9) of a metal is indirectly proportional to the electric conductivity;[2] a very small amount of impurity may reduce the conductivity considerably, especially if it forms a solid solution;[3] an increase in temperature reduces the conductivity; fused metal is not as good a conductor as when solid. Table 13 gives the electric conductivity of some metals, the values given represent 1 c.c. of metal; Hg at 0° C. $= 1.063 \times 10^4 = 10{,}630$.

TABLE 13.—ELECTRIC CONDUCTIVITY OF SOME METALS AT 0° C.

Metal	Conductivity	Resistance 1,000 m., 1 sq. mm. Ω	Metal	Conductivity	Resistance 1,000 m., 1 sq. mm. Ω
Ag	679,000	15	Mg	230,000	43
Al	324,000	28	Na	211,000	48
As	28,600	350	Ni	144,200	70
Au	461,000	21	Os	105,300	95
Bi	9,260	1,080	Pb	50,400	195
Ca	95,000	75	Pd	....	107
Cd	144,100	70	Pt	63,500	108
Co	103,000	97	Sb	27,100	380
Cu	620,000	16.7	Sn	76,600	100
Fe	131,000	76	Ti	54,600	....
Hg	10,630	941	Tl	....	180
K	150,500	66	Zn	186,000	57
Li	119,000	88			

[1] Northrup-Suydam, *J. Frankl. Inst.*, 1913, CLXXV, 153.

[2] Kurnakoff-Schemtschuschny, *Zt. anorg. Chem.*, 1909, LXIV, 149.

[3] Guertler, *Zt. anorg. Chem.*, 1906, LI, 397.

The influence of gases upon the electric conductivity has been studied by Sieverts.[1]

22. **Magnetic Permeability.**[2]—Fe, Ni and Co are the so-called magnetic or ferro-magnetic metals.[3] According to whether the permeability of a metal is greater or smaller than that of air, it is classed as being paramagnetic or diamagnetic. Faraday arranged the metals in the following order:

Paramagnetic: Fe, Ni, Co, Mn, Cr, Ti, Pd, Pt and Os.

Diamagnetic: Bi, Sb, Zn, Sn, Cd, Hg, Pb, Ag, Cu, As, Ur, Ir, and W.

The majority of the metals have a permeability less than that of air; the difference between the paramagnetism of iron (2,000) and air is very much greater than that of the diamagnetism of Bi (0.9998) and air. When a steel magnet is brought to 750° C., it loses its magnetism; at a red heat a metal loses its power of becoming magnetized. Tables of magnetic susceptibility are given in Landolt-Börnstein-Roth, "Physikalisch-Chemische Tabellen," Springer, Berlin, 1912, p. 1233.

23. **Chemical Properties.**—The chemical behavior of metals is shown in their tendencies to combine with the elements O, S, Cl, etc., to form oxides, sulphides, chlorides, etc., and with acids to form salts. These groups are taken up severally in Chapter IV. The chemical relation of the metals to one another is probably best shown in a general way by their positions in the electrochemical series of elements, and in the periodical system.

TABLE 14.—ELECTROCHEMICAL SERIES OF ELEMENTS[4]

Cs+, Rb, K, Na, Li, Ba, Sr, Ca, Mg, Al, Cr, Mn, Zn, Ga, Fe, Co, Ni, Tl, In, Pb, Cd, Sn, Bi, Cu, H, Hg, Ag, Sb, Te, Pd, Au, Ir, Rh, Pt, Os, Si, C, B, N, As, Se, P, S, I, Br, Cl, O, F—.

In this series each element is electropositive to all those that follow it. Two metals in contact in the presence of an electrolyte form a galvanic couple causing the electropositive metal to be attacked by electrolysis.

Taking H as the central element, the elements in the direction of Cs are electropositive and include most metals; those in the direction of F are electronegative and include most of the non-metals. Metals usually form basic oxides and hydroxides; as a rule they do not enter into the composition of the acids; they rarely form hydrides; in general their haloids are not decomposed by heat or water; allotropic forms are the exception. The reverse is true with the non-metals. There are, of course, several exceptions to this general summary.

The periodic law states that the properties of an element are periodic functions of the atomic weights. The elements of a group show similar properties, and these become the more pronounced the greater the atomic weight.

[1] *Int. Zt. Metallogr.*, 1912, III, 37.

[2] Kotaro-Honda, *Ann. Phys.*, 1910, XXXII, 1027, 1003; *Rev. Mét. Extr.*, 1910, VII, 659, 661.

[3] Weiss, *Proc. Int. Assoc. Test. Mat.*, New York, 1912, II[17].

[4] Gore, L., "The Art of Electrolytic Separation of Metals," Van Nostrand, New York, 1892, p. 50.

TABLE 15.—THE PERIODIC SYSTEM

Group	I	II	III	IV	V	VI	VII	VIII
Series. 1	H 1			RH^4	RH^3	RH^2	RH	Composition of hydrogen compounds
2	Li 7	Be 9 1	B 11	C 12	N 14	O 16	F 19	
3	Na 23	Mg 24.4	Al 27	Si 28	P 31	S 32	Cl 35.5	
4	K 39 1	Ca 40	Sc 44	Ti 48.1	V 51.2	Cr 52.3	Mn 55	Fe 56, Co 59.1, Ni 58.5, Cu 63 3
5	(Cu) 63.3	Zn 65 4	Ga 69 9	Ge 72.5	As 75	Se 79	Br 80	
6	Rb 85 4	Sr 87.5	Y 89	Zr 90.7	Nb 93.5	Mo 95 9	.	Ru 101.7, Rh 102.9, Pd 106 7, Ag 107 9
7	(Ag) 107.9	Cd 112 4	In 114.8	Sn 119	Sb 120.2	Te 125.2	I 126 9	
8	Cs 132.8	Ba 137	La 138 5	Ce 141.5	Di? 145	.	.	
9	.	.	.	.	.	.	.	
10	.	.	Yb 173.2	.	Ta 182.8	W 184	.	Os 190.9, Ir 193.1, Pt 195.2, Au 197.2
11	(Au) 197.2	Hg 200	Tl 204	Pb 207.1	Bi 208	.	.	
12	.	.	.	Th 232.4	.	U 238.5	.	
	R^2O	R^2O^2	R^2O^3	R^2O^4	R^2O^5	R^2O^6	R^2O^7	Composition of highest saline oxides.
				RO^2		RO^3		RO^4

Groups I and II contain the electropositive elements whose oxides form strong bases; groups VI and VII the electronegative elements whose oxides form strong acids; groups III, IV and V occupy an intermediary position, and the elements of group IV form neither strong bases or acids. In a series, the characteristic properties of the elements decrease from the end-groups toward the central.

24. Corrosion.[1]—The corrosion and pitting of metals by solutions of acids and salts is a phenomenon to which most metals are liable. They may be effected in three ways: By chemical action, in which a metal comes in contact

[1] Cushman A. S.,-Gardner, H. A., "The Corrosion and Preservation of Iron and Steel," McGraw-Hill Book Co., New York, 1910.

Sang, A., "The Corrosion of Iron and Steel," McGraw-Hill Book Co., New York, 1910.

Friend, J. N., "The Corrosion of Iron and Steel," Longmans, Green & Co., London and New York, 1911.

with an acid and is dissolved; by a combined action of a corroding liquid and the atmosphere; and by electrolytic action. Recent research[1] has paved the way for the prevailing view that all corrosion is electrolytic in character. Metals that are really chemically pure are very slowly dissolved. The presence of even a small amount of impurity greatly hastens solution owing to the formation of voltaic couples. Ordinary metals always contain some impurities which are not uniformly distributed. Further, in metals which are chemically pure, but not physically uniform, a difference in potential is set up between the harder and softer parts. Forgeable metal which has been rolled, hammered or drawn, is not uniformly strained; and in cast metal any uneven contraction due to quicker or slower cooling of parts causes a lack of physical uniformity.

[1] Whitney, *J. Am. Chem. Soc.*, 1903, XXV, 294.
Dunstan-Jowett-Goulding, *Tr. Chem. Soc.*, 1905, LXXXVII, 1548.
Walker-Cederholm-Bent, *J. Am. Chem. Soc.*, 1907, XXIX, 1251.
Cushman, *Proc. Am. Soc. Test. Mat.*, 1907, VII, 211; *J. Am. Chem. Soc.*, 1908, LXXXIII, 1356.
Friend, *J. I. and St. I.*, 1908, II, 5; 1909, II, 172–257.
Tilden, *Tr. Chem. Soc.*, 1908, XCIII, 1356.
Walker, *Proc. Am. Soc. Test. Mat.*, 1908, VIII, 238; *J. I. and St. I.*, 1909, I, 69.
Friend-Brown, *J. I. and St. I.*, 1911, I, 125.
Longmuir, *op. cit.*, 1911, I, 147.
Bengough, *J. Inst. Met.*, 1911, V, 28.
Dunstan-Hill, *Tr. Chem. Soc.*, 1911, XCIX, 1835.
Editor, *Met. Chem. Eng.*, 1912, X, 45.

CHAPTER III

ALLOYS

25. General.[1]—An alloy is a solidified solution of two or more metallic substances. Certain non-metallic elements, such as C, Si, P, and metallic oxides, such as Cu_2O, take the places of metallic substances, as combining with metals they do not deprive them of their metallic character. If Hg forms one

[1] GENERAL.—Gages, L., "Les Alliages Métalliques," Gauthier-Villars, Paris, 1903. Tassily, E., "Étude des Propriétés Physiques des Alliages Métalliques," Joanin & Co., Paris, 1904. Guillet, L., "Étude théoretique des Alliages Métalliques," Dunod, Paris, 1904. Guillet, L., "Les Alliages Métalliques," Dunod, Paris, 1906. Howe, H. M., "Iron, Steel and Other Alloys," Cambridge, Mass., 1906. Hiorns, A. H., "Metallography," Macmillan, London, 1902. Gulliver, G. H., "Metallic Alloys, Their Structure and Constitution," Griffin & Co., London, 1913. Goerens, P. and Ibbotson, F., "Introduction to Metallography," Longmans, Green & Co., London, 1908. Bornemann, K., "Die binären Legirungen," Knapp, Halle, I, 1909; II, 1912, in process of publication. Cavalier, J., "Leçons sur les Alliages Métalliques," Vuibert and Nony, Paris, 1909. Guertler, W., "Metallographie," Borntraeger, Berlin, I, 1911, in process of publication. Law, E. F., "Alloys and Their Industrial Applications," Griffin & Co., London, 1909. Ruer, R., and Mathewson, C. H., "The Elements of Metallography," Wiley, New York, 1909. Sexton, A. H., "Alloys, Non-ferrous," Scientific Pub. Co., Manchester, 1909. Desch, C. H., "Metallography," Longmans, Green & Co., London, 1910. Dessau, B., "Die physikalisch-chemischen Eigenschaften der Legirungen, Vieweg, Brunswick, 1910. Robin, F., "Traité de Métallographie," Honnann & Fils, Paris, 1912. Contributions à l'Étude des Alliages," Chamerot-Renouard, Paris, 1901. "Reports Alloys Research Committee," English Institution of Mechanical Engineers, since 1901. Campbell, W., Structure of Metals and Binary Alloys, *J. Frankl. Inst.*, 1902, CLIV, 1, 131, 201; *Metallographist*, 1902, V, 286. Campbell, W., Structure of Alloys, *Proc. Am. Inst. Test. Mat.*, 1904, IV, 381. Mathews, J. A., Alloys as Solutions, *Min. Ind.*, 1901, X, 723. Mathews, J. A., Upon the Constitution of Binary Alloys, *J. Frankl. Inst.*, 1902, CLIII, 1. Stead, J. E., Metallic Alloys, *Cleveland Inst. Eng.*, 1900; *Metallographist*, 1902, V, 110. Roozeboom, H. W. B., Erstarrungspunkte der Mischcrystalle zweier Stoffe, *Zt. phys. Chem.*, 1899, XXX, 385, 412.

SPECIAL.—Ledebur, A., "Die Metallverarbeitung auf chemisch-physikalischem Wege," Vieweg, Brunswick, 1882. Guettier, A., "Le Fondeur en Métaux," Bernard & Co., Paris, 1890. Wüst, F., "Handbuch der Metallgiesserei," Voigt, Weimar, 1897. Thurston, R. H., "Brasses, Bronzes and Other Alloys," Wiley, New York, 1900. Hiorns, A. H., "Mixed Metals or Metallic Alloys," Macmillan, London, 1902. Sack, M., "Bibliographie der Metallegirungen," Voss, Hamburg and Leipsic, 1903. Buchetti, J., "Les Alliages Métalliques," Béranger, Paris, 1905. Ledebur, A., "Die Legirungen," Krayn, Berlin, 1906. Brannt, W. T., "The Metallic Alloys," Baird, Philadelphia, 1908. Wüst, F., "Légir und Lötkunst," Voigt, Leipsic, 1908. Krupp, A., "Legirungen," Hartleben, Vienna, 1909. Guillet, L., "Traitments Thermiques des Products Métallurgiques," Dunod-Pinat, Paris 1909. Kaiser, E. W., "Zusammensetzung der Gebräuchlichen Metallegirungen," Knapp-Halle, 1911–12, reprint from *Metallurgie*, 1911, VIII, 257, 296. Fenchel, A., and Morris, H. J., "Metallurgy," J. Bale, Sons & Danielson, Ltd., London, 1911. Schott, E. A., "Die Metallgiesserei," Voigt, Leipsic, 1913.

PERIODICAL.—Internationale Zeitschrift für Metallographie, Editor, W. Guertler, Borntraeger, Berlin, Vol. 1, 1911.

of the essential metals, the alloy is called an amalgam. A molten alloy is a homogeneous mixture, that is, a solution of two or more metals. When this solution begins to freeze, it may split into its constituents, each of which has a distinct entity. The constituents can be pure metals, eutectic mixtures, solid solutions and definite chemical compounds. An alloy may contain at the same time several of these constituents. Molecular rearrangements take place, however, not only during, but also after, solidification while the mass cools to ordinary temperature.

According to the number of constituents that go to make up alloys, these are spoken of as binary, ternary and quaternary.

26. Constitution of Binary Alloys.[1]—In order to study the constitution or structural composition of binary alloys, it is necessary to classify them. The first systematic classification is probably that of Roozeboom,[2] which in its general features forms the basis of the following classification:

Classification of Binary Alloys.—*A*. The components do not form any chemical compounds with one another.

1. The liquid metals are mutually completely soluble and form a homogeneous solution.
   (*a*) The solid components are mutually insoluble and form alloys which for all compositions form two constituents when solidifying.
   (*b*) The solid components are mutually partly soluble and form alloys which between certain limiting compositions show two constituents when solid.
   (*c*) The solid components are mutually completely soluble and for all compositions form alloys which when solid show only one constituent.
2. The liquid metals are mutually only partly soluble and between certain compositions form conjugate liquid solutions.
   (*a*) The same as (*a*) above.
   (*b*) The same as (*b*) above.
   (*c*) Does not exist.
3. The liquid metals and solid metals are mutually insoluble.

B. The components form one or more chemical compounds with one another.

1. When liquid, the compound and component metals are mutually soluble in all proportions.
   (*a*) When solid, the compound and component metals are mutually insoluble.
   (*b*) When solid, the compound and component metals are mutually partly soluble.
   (*c*) When solid, the compound and component metals are mutually completely soluble.

[1] This discussion follows in the main points the "Notes on Metallography" prepared by Professor C. R. Hayward for the course given to mining and metallurgical students of the Massachusetts Institute of Technology.

[2] *Zt. phys. Chem.*, 1899, xxx, 385, 413.

2. When liquid, the metals and compound are mutually partly soluble, and between certain limiting compositions form conjugate solutions.
   (*a*) Same as (*a*) above.
   (*b*) Same as (*b*) above.
   (*c*) Does not exist.

3. When just liquid, the metals and compound are mutually completely insoluble.
   (*a*) Same as (*a*) above.
   (*b*) and (*c*) do not exist.

The main division, *A* and *B*, is characterized by the absence or presence of chemical compounds[1] of the component metals. While under the microscope only one constituent is visible with such a chemical compound, the physical properties differ from the component metals, and while the components are combined in atomic proportions, the chemical formulæ of these *intermetallic compounds* do not fall into line with the prevailing conception of valencies of atoms composing the molecules. $\overset{III}{Au}\,\overset{IV}{Sn}_4$,[2] $\overset{III}{Au}_2\,\overset{IV}{Al}$,[3] $\overset{III}{Au}_3\,\overset{II}{Zn}_5$,[4] are examples such compounds. It appears that the more similar the chemical behavior of the two components, the less regular is the formula of the compound.[5] A full explanation of the peculiarities of these compounds is lacking; in all other respects the physical and chemical behavior of intermetallic compounds is similar to that of the ordinary compounds.

The subdivisions 1, 2, and 3 are based upon the mutual solubility of components when molten. Remembering that alloys have been defined as solidified solutions, their behavior as regards solubility must be the same as that of liquid solutions. Two liquids, like alcohol (C_2H_6O) and water, are reciprocally soluble in all proportions and form a homogeneous solution. When an alloy of two metals, such as Ag and Au, solidifies, the constituents remain as intimately associated as when they were in the liquid state; they cannot be distinguished from one another by the microscope or any other physical means. They form a SOLID SOLUTION, sometimes still called isomorphous mixture or mixed crystal.

Phenol,[6] C_6H_6O, and water are soluble only within certain limits depending upon temperature. If the two bodies are brought together in a vessel, there will be formed two saturated solutions existing together (*conjugate solutions*), at the bottom the heavier phenol saturated with water, on the top the lighter water saturated with phenol. If the temperature is changed, the existing equilibrium will be disturbed, *i.e.*, the saturation-points will be shifted, and the heights of

[1] Determination by chemical methods: Portevin, *Rev. Mét.*, 1912, IX, 884.
[2] Vogt, *Zt. anorg. Chem.*, 1905, XLVI, 60.
[3] Heycock-Neville, *Phil. Trans.*, 1900, CXCIV, A, 201.
[4] Vogel, *Zt. anorg. Chem.*, 1906, XLVIII, 319.
[5] Tammann, *Zt. anorg. Chem.*, 1906, XLIX, 113; 1907, LV, 289.
Kurnakoff, *op. cit.*, 1900, XXIII, 439.
[6] Rothmund, *Zt. phys. Chem.*, 1898, XXVI, 433.
Heyn, *Berg. Hüttenm. Z.*, 1900, XLIX, 559.

the two layers changed. Similar phenomena[1] are observed with metals of limited solubility, such as Pb—Zn, Bi—Zn, Pb—Al, Bi—Al, Cd—Al.

Class 3 finds its counterpart in oil and water which are insoluble in one another.

The second sub-headings *a*, *b*, and *c*, are derived from the behavior of the components when in the solid state as resembling or differing from that in the liquid.

This general classification holds good also for fused metallic sulphides, arsenides, antimonides, silicates and other salts.

27. **Binary Alloys, with Components not Forming Chemical Compounds, and Completely Soluble in the Liquid, Insoluble in the Solid State.**—The cooling or temperature-time curve of a pure metal has been given in Fig. 12, *A—B*, it is reproduced in Fig. 13, *I*.

If two metals *A* and *B* are melted together, either may be considered as a solvent of the other. If *A* is the solvent, its freezing-point will be lowered by the addition of *B*. If a molten alloy with *A* as solvent and *B* as solute is cooled to freezing, *A* crystallizes out as pure metal as long as *A* and *B* are mutually insoluble in the solid state; *A* will retain some *B* in solid solution, if the solid metals are partly soluble in one another. According to the law of Raoul, the depression of the freezing-point is proportional to the number of molecules of metal *B* added to *A* (molecular depression). The freezing of such an alloy is shown in Fig. 13, *II*. The first break in the curve has been lowered from t_1 to t_2. The horizontal branch of pure metal, *A—B*, Fig. 12, or t_1, Fig 13, *I*, has assumed the sloping form t_2, Fig. 13, *II*, and the sharp point *b*, Fig. 12, *AB*, has been rounded off at t_2, Fig. 13, *II*. This is due to the fact that at the moment the first crystal of the solvent *A* separates, the remaining fluid metal (the mother metal) becomes enriched in the solute *B*, and has its freezing-point correspondingly lowered. As the cooling proceeds, the primary (excess) metal *A* continues to separate until the mother metal has become saturated with metal *B*. Solvent and solute now solidify side by side and cause a second halt at t_5. As there can be no further change in composition, the temperature of solidification must remain the same until the whole has solidified. While freezing, the molten metal splits up into a conglomerate of crystals of *A* and *B* forming interstratified plates. Now, only, will the temperature fall again through the cooling of the solidified alloy which consists of excess metal *A* and eutectic *A–B*. Fig. 13, *III* and *IV*, shows how the freezing-point of *A* is lowered by increasing amounts of *B*, and the length of the horizontal t_5 extended.

In general, the nearer the composition of the alloy approaches that of saturation, the lower will be the freezing-point, the shorter and flatter the branch $t_{1\cdot2\cdot3\cdot4}$–t_5, and the longer the horizontal branch t_5. Coming to the alloy, Fig. 13, *V*, there is no excess metal; the first freezing will take place at the temperature of t_5, when the alloy will solidify as a whole and split into parallel

[1] Alder-Wright-Thompson, *Proc. Roy. Soc.*, 1888 89, XLV, 461; 1890, XLVIII, 25; 1890–91, XLIX, 157, 174; 1892, L, 372; 1892–93, LII, 11, 530.
Spring-Romanoff, *Zt. anorg. Chem.*, 1896, XIII, 29.

plates. This saturated alloy is called the *eutectic alloy* or simply the *eutectic*, and the proportion of the two constituents the *eutectic ratio.*

In the same Fig. 13, curve *VII* represents the cooling curve of pure metal *B*, and curve *VI* an addition of *A* to *B*. The freezing-phenomena of curves *VII* to *V* are just the reverse of those discussed in the curves *I* to *V*.

If the results obtained from the cooling curves of Fig. 13 are plotted into a rectangular system of coordinates in which the abscissa represents composition

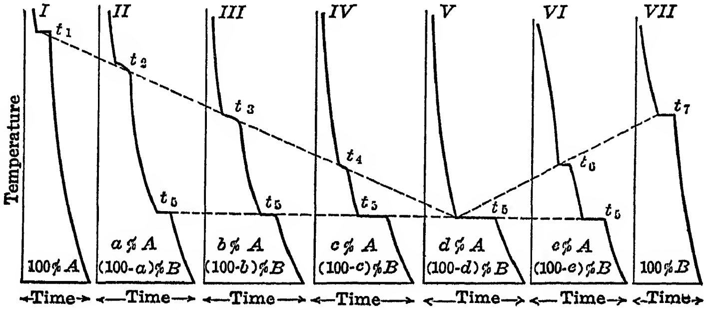


FIG. 13.—Cooling curves for a series of alloys.

and the ordinate temperature, there is obtained the V-shaped *freezing-point curve*, *A E B*, Fig. 14, which shows the freezing-points of all possible mixtures of two metals *A* and *B*, which form no chemical compounds, are soluble when liquid and insoluble when solid. The diagram as a whole is called the ***constitutional diagram;*** if it contains lines, as in Fig. 22, showing transformation taking place in the solid state, we have the ***equilibrium diagram;*** the second term including all the facts is often used for the first. The diagram, Fig. 14, consists

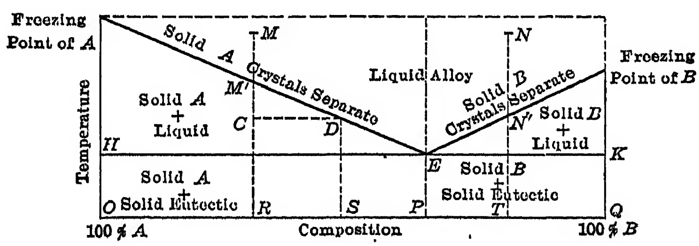


FIG. 14.—Freezing or time-composition curves derived from Fig. 13.

of two branches *A E* and *E B* cutting each other at the ***eutectic point***, ***E***, on the ***eutectic line***, *H K*, and forming the ***eutectic angle E***. Branch *A E* represents the separation of excess metal *A*, branch *E B* that of *B*, point *E* the simultaneous separation of *A* and *B* or that of the eutectic. Above the freezing-point curve *A E B*, or ***liquidus***, there is a homogeneous liquid alloy; below the ***melting-point curve***, *H E K*, or ***solidus***, there is solid alloy; in triangle *A E H*, solid metal *A* and liquid alloy; in triangle *B E K*, solid metal *B* and liquid alloy; in rectangle *H E P O*, solid metal *A* and solid eutectic; in rectangle *K E P Q*, solid metal *B* and solid eutectic; on line *E P*, solid eutectic.

The characteristics of the eutectic are, that it has the lowest melting-point of the series of alloys of the metals composing it; that its composition, and hence its freezing-point, are constant,[1] and that it is not a chemical compound but a distinct mixture of its two components. Assuming A as solvent and B as solute, all alloys to the left of the eutectic point are said to be *hypoeutectic*, as they contain less of B than the eutectic ratio; all alloys to the right of the eutectic point are *hypereutectic*, as they contain more B than the eutectic ratio.

An alloy of the composition R, when molten, can be represented by the point M. Upon cooling, it will remain unchanged until the point M' is reached, when metal A will begin to crystallize. The corresponding enrichment of the liquid alloy in B, accompanied by lowering of freezing-point, is traced on line $M'E$; metal A continues to fall out, the liquid becomes richer in B until it has reached the eutectic ratio at E, when it solidifies and at the same time splits into parallel plates. If the alloy M cools to the temperature indicated by point C and is suddenly chilled, it will consist theoretically of crystals A and another metal of the composition S, the latter being found by drawing from C a parallel to the abscissa to the intersection with A E at D, and dropping from this a perpendicular on to the abscissa. Changes taking place in the alloy T, repre-

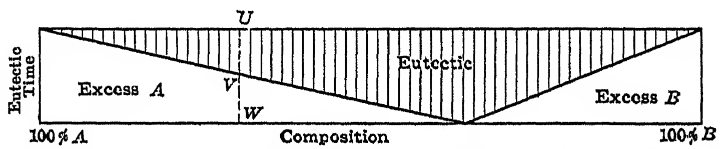


FIG. 15.—Time-composition curve with relative amounts of eutectic and excess metal in solid alloy.

sented when molten by point N, are the same as those noted with molten alloy M, excepting that the excess metal is B instead of A.

Comparatively few metals belong to this class of alloys; some of these are Zn–Cd,[2] Pb–Sn,[3] Pb–Ag,[4] Pb–Sb,[5] Au–Tl.[6]

Fig. 15 gives a diagrammatic representation of Tammann's[7] *time-composition curve* and with it of the proportion of excess metal and eutectic in a solid alloy. The time t_5, Fig. 13, taken by the eutectic to solidify is considered to be proportional to the amount of eutectic present. If the *eutectic times* are plotted as ordinates against compositions as abscissæ, there will be obtained the cross-hatched triangle of Fig. 15, showing how the times of eutectic solidification, greatest at E, Fig. 14, diminish toward H and K. If a rectangle is drawn on the composition abscissa with the maximum time of eutectic solidification as

[1] Benedicks-Apri, *Metallurgie*, 1907, IV, 416.
[2] Hindricks, *Zt. anorg. Chem.*, 1907, LV, 415.
[3] Rosenhain-Tucker, *Phil. Trans.*, A, 1908, CCIX, 89.
[4] Friedrich, *Metallurgie*, 1906, III, 396.
[5] Gonterman, *Zt. anorg. Chem.*, 1907, LV, 419.
[6] Levin, *op. cit.*, 1905, XLV, 31.
[7] Tammann, *Zt. anorg. Chem.*, 1903, XXXVII, 303; 1905, XLV, 24; XLVII, 280. Portevin, *Rev. Mét.*, 1910, VII, 1149.

height, there are obtained three areas representing the three constituents of the alloy-series A B. Any vertical drawn such as U W will be divided into two parts which are proportional to the amounts of eutectic and excess metal present. Thus in the alloy R, Fig. 14, eutectic and excess metal are present in the ratio $U\ V : V\ W$. If the height of the rectangle is divided into 100 parts, the ratio will be given in percentage.

Instead of expressing the composition of an alloy in per cent. by weight, physical chemists, especially the German, are using atomic percentages, Fig. 30, as these bring out the relationship that exists between freezing-point depression and atomic weight. In such a diagram, the 100 divisions of the abscissa represent the sum of the atomic weights of the two metals of the alloy reduced to 100. The following formulæ of Hoffmann[1] give the manner of converting per cent. by weight into atomic per cent., and *vice versa,* for binary and ternary alloys.

Conversion of Atomic Per Cent. into Weight Per Cent. and Vice Versa.—Component 1.2.3; atomic weight $A_1.A_2.A_3$; atomic per cent. $q_1.q_2.q_3$; weight per cent. $p_1.p_2.p_3$.

Fig. 16.—Eutectoid Fe–Fe_3C, Pearlite. (Osmond.)

(*a*) Atomic per cent. to weight per cent. in binary alloys:

$$p_1 = \frac{100 q_1 \times A_1}{q_1 \times A_1 + q_2 \times A_2} \text{ where } q_1 + q_2 = 100$$

(*b*) Weight per cent. to atomic per cent. in binary alloys:

$$q_1 = \frac{100 p_1 \times A_2}{p_1 \times A_2 + p_2 \times A_1} \text{ where } p_1 + p_2 = 100$$

(*c*) Atomic per cent. to weight per cent. in ternary alloys:

$$p_1 = \frac{100 q_1 \times A_1}{q_1 \times A_1 + q_2 \times A_2 + q_3 \times A_3} \text{ where } q_1 + q_2 + q_3 = 100$$

[1] *Metallurgie,* 1912, IX, 133.

(d) Weight per cent. to atomic per cent. in ternary alloys:

$$q_1 = \frac{100 p_1 \times A_2 \times A_3}{p_1 \times A_2 \times A_3 + p_2 \times A_1 \times A_3 + p_3 \times A_1 \times A_2} \text{ where } p_1 + p_2 + p_3 = 100$$

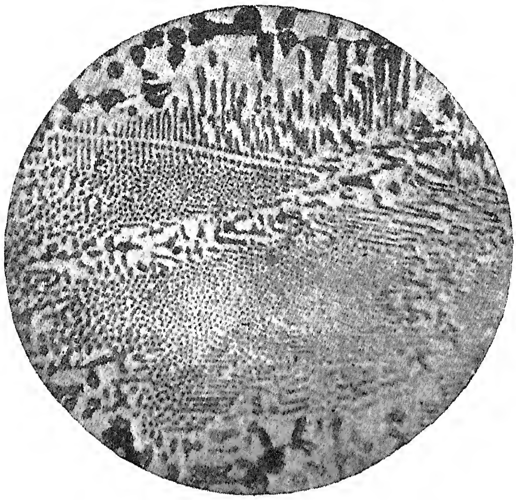


FIG. 17.—Eutectic Fe–Fe_3P. (Stead.)

FIG. 18.—Eutectic Ag–Pb. (Stead.)

There remains the discussion of the structure of the eutectic alloy. In general, an eutectic is laminated. Stead[1] distinguishes three forms, the curviplanal, the honey-combed or cellular, and the rectiplanal. The curviplanal eutectic has curved plates in juxtaposition. The eutectoid Fe–Fe_3C, Fig. 16

[1] *Proc. Cleveland Inst. Eng.*, February, 1900; *Metallographist*, 1902, v, 114.

(Osmond), shows the parallel arrangement of dark-curved bands of ferrite (Fe), and white ones of cementite (Fe_3C). The honey-combed or cellular structure is represented by the eutectic of saturated solution of Fe_3P in P–Fe_3P in Fig. 17 (Stead). Here dark crystallites of Fe alternate with the white of Fe_3P. The rectiplanal eutectic of Pb–Ag, Fig. 18 (Stead), shows alternating plane plates of the two constituents arranged in triangular groups.

In Figs. 19 to 21 (Hayward) are given three photomicrographs of the Cu–Ag series of alloys. These form the eutectic alloy containing 28 per cent. Cu

FIG. 19.—Eutectic Cu–Ag. (Hayward.)

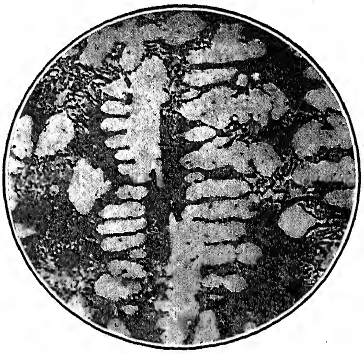


FIG. 20.—Eutectic Cu–Ag with excess Ag. (Hayward.)

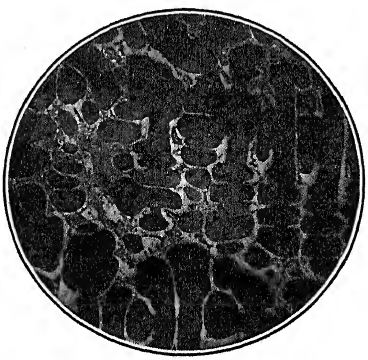


FIG. 21.—Eutectic Cu–Ag with excess Cu. (Hayward.)

and 72 per cent. Ag. More recent investigations of Lepkowski[1] have shown that both Cu and Ag form, to a small extent, solid solutions with Ag and Cu. Fig. 19 brings out clearly the laminated curviplanal structure. Fig. 20 represents the alloy Cu 20 per cent.–Ag, 80 per cent., with its large white islands of the excess-metal Ag floating in a sea of eutectic; while in Fig. 21, standing for the alloy Cu 70 per cent.–Ag 30 per cent., the dark patches of excess-metal Cu are surrounded by eutectic.

[1] *Zt. anorg. Chem.*, 1908, LIX, 285.

28. Binary Alloys, with Components not Forming Chemical Compounds, and Completely Soluble in the Liquid, Partly Soluble in the Solid State.—While there exist only a few of the alloys belonging to the preceding class, the alloys of this division are numerous as *e.g.*, Ag–Bi,[1] Ag–Cu,[2] Au–Bi,[3] Bi–Pb,[4] Cd–Pb and Cd–Sn,[5] etc.

The equilibrium diagram for a series of such alloys is given in Fig. 22. The general form of the freezing-point curve *A E B* is the same as that given in Fig. 14; the eutectic line, however, does not extend to the limits of the diagram, but only to the points *M* and *N* which mark the limits of solubility of *B* in *A*, and of *A* in *B* when just solidified. The decrease of solubility of the two series of

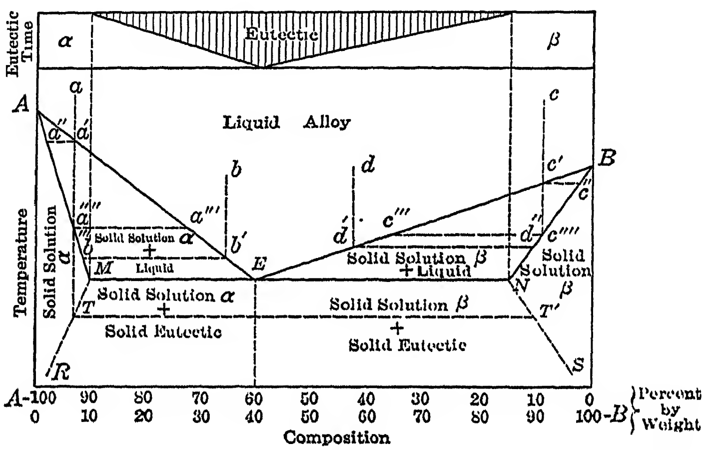


FIG. 22.—Equilibrium diagram, no chemical compound, liquid metals mutually completely soluble, solid metals partly soluble.

solid solutions with falling temperature is traced by the diverging *solubility curves M R* and *N S*, and the increase with rising temperature by the converging lines *M A* and *N B*. The constituents of each of the seven fields are inscribed in the diagram. Its full meaning is made clear by following the changes that take place in five alloys during solidification. The alloy represented in composition and temperature by *a*, upon cooling, will begin to freeze when it reaches the point *a′* on the liquidus *A E*, but, instead of depositing pure *A*, as in Fig. 14, a crystal of solid solution of *B* in *A* will separate, represented in composition by *a″*, found by drawing a horizontal from *a′* to the solidus *A M*. By the separation of this solid solution the freezing-point of the molten mother-metal is lowered, as it has become richer in *B;* this process continues as discussed in

[1] Petrenko, *Zt. anorg. Chem.*, 1906, L, 133.

[2] Friedrich-Leroux, *Metallurgie*, 1907, IV, 293.
Lepkowski, *Zt. anorg. Chem.*, 1908, LIX, 285.

[3] Vogel, *op. cit.*, 1906, L, 145.

[4] Portevin, *Rev. Mét.*, 1907, IV, 389.
Stoffel, *Zt. anorg. Chem.*, 1907, LIII, 137.

[5] Kapp, *Ann. Phys.*, 1901, VI, 754.
Stoffel, *Zt. anorg. Chem.*, 1907, LIII, 137.

connection with Fig. 14. The compositions of the crystals of solid solution which separate, change along the line $A\ M$, the crystals becoming richer in B as does the mother-metal.

If the separating mass of solid crystals is to be in equilibrium with the liquid mother-metal, there must take place a diffusion of metal A from an earlier crystal to attain the composition of a later which is in equilibrium with the molten metal. The process taking place may be imagined to be the solution by the liquid metal of the small amount of A in excess of the equilibrium composition and its immediate redeposition with the correct amount of B to meet the requirements of the new equilibrium. However, unless the cooling of the alloy is extremely slow, the diffusion will be imperfect, the composition of the first crystals will remain almost unchanged, the earlier crystals will be enclosed by layers of later crystals which hinder the direct contact with the liquid necessary for quick diffusion. As the solidification continues, the crystals therefore gradually change in composition. If perfect diffusion had taken place, the crystals separating at point a'''' (which lies on the vertical drawn through a' and represents the complete solidification of the alloy) as well as the crystals that were deposited between a'' and a'''', that is, the whole mass of crystals ought to have the same composition as the original alloy.

The last particle of molten alloy to freeze would have the composition a''' if diffusion were perfect; as this is not the case, its composition will lie somewhere between a''' and E. On account of the same lack of perfect diffusion, the metal separating along $A\ M$ will consist of crystal aggregates in which the amount of B gradually increases from center to circumference.

Following the changes in alloy b, freezing begins at b', the crystals separating have the composition b'''; as cooling progresses the composition of the molten mother-metal will change along $b'\ E$, that of the crystals along $b'''\ M$. At the eutectic point E, the highest saturation of B in A, or the eutectic ratio, has been reached. The liquid of eutectic composition solidifies and at the same time splits into parallel plates, but these, instead of consisting of bands of pure A, and pure B, as in Fig. 14, are composed of solid solution of B in A, and of A in B, of the compositions M and N respectively. All the alloys to the left of the eutectic line will be composed of solid solution α and eutectic mixture.

If the molten alloy had the eutectic composition indicated by E, freezing would begin and end at E, and the solidified alloy would be composed solely of eutectic mixture.

The freezing phenomena of alloy d are similar to those of b, with the difference that crystals of β solid solution, *i.e.*, of metal A in B, of the composition d'' would separate and undergo changes similar to those of b'''. The freezing of alloy c resembles in its changes that of a, solid solution β replacing α. All alloys to the right of the eutectic line and to the left of $N\ S$ are composed of solid solution β and eutectic mixture.

There remain to be considered the changes taking place at a temperature below that of the eutectic line $M\ N$. At the eutectic temperature, M represented the amount of B required to saturate A, and N the amount of A neces-

sary to saturate B. The mutual solubility decreases along the lines $M\ R$ and $N\ S$ as the temperature falls. The alloy a at solidification consisted of a'''', an undersaturated solid solution of B in A, the saturation-point lying at M. As the temperature falls, the amount of B that A can hold in solid solution diminishes along $M\ R$, which line intersects the continuation of line a–a'''' at T. Here the solid solution α is saturated.

If held for some time at this temperature, a second constituent of the composition T' may appear, on account the original homogeneous alloy breaking up into two conjugate solutions (saturated solutions in contact with one another). As the temperature falls below T, an interchange takes place and at the left of the eutectic line solution β increases over that of α. These transformations in the solid can be brought about by annealing, *i.e.*, holding an alloy at the required temperature for the period of time necessary to bring about the change.

29. Binary Alloys, with Components not Forming Chemical Compounds, and Completely Soluble in the Liquid and Solid States.—Alloys under this heading

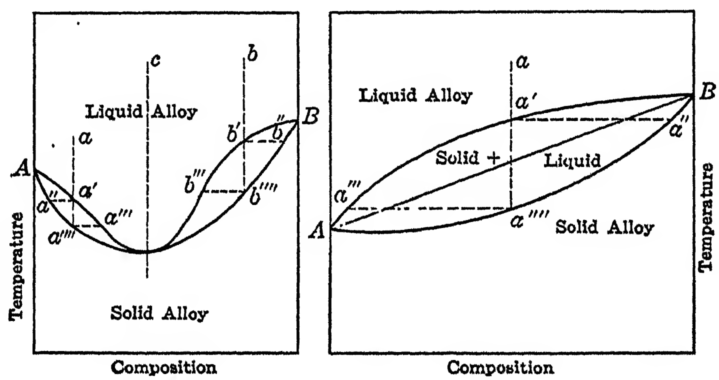


FIG. 23. FIG. 24.

FIGS. 23 and 24.—Equilibrium diagram, no chemical compounds, liquid and solid metals mutually completely soluble.

may be considered as forming a special case of those discussed in §28. Supposing that the solubilities of the metals A and B in Fig. 22 increase to a degree that the points M and N merge into E, the eutectic line $M\ N$ will disappear, but not the liquidus nor the solidus which usually are curved instead of straight lines. Fig. 23 represents such a case in which the upper curve (liquidus) and the lower (solidus) pass through a minimum. The liquid alloy a upon cooling to the freezing-point a' will deposit crystals of the composition a''. As the temperature falls, the composition of the liquid mother-metal will change from a' to a''', and that of the crystals from a'' to a''''. If diffusion is complete, the composition of the solid alloy will be the same as that of the original; if not, the alloy will be made up of crystal aggregates varying in composition from the center to the outside. The liquid alloy c is the one of the lowest freezing-point; it will solidify to a homogeneous solid. The liquid alloy b will begin to freeze at b', as did a at a', depositing crystals of the composition b'', which will have

changed to b'''' when the solidification is complete. The alloy-series Cu–Mn[1] is characteristic for this class.

In Fig. 24 is represented a second case in which the freezing-points of the alloys lie entirely between the freezing-points of the pure metals *A* and *B;* the liquidus is curved upward and the solidus downward. In the selective freezing taking place during solidification, the separating crystal is always richer in the metal of higher freezing-point than the original alloy, and the mother-metal correspondingly poorer. The alloy *a* beginning to freeze at a' causes crystals of the composition a'' to form; during the continued solidification the composition of the mother-metal changes from a' to a''', and that of the crystals from a'' to a''''. With complete diffusion the solidified alloy will be homogeneous. With incomplete diffusion it will have a heterogeneous structure, the compositions of the crystals varying from center to outside. The alloy-series Au–Pt,[2] Cu–Ni,[3] Pd–Ag,[4] Pd–Au[5] and Pd–Cu[6] belong to this class.

30. Binary Alloys, with Components not Forming Chemical Compounds, and Partly Soluble in the Liquid State.—The limited mutual solubility of two fluid metals has been referred to in § 26 by comparing their behavior with that of phenol and water. Fig. 25 gives the solubility curve for two metals with which the solubility increases as the temperature rises. Let *x* represent the largest amount of *A* soluble in *B*, and *z* that of *B* in *A* at the temperature represented by line *A B;* alloys with a composition lying between *A* and *x* will be unsaturated solutions of *B* in *A*, and alloys between *B* and *z* unsaturated solutions of *A* in *B;* alloys lying between *x* and *z* will consist of two saturated solutions having the compositions *x* and *z* respectively. If the two conjugate solutions have the same specific gravity, their structures will resemble an emulsion; if there is a difference, the lighter will float on top of the heavier, and the line of demarcation will be the more pronounced the greater the difference. If the temperature is raised, the mutual solubility of the components is increased and grows along the lines *x y* and *z y* which form the *saturation-point curve.* At the temperature indicated by line *ad*, metal *A* will absorb *ab* per cent. of metal *B*, and *B* will dissolve *cd* per cent. of *A*. At temperature *y* the metals

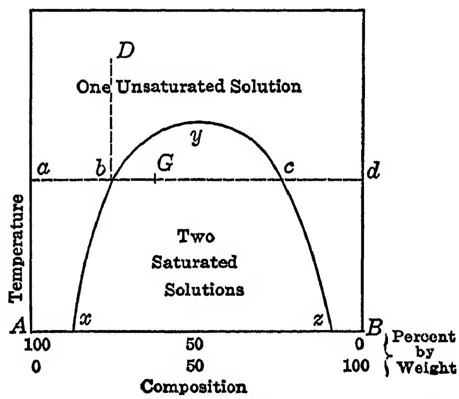


FIG. 25.—Equilibrium diagram, no chemical compounds, liquid metals partly soluble.

[1] Schemtschuschny-Urazoff-Rykowkoff, *Zt. anorg. Chem.*, 1908, LVII, 253.
[2] Doerinckel, *Zt. anorg. Chem.*, 1907, LIV, 333.
[3] Kurnakoff-Schemtschuschny, *op. cit.*, 1907, LIV, 153.
[4] Ruer, *op. cit.*, 1906, LI, 315.
[5] Ruer, *op. cit.*, 1906, LI, 391.
[6] Ruer, *op. cit.*, 1906, LI, 223.

A and B are soluble in all proportions; the point y is the *critical point*, the temperature, the *critical temperature*, and the composition, the *critical concentration*.

A liquid alloy represented in composition and temperature by D will be a homogeneous solution, as it lies outside of the solubility curve. If the temperature is reduced, the single solution will be broken up into a saturated b-solution and a saturated c-solution as soon as the temperature has fallen to the point b on line *ad;* with a further fall the mutual solubility of A and B will decrease in the direction of $y\ x$ and $y\ z$.

Supposing there are added at temperature *ad*, to the pure metal A, successive portions of B, the latter will dissolve in A until the composition b, the saturation point on the solubility curve, is reached. With a further addition of B, a second saturated solution of the composition c will appear. Adding enough B to give the alloy the composition G, the ratio of b-solution and c-solution present will be as $Gc:Gb$. As more B is added, the point G is shifted toward c, the ratio $Gc: Gb$ will decrease until at c the b-solution will disappear and the alloy consist of a single saturated solution of A in B. If still more B is added to the alloy, the solution of A in B becomes more dilute, but does not change in structure.

31. Binary Alloys, with Components not Forming Chemical Compounds, and Partly Soluble in the Liquid and Insoluble in the Solid State.—The diagram

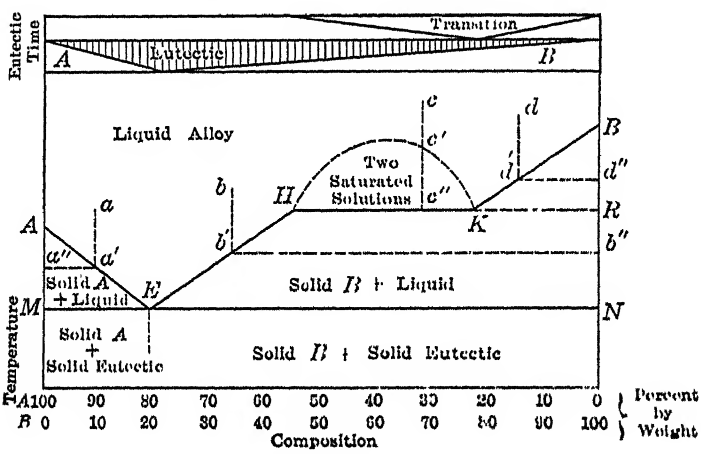


FIG. 26.—Equilibrium diagram, no chemical compounds, liquid metals mutually partly soluble, solid metals insoluble.

given in Fig. 26 represents such a case in which the branch $B\ E$ of the V-shaped curve is intersected by the liquid solubility curve in points H and K; the branch $A\ E$ is identical with that of Fig. 14. In the latter, the metal a on cooling to a' will liberate crystals of pure A. As cooling and crystallization continue, the mother-metal changes in composition along line $a'\ E$ until at the eutectic point E the eutectic of A and B is formed and crystallizes. The behavior of metal b with fall of temperature is identical with that of metal a, excepting that pure

crystals of B separate instead of A. The case is different with metal c, as its composition lies within the range of limited solubility. At point c, the alloy is an unsaturated solution of the two components. As the temperature falls and reaches c' on the saturation-point curve, the single solution breaks up into two conjugate solutions, the concentrations of which are given by the distances from the vertical $c'\ c''$ to the dotted curve. When the temperature has fallen to c'', the two saturated solutions have the composition H and K respectively, and their weights are to each other as $c''\ K : c''\ H$; H and K form the transition points.

At the temperature c'' begins the deposition of pure B from K-solution; this deposition would cause a supersaturation in A of K-solution if the excess-A were not removed; this removal is brought about by the excess-A joining with enough B to attain the concentration H. In this manner the K-solution decreases in quantity with deposition of pure B while the H-solution increases, and the alloy as a whole becomes richer in A. If metal B is lighter than A, the liquid solution K will float upon H. The heat evolved by the freezing of B and the reactions between the solutions cause the temperature to remain constant until the K-solution has disappeared and the remaining liquid alloy has attained the composition H, a saturated solution of B in A. Metal B now continues to separate along line $H\ E$ as does metal A along $A\ E$.

The alloy d is an unsaturated solution of A in B; as it cools to d' freezing begins and pure metal B is deposited, the mother-metal is enriched in A and its freezing-point lowered until at the transition-point K it has become saturated with metal A for that temperature and been converted into K-solution; as freezing continues, a small quantity of H-solution appears; this increases while K-solution decreases until it disappears altogether. The freezing of H-solution is the same as above.

The diagram shows the liquidus $A\ E\ H\ K\ B$, the solidus $A\ M\ E\ N\ B$, and the vertical dropped from the eutectic point E; these divide the freezing plane into five fields, all of which are named.

Above the diagram are shown the relative amounts of eutectic and excess metals for all compositions, also the halts in temperature caused by the readjustments of the two liquid solutions.

The general structure of the solid alloys is the same as that with the alloys treated in §27. Alloys with compositions lying between the transition-points H and K, however, are likely to be non-homogeneous. With metal B lighter than A, the crystals of B liberated at K will rise, while the heavier liquid will sink, and the two may become entangled. In the solid alloy the upper part of a bar will contain many B crystals with the eutectic; they will decrease toward the bottom, of which the lowest part will be eutectic alone. Examples are found in the series Bi–Zn,[1] Cu–Pb[2] and others.

[1] Heycock-Neville, *J. Chem. Soc.*, 1897, LXXI, 390.
Spring-Romanoff, *Zt. anorg. Chem.*, 1897, XIII, 29.

[2] Heycock-Neville, *Phil. Trans.*, *A*, 1897, CLXXXIX, 25.
Friedrich-Leroux, *Metallurgie*, 1907, IV, 293.

32. **Binary Alloys with Components, not Forming Chemical Compounds, and Partly Soluble in the Liquid and Solid States.**—The equilibrium diagram for alloys of this class in which the right-hand branch of the V-shaped curve is intersected by the solubility curve is shown in Fig. 27. The differences between Figs. 27 and 26 are the same as those between Figs. 21 and 14, hence no detailed discussion of the diagram is necessary. The liquidus is *A E H K B*, the solidus *A M E N R B*, the curve of solubility for the liquid metals *H P K*; *M S* and *N T* are branches of the solubility curve for the solid metals. The constitutions of alloys in different fields are given in the figure.

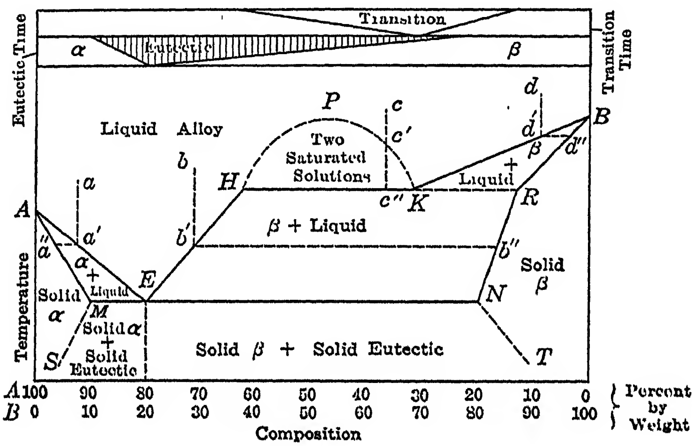


FIG. 27.—Equilibrium diagram, no chemical compounds, liquid and solid metals partly soluble.

33. **Binary Alloys with Components, not Forming Chemical Compounds, and Insoluble in the Liquid and Solid States.**—If alloys are solidified solutions, there are no alloys of this class; there can exist only mechanical mixtures resembling emulsions. Industrial mixtures of this kind have been prepared by Friedrich,[1] Guertler,[2] and by Hannover.[3]

34. **Binary Alloys with Components, Forming Chemical Compounds, and Soluble in the Liquid, Insoluble in the Solid State.**—A list of binary intermetallic compounds well established up to 1909 has been published by Desch.[4] The characteristics of the intermetallic compound have been given on page 35. If the compound is of such a nature that it can be brought to its melting-point without being decomposed, or, which is the same thing, that it has a definite melting-point, its behavior in freezing is the same as that of a pure metal. Thus its freezing-point will be lowered by the addition of either of its components

[1] *Metallurgie*, 1910, VII, 98.
[2] *Op. cit.*, 1910, VII, 264.
[3] *Rev. Mét.*, 1912, X, 509.
[4] *J. Inst. Met.*, 1909, I, 238.

and the compound in the freezing-point curve will be shown by a maximum, the location of which agrees with simple atomic proportions of the metals.

As the compound is decomposed upon melting, the peak of the maximum will not be sharp, but rounded. This is shown in point C of the equilibrium diagram, Fig. 28, which represents the compound $AxBy$ consisting of the components A and B, and containing $A\ O$ per cent. of metal A with $O\ B$ per cent. of metal B. The diagram may be considered as being a combination of two diagrams similar to those of Fig. 14, with the exception that the resulting sharp point representing the maximum is rounded off. The alloy a on freezing at a' deposits crystals a'' which have the same composition as metal A; the freezing-point of the mother-metal enriched in B is lowered along line $a'\ E_1$ until the eutectic point E_1 is reached, when the eutectic freezes which is composed of lam-

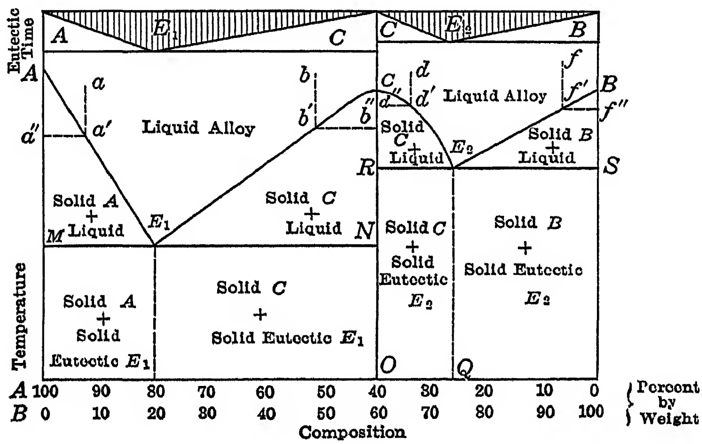


FIG. 28.—Equilibrium diagram, chemical compounds, metals and compounds, completely soluble in the liquid, insoluble in the solid state.

inæ of metal A and compound C. From the alloy b freezing at b' there are split off crystals b'' which have the composition of compound C. With the falling of temperature, the freezing-point of the liquid metal is lowered until point E_1 is reached where the eutectic $A+C$ solidifies. The alloy of the composition O, representing the intermetallic compound $AxBy$ will solidify without change at the point C. The alloy d freezing at d' will deposit crystals d'' having the composition of compound C; the eutectic alloy E_2 will be composed of compound C and metal B; the alloy f cooled to f' will deposit crystals f'' identical with metal B; the solid alloys between B and E_2 will be composed of B and eutectic E_2. In the diagram the liquidus $AE_1\ CE_2\ B$, the solidus $A\ M\ N\ R\ S$, and the verticals $E_1\ P$ and E_2 Q divide the plane into ten fields with the contents inscribed.

It will be noted that metals A and B do not occur together in a solid alloy, therefore there are present two series of alloys of different properties. If there were present two compounds instead of one, there would be three eutectics;

with three compounds, four eutectics. The largest series of compounds studied so far is probably that of Na–Pb,[1] with four maxima shown in Fig. 30.

In the upper part of Fig. 28 are given the eutectic arrests of E_1 and E_2. It will be seen that, diminishing toward either side, the shading disappears at the limits of the diagram and at the vertical *CO*, the composition vertical of the compound.

There exist many alloys in which the chemical compound is a transition product, *i.e.*, it breaks up below its melting-point. Such a case is represented in Fig. 29, the compound being stable only below the line *R S*. The alloy of the composition *O* in cooling from the liquid state will begin to solidify when the line *B R* is reached; the compound *c*, however, cannot form; instead, crystals *B*

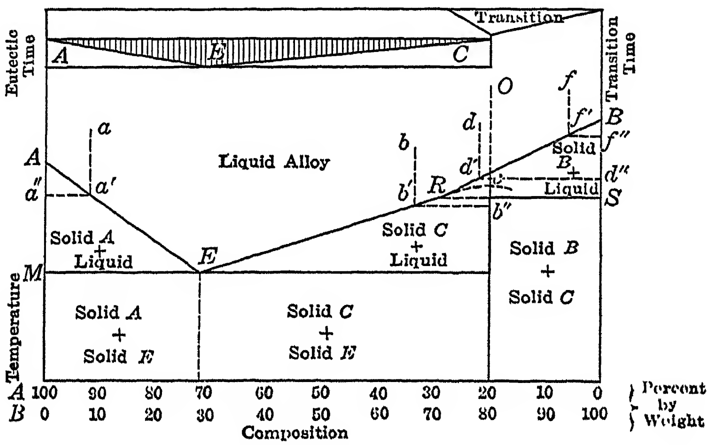


FIG. 29.—Equilibrium diagram, chemical compounds, metals and compounds, completely soluble in the liquid, insoluble in the solid state; the compound is a transition product.

begin to separate from the liquid metal accompanied by a lowering of temperature, until its composition has been changed to *R*, which is also the transition-point at the temperature *R S*. The original composition of the alloy is now restored to that of the compound *c* by a reaction between the crystals *B* and the excess of *A* in the liquid alloy. The restored compound now can solidify; the temperature will not fall, it may even rise, as indicated by the dotted curve *R c*, owing to the reaction between crystals *B* and the still-liquid alloy.

The molten alloy *f*, upon cooling, will begin to solidify at *f'* with the separation of crystals *f''* (= *B*). Its temperature and composition will change along line *f' R*, and at the transition point *R* the solidification will be complete with the reactions just described taking place. As the liquid alloy *f* contains more metal *B* than is necessary to form compound *c*, the solid alloy will consist of metal *B* surrounded by compound *c*.

With alloy *d* cooling to *d'*, crystals of composition *d''* will separate; with a falling temperature freezing will continue as before, but on account of the presence of more metal *A* than is required to form compound *c*, freezing will not stop

[1] Mathewson, *Zt. anorg. Chem.*, 1906, L, 171.

at the transition-point R, but will continue along line $R\ E$ with the separation of compound c until the temperature and composition have reached the eutectic point along $b'\ E$, when freezing is completed by the separation of the eutectic, consisting of metal A and compound c. The liquid alloy b begins to freeze at b' accompanied by the separation of the excess constituent b'' which has the composition of compound c; freezing then continues in the same manner as with alloy d after it has passed point R. The freezing of melted alloy a needs no further comment.

The top of the diagram shows in $E\ A\ C$ the eutectic times and the amounts of eutectic and excess material in the eutectiferous series of alloys. It shows

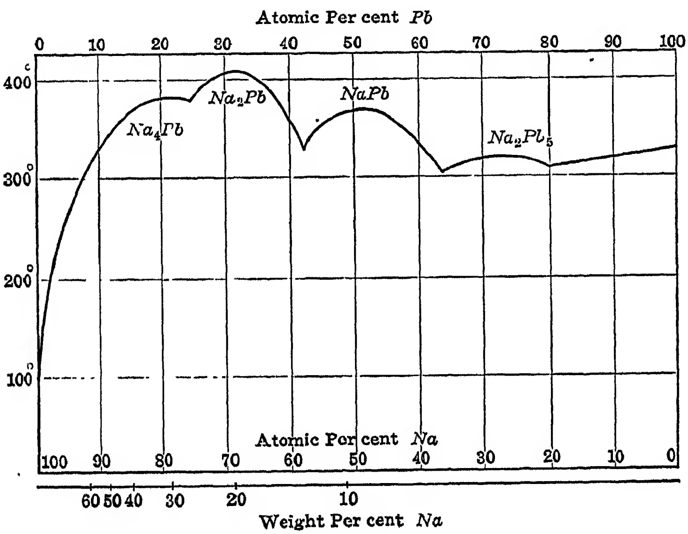


FIG. 30.—Equilibrium diagram of sodium-lead alloys.

in the triangle marked *Transition* the arrest-curve plotted from the retardations in the cooling curves which were caused by the reactions between crystals B and the still-liquid alloy. The eutectic times due to the solidification of eutectic become smaller on either side of the eutectic composition. They disappear on the left when the alloy has changed into pure metal A, and on the right when the compound c has been reached. The longest duration of the transition arrests, due to the greatest development of heat caused by the reaction between metal B and liquid alloy, corresponds with the composition of compound c, and coincides with the vanishing point of the eutectic arrests.

The Cu–As series of alloys[1] forms a good example.

35. Binary Alloys, with Components Forming Chemical Compounds, and Completely Soluble in the Liquid, Partly Soluble in the Solid State.—The equilibrium diagram of such an alloy is shown in Fig. 31. The relation between

[1] Friedrich, *Metallurgie*, 1905, II, 477.
Bengough, *J. Inst. Met.*, 1910, III, 34.

this and Fig. 28 is the same as that between Figs. 22 and 14, *i.e.*, instead of pure metals A and B and pure compound C crystallizing, there separate solid solutions α, β, γ and δ. Concerning the chemical compound, the case is the same as that discussed in Fig. 27, in that it has a definite melting-point. An analysis of the diagram is not necessary.

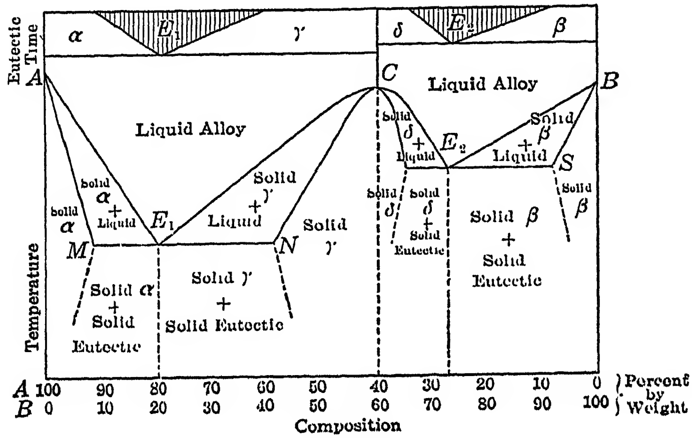


FIG. 31.—Equilibrium diagram, chemical compounds, metals and compounds completely soluble in the liquid, partly soluble in the solid state.

36. Binary Alloys, with Components Forming Chemical Compounds, and Completely Soluble in the Liquid and Solid States.—Fig. 32 is the constitutional diagram of a case in which the chemical compound has a definite melting-point below the melting-points of the pure metals A and B. The diagram resembles very much Fig. 22, treated in §20, which referred to conditions in

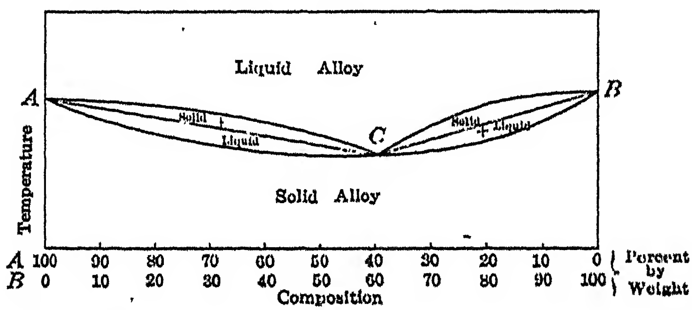


FIG. 32.—Equilibrium diagram, chemical compound, metal and compound completely soluble in the liquid and in the solid states.

which no chemical compound was formed. There the apex C referred to a point on a continuous curve. Here it denotes the intersection of two curves. The alloy-series Mg-Cd[1] forming the compound MgCd forms a good example.

[1] Grube, *Zt. anorg. Chem.*, 1906, XLIX, 72.

37. Binary Alloys, with Components Forming Chemical Compounds, and Partly Soluble in the Liquid, Insoluble in the Solid State.—The equilibrium diagram, Fig. 33, may be considered as being made up of two diagrams, like Fig. 25, placed one beside the other with the descending branch of the second

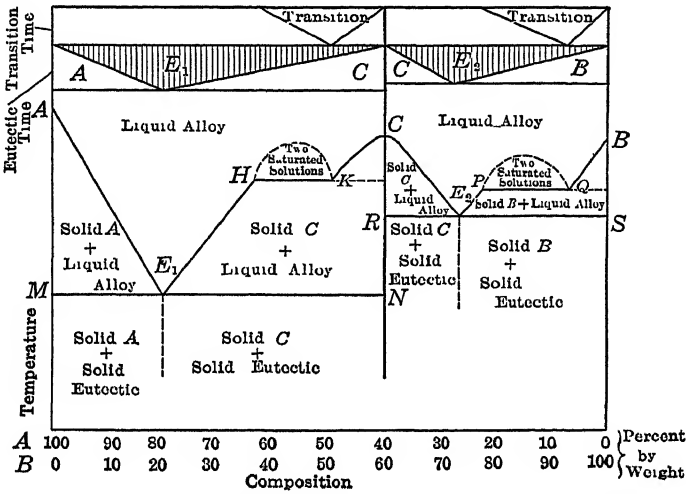


FIG. 33.—Equilibrium diagram, chemical compound, metal and compound partly soluble in the liquid, insoluble in the solid state.

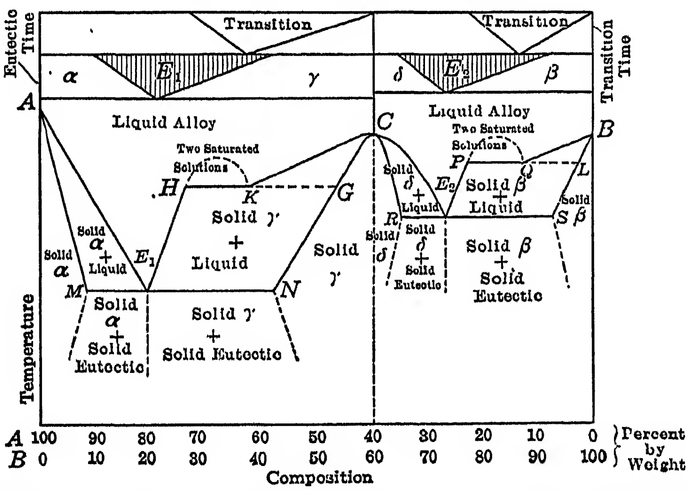


FIG. 34.—Equilibrium diagram, chemical compounds, metal and compound partly soluble in the liquid and in the solid states.

starting at the point *B*, and with the point *C*, Fig. 33, flattened. The discussion given in §31 holds, if compound *C* is substituted on the left side of the diagram for metal *B* in Fig. 25, and on the right side for metal *A*.

38. Binary Alloys, with Components Forming Chemical Compounds, and Partly Soluble in the Liquid and Solid States.—The details of the equilibrium diagram, Fig. 34, offer nothing new, if the relations of Figs. 34 to 33 are com-

pared with those of Figs. 31 to 28. Instead of having pure A, B and C of Fig. 32, there are found solid solutions α, β, γ, and δ.

39. Binary Alloys, with Components Forming Chemical Compounds, and Insoluble in the Liquid and Solid States.—The statement made on a similar condition in §33 holds good here.

40. Ternary Alloys.[1]—The phenomena exhibited in the freezing of ternary alloys are more complex than in those of binary. Ternary alloys may be classified according to the same points of view as binary alloys, but the subdivisions become almost innumerable and only a few alloys have been investigated.

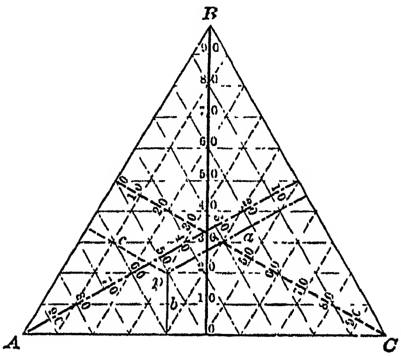


FIG. 35.—Triaxial diagram.

The constitution of the ternary system is represented graphically by means of the equilateral triangle first used by Gibbs[2] and shown in Fig. 35. The corners A, B, C represent the pure metals, and a point on the side of the triangle gives the composition of the binary alloy made up of the two adjoining corner-metals. The composition of an alloy inside the triangle, represented, *e.g.*, by a point p, is ascertained by measuring the perpendicular distances to the sides; their sum is equal to the height of the triangle, and if this is divided into 100 parts, the lengths of the perpendiculars give the percentage composition; lines drawn at right angles to the normals permit reading it off. Thus, the alloy represented by point p has the composition: 50 per cent. A, 20 per cent. B, and 30 per cent. C. If in addition the freezing-points are to be represented, this can be done by erecting verticals on the basal triangle from the points showing the compositions and joining their upper ends; this will constitute a *freezing-point surface* corresponding to the freezing-point curve of binary alloys. This method of representing ternary alloys was first used by Thurston,[3] who also constructed models of three dimensions to represent the physical properties of Cu-Zn alloys.[4] The ternary series Pb-Sn-Bi, studied by Charpy,[5] may serve as an example

[1] Alder-Wright-Thompson, *Proc. Roy. Soc.*, 1888-89, XLV, 461; 1890, XLVIII, 25; 1890-91, XLIX, 156.

Wright, *op. cit.*, 1892, L, 372; 1892-93, LII, 11, 530; 1894, LV, 130.

J. phys. Chem., 1897, I, 403; 1899, III, 217 (Bancroft); 1902, VI, 287 (Browne); 1904, VIII, 257 (Greer).

Zt. phys. Chem., 1897, XXIV, 441 (Kriloff); 1904, L, 169; 1905, LI, 547, LII, 513 (Scheinemakers); 1905, LIII, 449 (Roozeboom-Aten); 1907, LX, 399 (Jänecke); 1908, LX, 357 (Ruer); 507 (Sahmen-Vegesack).

[2] *Conn. Academy*, 1876, III, 176.

[3] Howe, H. M., *Tr. A. I. M. E.*, 1898, XXVIII, 346.

[4] Thurston, R. H., "A Treatise on Brasses, Bronzes, and Other Alloys," Wiley, New York. 1900, p. 425.

[5] *Compt. Rend.*, 1898, CXXVI, 1569, 1645; *Bull. soc. d'encour.*, 1898, III, 670; "Contributions à l'étude des Alliages," 1901, p. 203; *Metallographist*, 1899, II, 9.

of alloys which form neither chemical compounds nor solid solutions.[1] Fig. 36, a projection of the freezing-point surface on the basal plane of the prism, is made up of three zones which meet along the lines *Ee*, *E'e* and *E''e*, and form valleys showing the separation of the binary eutectics Pb–Bi, Pb–Sn and Sn—Bi, which run downward to the eutectic point *e*.

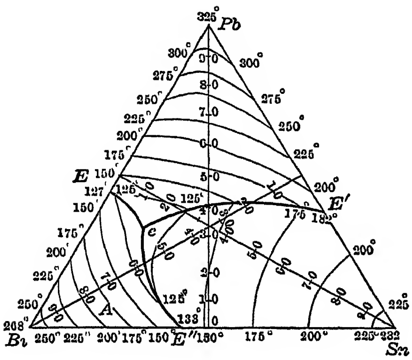


FIG. 36.

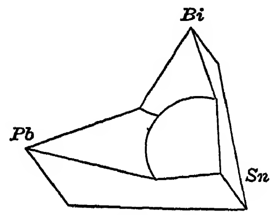


FIG. 36A.

FIGS. 36 and 36A.—Equilibrium diagram and space model of the alloy series, lead-tin-bismuth.

The space-model is shown in Fig. 36*A* as a triangular prism with vertical sides; the figure is slightly tilted to show the V-shaped curves.

Alloys in the zone *Pb E e E'* will deposit Pb when they begin to freeze, alloys in the zones *Bi E e E''* and *Sn E'e E''* will deposit Bi and Sn respectively. Alloys situated on the binary eutectic line *E'e* on freezing will deposit both Pb and Sn, those on *E e* both Pb and Bi; those on *E''e* both Bi and Sn. At *e*, the ternary eutectic with Pb 32 per cent., Sn 15.5 per cent., and Bi 52.5 per cent., will freeze at 96° C. Its composition resembles closely the Newton fusible alloy made up of 4 parts Pb, 3 Sn and 8 Bi. The light curves are isothermals, 25° C. apart. The different phases in the process of freezing are made clear by following an alloy as it passes from the liquid to the solid state. Its cooling curve, Fig. 37, will show three retardations; the first, *A*, is due to the solidification of pure metal; the second, *B*, to the separation of the binary eutectic; the third, *C*, to the freezing of the ternary eutectic which is completed at *c*.

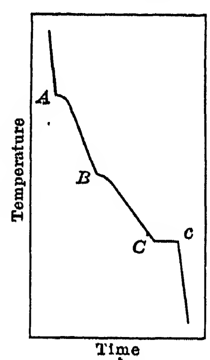


FIG. 37.—Cooling curve of a ternary alloy.

Let the molten alloy *A*, Fig. 36, be at 250° C. When it has cooled to 175°, it will begin to freeze; Bi will crystallize, the mother-metal will become enriched in Pb and Sn, and its freezing-point lowered. The composition changes along a line drawn from Bi through *A* to the intersection of the eutectic line *E''e* at 125°. Now the eutectic of Bi and Sn separates and

[1] Recent research (*Min. Ind.*, 1910, XIX, 437) has shown that Sn and Pb form solid solutions to a small extent, but this does not affect the freezing-point surface.

causes the composition and temperature of the molten alloy to travel along line $E''e$ to the point e, where the ternary eutectic solidifies and splits into its components. The photomicrograph, Fig. 38, shows light areas of Bi enclosed by the darker binary eutectic Bi+Sn, and the dark ternary eutectic occupying the rest of the field.

Other examples are Cu–Ag–Pb,[1] Cu–Ag–Au and Fe–Co–Ni,[2] Sn–Cd–Pb and Sn–Cd–Bi,[3] Cu–Ni–Zn,[4] Pb–Sn–Sb,[5] Pb–Cd–Bi,[6] Pb–Ag–Sn,[7] Fe–C and other metals, such as P,[8] Sb and Sn[9] and others.[10]

FIG. 38.—Photomicrograph of a lead-tin-bismuth ternary eutectic. (Charpy).

41. **Quaternary Alloys.**—The constitution of these alloys has not been studied to any extent. The graphical representation of each alloy would require a tetrahedron. The fusible alloys of Wood (Sn 2 Bi 7–8, Cd 1–2 parts) and Lipowitz (Sn 4, Pb 8, Bi 15, Cd 3 parts) are probably quaternary alloys. Alloy-steels, such as Ni–Cr, W–Cr, Ni–Va, Cr–Va, etc., form a division of increasing importance.

42. **Transformations in Alloys and Metals.**—When alloys are heated to temperatures below their melting-points they often undergo changes (*trans-*

[1] Friedrich, *Metallurgie*, 1907, IV, 293.
[2] Jänecke, *op. cit.*, 1910, VII, 250.
[3] Stoffel, *Zt. anorg. Chem.*, 1907, LIII, 137.
[4] Tafel, *Metallurgie*, 1908, V, 343, 375, 413.
[5] Loebe, *Metallurgie*, 1911, VIII, 7, 33.
Campbell-Elder, *School Min. Quart.*, 1911, XXXII, 244.
[6] Barlow, *Zt. anorg. Chem.*, 1911, LXX, 178; *J. Am. Chem. Soc.*, 1910, XXXII, 1390.
[7] Parravano, *Internat. Zt. Metallographie*, 1911, I, 89.
[8] Goerens-Dobbelstein, *Metallurgie*, 1908, V, 501.
[9] Goerens-Ellinger, *op. cit.*, 1910, VII, 72.
[10] Goerens, *Zt. Electrochem.*, 1909, XV, 617.

formations) which may be only physical, as seen in the equilibrium diagrams given. Frequently these changes are also chemical. Thus, if the Newton fusible alloy,[1] mentioned above, is heated, chilled in water, and then quickly removed from the water, its temperature is found to rise sufficiently to burn the hand. Its fracture just after chilling is porcelanic and the luster resinous to vitreous; after the rise of temperature caused by the heat evolved from molecular changes of constituents, it is granular and dull. If in the eutectiferous Fe–C series of alloys a steel with 0.89 per cent. C at say 1000° C., in which state the C and Fe form a solid solution called austenite, is cooled down slowly to 690° C., a transformation involving a chemical change takes place; the solid solution is transformed into a eutectoid called pearlite which consists of 88 per cent. Fe and 12 per cent. Fe_3C. The term *Eutectoid* has been given by Howe to the product of the lowest transformation in a solid alloy, in contradistinction to eutectic, which is the product that solidifies last upon cooling a molten eutectiferous alloy. The form of the curve is V-shaped and shows progressive changes in the solid solution just as is the case with the freezing-point curve in the liquid solution. Eutectoid structures resemble eutectic structures. The prefixes *hypo-* and *hyper-* are used in a similar sense as with eutectic, giving *hypo-eutectoid* and *hyper-eutectoid* alloys.[2] Such a transformation takes place rapidly and may evolve sufficient heat to cause the temperature to rise suddenly, a phenomenon known as *recalescence*. Fig. 39 marks the cooling curve, by Osmond,[3] of a hard steel with 1.25 per cent. C, 0.19 Si, 0.02 S, 0.02 P, and 0.10 Mn. While cooling from 1,065° C. a slight accidental retardation is noticed at 860° C., the recalescence point lies at 674° C.; the transformation begins at 720°, closes at 645°, and lasts 64 seconds. The observations necessary for plotting such a curve are to note the time in seconds it takes the deflection from the galvanometer of a thermoelectric pyrometer to pass over equal increments (say 5 mm.) of the scale.

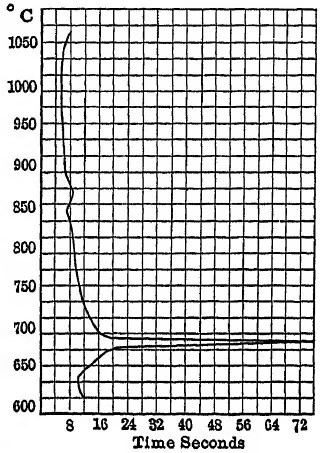


FIG. 39.—Cooling curve of steel with 1.25 per cent. C.

In heating a steel with 0.89 per cent. C. above the critical temperature of 690° C., the eutectoid is again transformed into a solid solution. Such transformations are called *reversible;* if the change upon cooling cannot be restored by heating, the transformation is *irreversible*. The critical temperature at which a transformation is noted in cooling is not the same as that in heating, as the

[1] Person, *Compt. rend.*, 1847, XXV, 444.

[2] Discussion of nomenclature, *Eng. Min. J.*, 1903, LXXV, 144 (Howe), 323 (Brough), 360 (Stansfield), 361 (Arnold), 435 (Sauveur), 472 (Osmond), 588 (Howe), 588 (Grohmann).

[3] "Transformations du fer et carbone dans les fers, les aciers et les fontes blanches," Baudoin & Co., Paris, 1888, pp. 24 and 27, Plate 2.

transformation is not instantaneous because diffusion in solid metals takes considerable time. In Fig. 40, A_c (*arrêt chauffant*, arrest in heating) lies higher than A_r (*arrêt refroidissant*, arrest in cooling). Between the two points lies the *critical range*. The full-drawn lines show the temperatures at which the alloy is below the critical temperatures, the dotted lines at which it is above them.

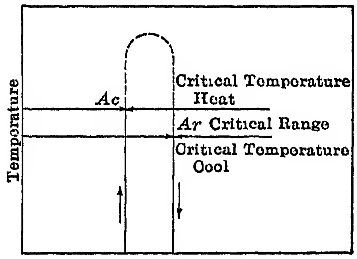


FIG. 40.—Cooling and heating arrests.

Transformations in steels, more highly carbonized iron, brasses, bronzes, etc., are of far-reaching importance in their effects upon the physical properties. Transformations in metals and intermetallic compounds are of frequent occurrence; these substances are said to be *polymorphic*. Iron is known to exist in the three allotropic forms of γ-, β- and α-iron. The cooling and heating curves of iron are shown in Fig. 41. The first jog in the solid-drawn cooling curve, due to the solidification of iron, occurs at 1,505° C. (the figure generally accepted at present is 1,520°), the iron is present as γ-iron, and continues in that form until the temperature has fallen to Ar_3 at 880° C., when it changes into β-iron, and this is transformed at Ar_2 with 780° into α-iron. In the dotted heating curve the corresponding arrests Ac_2 and Ac_3 lie from 20 to 30° higher.

In a eutectiferous series of alloys the transformation of a polymorphic metal or intermetallic compound takes place at the same temperature whether it is present as a primary crystal or as a constituent of the eutectic, hence the change is represented in the equilibrium diagram by a horizontal line which extends as far as the eutectoid. With solid solutions the transformation temperature varies with the composition; in the graphical representation there will be required two lines which will correspond closely with the liquidus and solidus.

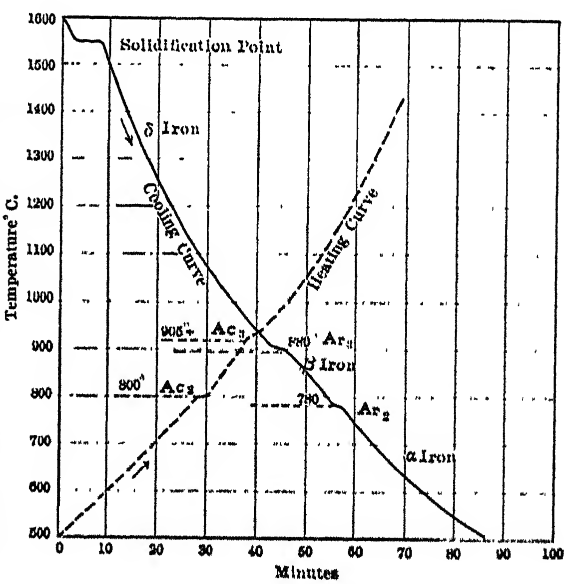


FIG. 41.—Cooling and heating curve of pure iron.

43. Preparation of Alloys.—Alloys are usually prepared by fusion;[1] amalgams, of course, form an exception to this general statement. Metals in a finely divided state have

[1] Desch, "Some Defects in Alloys," *J. Inst. Met.*, 1910, IV, 238.

been united by Spring[1] by subjecting them to strong pressures. In the cementation process steel is produced by the diffusion, at an elevated temperature, of C into carbonless iron. Again, Cu, *e.g.*, may be exposed to the vapors of Zn, and brass formed without fusing. Further, from an aqueous solution metals may be deposited together as an alloy;[2] thus Cu and Zn are deposited together as brass.[3]

In the fusion method, the metal of a higher melting-point is usually first liquefied in a crucible, kettle or furnace, and then the more readily fusible metal charged, ordinarily in the solid state after having been warmed, rarely in the liquid state. If two metals are to be alloyed, and one is volatile at the melting-point of the other (*e.g.*, Zn and Cu), they are charged together in order that the readily fusible metal may dissolve the more refractory. In order to prevent oxidation during fusion, the charge is covered with charcoal or with a flux such as borax glass. Sometimes metals or alloys that have a very much stronger affinity for O than the charge are added in small amounts during or after fusion, in order that they may combine with the O of a constituent of the alloy that has become superficially oxidized. The oxide of the purifying metal may not, of course, combine with the alloy; a small excess of the metal added also may not injure the properties of the alloy. The purifying metals in common use are Mg, Al, Mn, Si, and P.

Metals that are imperfectly miscible have to be kept molten for some time at a sufficiently elevated temperature and repeatedly stirred if a uniform alloy is to be the result. It is often necessary to remelt an alloy several times before the desired uniformity can be obtained.

In casting an alloy, it is important to avoid liquation, *i.e.*, the segregation of readily fusible constituents from those which fuse with difficulty. Liquation will not occur with metals forming solid solutions, but will be common with eutectiferous mixtures, and will be especially the case if the reciprocal solubility is limited, and the difference in freezing-points of eutectics and constituent metals is large. Liquation is favored by slow cooling; chilling sets up internal strains; hence, in casting one has to steer between the two extremes or to chill and then anneal the alloy. In casting sample bars of Pb–Ag,[4] or of Cu–Ag[5] the alloy will be poured into a cold heavy iron mold, which abstracts heat quickly and, chilling the alloy, prevents segregation.[6] Irrespective of any liquation,

[1] *Ber. deutsch. chem. Ges.*, 1882, XV, 595; *Zt. phys. Chem.*, 1888, II, 532, 536; *Bull. Soc. Chim.*, 1888, XLIX, 215.
Masing, *Zt. anorg. Chem.*, 1909, LXII, 265.
Drewitz, *Verh. Ver. Beförd. Gewer befleiss.*, 1902, LXXXI, 325.
Tammann, *Zt. Electrochem.*, 1909, XV, 44.

[2] Langbein, G., and Brannt, W. H., Electrodeposition of Metals, Baird, Philadelphia, 1905.
McMillan, W. G., Treatise on Electrometallurgy, Lippincott, Philadelphia, 1899.

[3] De Kay Thompson, *Met. Chem. Eng.*, 1912, X, 458.

[4] Hofman, "Metallurgy of Lead," New York, 1899, 350.

[5] Keller, *Tr. A. I. M. E.*, 1897, XXVII, 106.

[6] Roberts Austen-Harbord, "Introduction to the Study of Metallurgy," 1910, p. 76.

there is for each metal or alloy a certain casting temperature and a certain rate of cooling[1] which gives the most satisfactory product, as the crystals will be of the required size and their arrangement of such a sort as to furnish the greatest strength and the highest degree of durability. If the crystals are too large, on account of a high casting temperature, there will be found weak lines along cleavage places; if they are too small there will not be enough interlocking and the product again will be weak.

44. General Properties of Alloys.[2] DENSITY.—The specific gravity of an alloy may be the mean of that of the constituent metals if it is composed of an eutectic mixture; if of a solid solution, or when the components have formed intermetallic compounds, it is higher. Alloys, the density of which is greater than the mean of the constituents, are those of Au–(Zn, Sn, Bi, Sb, Co), Ag–(Zn, Sn, Bi, Sb), Cu–(Zn, Sn, Pd, Bi), Pb–Sb, Pt–Mo, Pd–Bi. A striking example is that of a bronze with 38 per cent. Sn which has a specific gravity of 8.91, while that of Cu is 8.89 and of Sn 7.31. Alloys the density of which is smaller than the calculated mean are Au–(Ag, Te, Pb, Cu, Ir, Ni), Ag–(Cu, Te, Bi, Pb), Sn–(Pb, Pd), Ni–As, Zn–Sb. A good example of the second class is the alloy Al 18 per cent. and Sb 82 per cent. with a specific gravity 4.217 (calculated mean 5.22), the specific gravity of Al being 2.67 and of Sb 6.72.

45. Color.—In § 6 it has been shown that with the exception of Cu and Au all metals are gray or white. By the union of gray and white metals alloys of similar colors are produced. Cu alloyed with other metals gives yellow (Zn – Al), white (Zn, Sn, Ni), violet (Sb), etc., alloys. Au by an admixture of Cu becomes reddish, of Ag greenish-yellow to whitish, of Al[3] white, pink, ruby-colored ($AuAl_2$) and so on. Ledebur arranged the leading metals according to their coloring power in the following order: Sn, Ni, Al, Mn, Fe, Cu, Zn, Pb, Pt, Ag, Au, in which Sn has the greatest decolorizing effect. Thus the yellow of Au will be obliterated by a comparatively small amount of Sn. The production of color effects is a special art.[4]

46. Crystallization.—This is of little industrial importance. In a general way it may be said that if the constituents of an alloy crystallize in one system, the alloys will show forms of the same order; if not, it will generally assume shapes belonging to the system of one of the constituents. Thus Cu–Zn, Au–Ag, Pb–Ag, Hg–Ag form isometric; Au–Sn, Fe–Sn tetragonal; Sb–Zn, Fe–Mn orthorhombic; Cu–Sn hexagonal crystals.

[1] Primrose, *J. Inst. Met.*, 1910, IV, 248.

[2] Calvert-Johnson, *Phil. Mag.*, 1859, XVIII, 354.
Matthiesen, *Proc. Roy. Soc.*, 1859–60, X, 12, *Phil. Trans. Roy. Soc*, A, 1860, 177.
Riche, *Compt. Rend.*, 1862, LV, 143; 1868, LXVII, 1138; 1869, LXIX, 343; *Ann. de Chim.*, 1873, XXX, 35.
Macy, *Zt. phys. Chem.*, 1899, XXIX, 119; 1901, XXXVIII, 289, 292; 1904, L, 200.
Retgers, *op. cit.*, 1889, III, 497.
Hannover, *Rev. Mét.*, 1912, IX, 641.

[3] Roberts-Austen, *Proc. Roy. Soc.*, 1891, XLIX, 347.

[4] Hiorns, A. H., "Metal Coloring and Bronzing," Macmillan, London, 1892.
Buchner, G., "Die Metallfärbung," Krayn, Berlin, 1906.
Hartmann, F., "Das Färben der Metalle," Hartleben, Vienna, 1912.

47. Structure.—The structure of an alloy is governed not only by its chemical composition, but also its rate of cooling and any mechanical treatment it may have been subjected to. It is studied in the same manner as is that of metals, by fracture and microscopical examination of polished surfaces.

48. Hardness.—A metal when pure is always softer than when it is alloyed with another, hence for mechanical purposes alloys are in more general use. The hardening effects the metals may have upon one another vary greatly. Sb has a strong hardening power; Cu is hardened by means of Zn, Sn, Al, P; Pb by Zn, Sn, Sb; Zn by Fe; Au by Ag and Cu; Fe by C, Cr, Wo, Mo, Vd, etc.

Kurnakoff and Schemtschuschny[1] have studied the relations existing between the hardness and constitution of alloys. Metals are: (1) Mutually insoluble and form a eutectic conglomerate; the hardness is a linear function of the com-

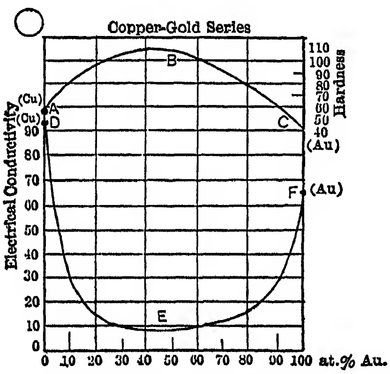


Fig. 42.—Hardness and electric conductivity curves of solid solution alloys.

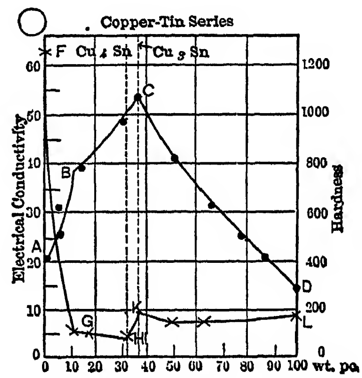


Fig. 43.—Hardness and electric conductivity curves of alloys forming chemical compounds.

position and the curve a straight line; (2) mutually soluble and form a series of solid solutions; the hardness curve is a continuous line having a maximum near the point representing equal weights of the two metals. The upper part of Fig. 42, representing the Cu–Au series of alloys, shows this in the curve *A B C*; (3) mutually insoluble and form a chemical compound; the hardness curve consists of two straight lines intersecting at the composition point. This is shown in the upper part of Fig. 43, representing the Cu–Sn series of alloys with the chemical compound Cu_3Sn well established; the compound Cu_4Sn is still doubtful. The relation between hardness and electric conductivity shown in Figs. 42 and 43 is discussed in §54.

49. Strength.—This is governed by the thermal and mechanical treatment an alloy has received. In general it may be stated: (1) That the tensile strength of a metal is increased up to a certain point by the addition of a second metal and then decreased unless the second metal is stronger than the first; (2) that

[1] *Zt. anorg. Chem.*, 1908, LX, 1.

by the addition of a third metal to a binary alloy, the tensile strength of the latter is often increased; (3) that the elastic limit of an alloy grows with the ultimate strength, but does this faster and often continues to grow when the ultimate strength has reached its maximum; (4) that the effect of high temperature[1] is irregular; (5) that the relations that may exist between tensile strength or elastic limit and constitution have not yet been defined.

50. Plasticity.—As a rule, pure metals are more malleable and ductile than their alloys. Thus, *e.g.*, the malleability and ductility of Cu are reduced nearly to zero by the presence of 6 per cent. Sn. Au and Ag are rendered less plastic by the addition of Cu; 0.05 per cent. Bi makes Pb, Sn, Cu, etc., brittle.

51. Expansion and Contraction.[2]—The expansion by heat of alloys as a rule differs greatly from that found by calculation. In some cases it is larger than the one calculated; in others smaller; in a few only (Sn–Pb, Cd–Pb, Zn–Sn, Pb–Bi) is it the same. Wüst[3] states that alloys mutually insoluble in the solid state show a shrinkage higher than the component metals, that an alloy of eutectic composition has the lowest shrinkage, and that alloys forming solid solutions show a higher shrinkage than the component metals. Some of his experimental results are given in Tables 16 and 17.

TABLE 16. ANALYSES OF LEAD, TIN AND ZINC ALLOYS (EUTECTIFEROUS)

Number	Pb	Sn	Zn	Sb	Cu
9	80.82	18.27			
10	29.10	70.01			
11	18.39	80.90			
12	80.61			19.20	
13	85.20			14.68	
14		50.83	49.04		
15		85.40	14.52		
16		95.15	4.81		
33	1.66	14.48	79.44		4.35
32	0.94	45.84	51.22		1.90

[1] Martens, *Mitt. Kgl. Tech. Versuchsanstalt.*, 1890, VIII, 59.
Le Chatelier, *Congrès des Méthodes d'Essais*, Paris, 1901, Vol. II, p. 51.
Webb, *Proc. Engl. Inst. Civ. Eng.*, 1901–02, CL, 87.
Bengough, *J. Inst. Met.*, 1912, VII, 123.

[2] Calvert, *Rep. Brit. Assoc.*, 1858, II, 46.
Calvert-Johnson-Lowe, *Phil. Mag.*, 1860, XX, 230; *Chem. News*, 1861, III, 315, 357, 371.
Le Chatelier, "Contributions à l' Étude des Alliages," Paris, 1901, p. 386.

[3] *Metallurgie*, 1909, VI, 769; *Iron Age*, 1910, LXXXV, 790.

TABLE 17.—RESULTS OF SHRINKAGE TESTS

Number	Casting temperature, deg. C.	First arrest, deg. C.	Second arrest, (eutectic) deg. C.	Total shrinkage, per cent.
9	650	251	....	0.56
10	550	....	174	0.44
11	550	....	180	0 50
12	650	....	239	0 54
13	500	....	232	0.56
14	550	....	200	0.50
15	500	....	195	0.46
16	500	....	190	0 49
33	500	379	374	1 02
32	550	340	334	0 73

In all the cases the total shrinkage is less than that of the component metals.

Alloys of copper-zinc, copper-tin and copper-nickel-zinc act differently from the above, the alloys expanding after the beginning of solidification, the expansion rising to a maximum, and then the alloys shrink.

TABLE 18.—ANALYSES OF COPPER-ZINC ALLOYS (SOLID SOLUTIONS)

Number	Cu	Sn	Zn	Ni	
17	83.45	....	16.24	....	Tombac yellow
18	66.60	....	32.90	....	Tombac yellow
19	63.10	....	36.24	....	Brass
20	63.93	....	35.25	....	Brass
21	94.70	5.08	....	....	Bronze
22	89.65	10.22	....	....	Bronze
23	80.66	19.08	....	....	
24	61.57	....	22.16	16.10	German silver
25	56.20	....	23.36	20.40	German silver
26	51.40	....	22.30	26.22	German silver
27	46.10	....	18.00	35.80	German silver

These alloys show a greater shrinkage than do the component metals.

With white metals the shrinkage takes place in two phases, the transition from one to the other being abrupt. No expansion is noticed while freezing.

TABLE 19.—RESULTS OF SHRINKAGE TESTS

Number	Temperature, deg. C.	First arrest, deg. C.	Second arrest deg. C.	Beginning expansion, deg. C.	Per cent. expansion	Beginning shrinkage deg. C.	Per cent. shrinkage
17	1,000 to 1,050	993	....	1,000	0.3	973	2.17
18	950	902	....	904	0.03	870	1.97
19	950	880	....	935	0.03	879	1.90
20	900 to 1,000	881	....	990	0.033	879	1.90
21	1,050	1,035	....	1,032	0.085	786	1.66
22	995 to 1,150	996	....	980	0.122	706	1.44
23	900	....	760	835	0.01	752	1.52
24	1,100	1,020	955	1,068	0.045	917	2.025
25	1,065 to 1,090	1,060	1,025	1,049	0.039	924	2.05
26	1,100 to 1,150	1,087	890	1,080	0.027	940	2.03
27	1,170 to 1,200	1,085	....	1,090	0.032	1,010	1.935

TABLE 20.—ANALYSES OF ALLOYS (WHITE METALS)

Number	Sn	Pb	Sb	Cu	
34	....	78.89	12.5	8.45	....
35	19.80	58.84	21.4	....	....
36	85.42	....	9.45	5.10	White metal
37	90.20	....	8.10	1.85	Britannia metal
38	70.83	0.21	15.1	4.04	Britannia metal

TABLE 21.—RESULTS OF SHRINKAGE TESTS

Number	Casting temperature, deg. C.	First arrest, deg. C.	Second arrest, deg. C.	Beginning of shrinkage, deg. C.	Per cent. shrinkage
34	600 to 650	250	230	260	0.55
35	600	263	238	247	0.49
36	500 to 550	225	....	225	0.51
37	600 to 700	228	....	226	0.55
38	550 to 600	259	187	228	0.42

In the copper-tin-zinc-lead alloys given below, the last two correspond to commercial gun metal. In these alloys, the cooling curves show four arrests. The shrinkage curves show first an expansion and then a contraction.

TABLE 22.—ANALYSES OF COPPER-TIN-ZINC ALLOYS

Number	Cu	Sn	Zn	Pb
28	87.10	2.68	8.05	2 28
29	81.06	17.50	1.53	....
30	88.75	9.65	1.6	....
31	86.65	9.84	2 0	1.44

TABLE 23.—RESULTS OF SHRINKAGE TESTS

Number	Casting temperature, deg. C.	First arrest, deg. C.	Second arrest, deg. C.	Third arrest, deg. C.	Fourth arrest, deg. C.	Beginning of expansion, deg. C.	Per cent. expansion	Beginning of shrinkage, deg. C.	Per cent. shrinkage
28	1,000	992	895	825	685	973	0 025	840	1.76
29	950 to 1,000	873	775	737	603	854	0.024	756	1.50
30	950 to 1,020	977	990	824	745	955	0 058	726	1.47
31	1,000	965	840	778	748	944	0.075	750	1.47

The shrinkage is higher than that of the component metals.

52. **Specific Heat.**—Regnault[1] found that the specific heat of alloys at temperatures considerably removed from their melting-points (Sb–Sn, Pb–Sb, Bi–Sn) was exactly the mean of the specific heats of the metals which compose them. With alloys, however, which melt at or near 100° C. (Pb–Sn–Bi) the specific heats are much higher than those calculated from the constituents.

53. **Thermal Conductivity.**—Alloys never conduct heat as well as the component metals. The researches of Calvert and Johnson[2] have shown that the thermal conductivity is never higher than that calculated from the conductivities of the component metals, that in a few instances only (Pb–Sn, Bi–Sn) is it the same; and that in the majority of cases it is lower, even lower than that of the component showing the lowest figure.

54. **Electric Conductivity.**[3]—The electric conductivity of an alloy is nearly always lower than that of the constituent metals. Matthiesen has shown that the actual conductivity of an alloy is always smaller than that calculated

[1] *Ann. chim. phys.*, 1841, I, 129–207.

[2] *Phil. Mag.*, 1858, XVI, 381.

[3] Matthiesen, *Phil. Trans. Roy. Soc.*, 1858, CXLVIII, 383; 1860, CL, 161; 1862, CLII, 1; *Proc. Roy. Soc.*, 1859–60, X, 205; 1860–62, XI, 126; *Phil. Mag.*, 1858, XVI, 219; 1860, XX, 63; *Rep. Brit. Assoc.*, 1863, XXXIII, 39.

Le Chatelier, Contributions à l'Étude des Alliages, 1901, 413, 441.

Guertler, *Zt. anorg. Chem.*, 1906, LI, 397; *Metallurgie*, 1907, IV, 115; *Zt. Electrochem.*, 1912, XVIII, 601; *J. Inst. Met.*, 1911, VI, 135.

Broniewski, *Rev. Mét.*, 1911, VIII, 320, includes bibliography.

from the volumes of the constituents; that in a few cases only (Pb–Sn, Zn–Cd with one another) is it the same; and that with all other metals it is less and often less than that of the least conductive constituent. He also showed that conductivity curves belong to one of three types:

The U-shaped type, Fig. 42, with a steep fall at the end and flattening in the intermediate portion, is characteristic for solid-solution alloys. This is to be expected, as the properties of solid solutions vary progressively without interruption from one end of a series to the other. The remarkable degree in which the conductivity of a metal is lowered by the addition of a small quantity of a second metal is explained by Guertler[1] by the fact that the latter being dissolved completely permeates the former.

The straight-line type, Fig. 43, is typical for alloys which are eutectiferous throughout. The reason for this is that the current in passing through the

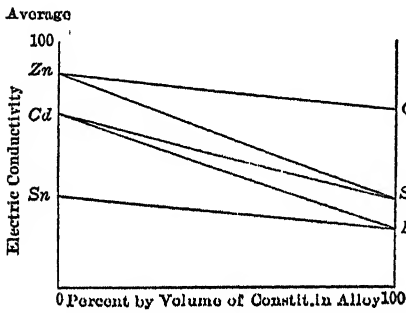


FIG. 44.—Electric-conductivity curves of alloys forming eutectics

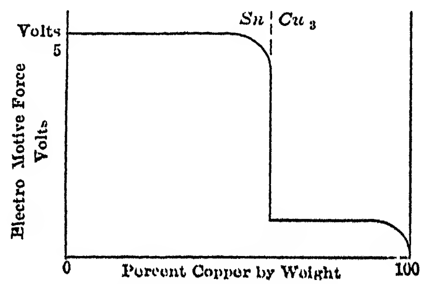


FIG. 45.—Electromotive force of alloys forming chemical compounds.

alloy, crosses what may be considered to be a mechanical mixture of the constituents. If the constituents are partly soluble in one another, forming solid solutions toward the ends and an eutectic toward the middle, the conductivity curve will be a combination of the two preceding types.

The broken-line type, Fig. 45, is produced with alloys forming chemical compounds. The chemical compound must, of course, have a conductivity different from that of the two branches of the freezing-point curve of which it forms the maximum.

The effect of temperature upon electric conductivity has recently been studied by Guertler.[2] The conductivity of eutectic and solid-solution alloys decreases with rising temperature, that of intermetallic compounds increases. The conductivity of molten alloys[3] is similar to that of solid solutions unless a chemical compound is in the alloy.

55. Electromotive Force.[4]—In a eutectiferous alloy, the electromotive

[1] *Zt. anorg. Chem.*, 1906, LI, 397.

[2] *Zt. anorg. Chem.*, 1907, LIV, 58.

[3] Müller, *Metallurgie*, 1910, VII, 730, 735.

Bornemann-Müller-Rauschenplat, *Metallurgie*, 1910, VII, 396; 1912, IX, 473, 505, *Stahl. u. Eisen*, 1910, XXX, 1519.

[4] Touchine, *Rev. Mét.*, 1907, IV, 926.

force observed is that of the more electropositive component metal which goes into solution while the other remains inert. With a solid-solution alloy the curve will be a continuous line differing in direction from that of the component metals. A combination of the two is shown in the Pb–Sn curve.[1] With a chemical compound, Fig. 45, the direction of the curve leading up to the chemical compound will be suddenly changed when the chemical compound has been reached, and will point again, just as suddenly, away from it, when an excess metal begins to make its appearance.

56. Thermo-electric Force.—This power is of interest in the study of transformations occurring in alloys in the solid state. It has been studied especially by Steinmann,[2] Reichardt,[3] Harrison,[4] Belloc[5] and others.

57. Magnetic Permeability.—Of the three magnetic metals, Fe, Ni, Co, only Fe and Ni are of industrial importance. Studies in permeability have to do with the alloys of these two metals. This statement was correct until 1903 when Heussler[6] prepared alloys of Cu, Mn, and Al which are magnetic and have been found to show a behavior similar to that of the ferromagnetic alloys of Fe, Ni and Co. Analyses of some typical alloys are given in Table 24.

TABLE 24.—ANALYSES OF HEUSSLER ALLOYS
(Per cent.)

Si	0.08	0.07	0.02	0.16	0.17	0.05
Pb	0.07	0 07	0.13	2.03	3.14	3.84
Cu	64.49	70.14	75.83	59.43	65.22	73.68
Mn	20.39	18.03	14.66	22.60	19.76	13.73
Al	13 25	10.03	8.64	14.50	11.13	8.33
Fe	1.05	0.99	0.55	1.31	0.67	0.46

The magnetic property appears to be connected with the formation of the intermetallic compounds Cu_3Al and Mn_3Al. However, beside Al, metals like Sn, Sb, Bi, As and B have been found to form magnetic alloys with the isomorphous metals Cu and Mn.

[1] Laurie, *J. Chem. Soc.* (London), 1888, LIII, 104; 1889, LV, 677; 1894, LXV, 1030.
[2] *Compt. Rend.*, 1900, CXXX, 1300.
[3] *Ann. Phys.*, 1901, VI, 832, *Zt. Electrochem.*, 1902, VIII, 131
[4] *Phil. Mag.*, 1902, III, 177.
[5] *Compt. Rend.*, 1902, CXXXIV, 105; *Ann. chim. phys.*, 1903, XXX, 42.
[6] Heussler, *Verh. deutsch. phys. Ges.*, 1903, V, 219.
Stark and Haupt, *ibid.*, 222.
Heussler, *Zt. angew. Chem.*, 1904, 260.
Hadfield, *Rep. Brit. Assoc.*, 1904, 685; *Faraday Soc.*, April 13, 1912.
Flemming-Hadfield, *Proc. Roy. Soc.*, A, 1905, LXXVI, 271; *Electrochem. Met. Ind.*, 1905, III, 345.
Hill, *Phys. Rev.*, 1905, XXI, 335; 1906, XXII, 498.
Gray, *Proc. Roy. Soc.*, A, 1906, LXXVII, 256.
Ross-Gray, *Zt. anorg. Chem.*, 1909, LXIII, 349.
Ross, *J. Inst. Met.*, 1910, IV, 68.
Stevenson, *Bull.* 47, University Ill., 1910, VIII, No. 17.

58. Corrosion.[1]—The resistance of alloys to corrosion by atmospheric and other agencies in their industrial use is of the greatest practical importance. It has been shown in §24 that impure metals are, as a rule, more easily attacked by solvents than pure metals, as the impurity, the metal and solvent may form a galvanic couple, the current of which causes the more electro-positive constituent to be attacked. Parts of a metal which have been subjected accidentally or intentionally to thermal or mechanical treatment different from that of the rest of the metal may show a difference in potential when connected by a conductor. With alloys, especially with those having a eutectic structure, the conditions are similar to those of impure metals. In an eutectiferous alloy the constituents in contact with a conducting solution may be easily corroded in the order of their positions in the electrochemical series. Solid-solution alloys are also more easily corroded than those forming chemical compounds. The subjoined metals and alloys, *e.g.*, will be corroded in the given order when immersed in sea-water:[2] +Zn, Al, Fe, Sn, brass (Cu 58–64, Zn 42–37 per cent.); aluminum-bronze (Cu 90, Al 10 per cent.); iron-bearing aluminum-bronze (Cu 88, Al 7, Si 1.5, Pb, Zn, etc., 0.5 per cent.); pure bronze (Cu 89, Sn 11 per cent.); zincy bronze (Cu 88, Sn 8, Zn 4 per cent.); copper, copper-phosphor bronze (Cu 94, Sn 6 per cent., some P). Al and Fe show little difference in potential, the same is the case with alloys standing between aluminum-bronze and phosphor-bronze.

TABLE 25.—HEAT OF FORMATION OF SOME COPPER ALLOYS

Formula	Molecular weights	Molecular heat of formation
(Cu,Zn^2)	63.6+130=193.6	10,143
(Cu,Zn)	63.6+65=128.6	5,783
(Cu^3,Al)	190.8+27=217.8	26,010
(Cu^2,Al)	127.2+27=154.2	21,278
(Cu^3,Al^2)	190.8+54=244.8	17,305
(Cu,Al)	63.6+27=90.6	1,887
(Cu^2,Al^3)	127.2+81=208.2	10,106
(Cu,Al^2)	63.6+54=117.6	−6,738

[1] Sexton, A. H., "The Corrosion and Protection of Metals," Scientific Publishing Co., Manchester, 1906.

Cushman, A. S., and Gardner, H. A., "Corrosion and Preservation of Iron and Steel," Authors, Philadelphia, 1910.

Bengough, *J. Inst. Met.*, 1911, V, 24.

[2] Diegel, *Verh. Verein. Beförd. Gewerbefl.*, 1899, LXXVIII, 1899, 321.

TABLE 26.—HEAT OF FORMATION OF SOME AMALGAMS.

Formula	Molecular weights	Molecular heat of formation	In dilute mercury solution
(Hg^{12},K)	$2,400+39=2,439$	34,600	25,600
(Hg^{4},K)	$800+39=839$	29,700	25,600
(Hg^{6},Na)	$1,200+23=1,223$	21,900	19,000
(Hg^{x},Au)	$x+197=197+x$	....	2,580
(Hg^{x},Ag)	$x+108=108+x$	....	2,470

CHAPTER IV

METALLIC COMPOUNDS

59. **Oxides.**—All metals can be made to enter into combination with O. Many oxides are sufficiently stable to serve as ore-forming minerals and to resist high temperatures. All iron and manganese ores, *e.g.*, are oxides; other oxides readily give up this O, *e.g.*, platinum and gold oxides. In the table of heats of formation, given below, Al_2O_3 is seen to be the strongest of the ordinary metallic oxides, and Au_2O_3 the weakest. Most of the lower oxides and several sesquioxides of the metals are basic oxides, *i.e.*, oxides which react with acids or with oxides more negative than themselves, *e.g.*, ZnO, HgO, Al_2O_3, FeO, Fe_2O_3, MnO, Mn_2O_3, NiO, PbO, Bi_2O_3. Some oxides are acidic (anhydrides), *i.e.*, oxides which react with water to produce acids, or are formed by removing water from acids, or react with oxides more positive than themselves to form salts, *e.g.*, As_2O_3, As_2O_5, Sb_2O_3, Sb_2O_5, WO_3. A few metals form important peroxides, *i.e.*, oxides which react with acids to form salts which correspond with oxides containing less O than the peroxides, *e.g.*, PbO_2, MnO_2, CrO_3, CaO_2. Some metals give rise to the formation of neutral or indifferent oxides, *i.e.*, oxides which do not form acids with water, which are not obtained by removing water from acids, and which do not form salts by reacting with either basic or acidic oxides, or with acids, *e.g.*, Fe_3O_4, PbO_2, Mn_3O_4.

All metallic oxides are solids; most of them show no luster; their colors differ from those of the metals from which they are prepared, and their specific gravity is lower. They are non-conductors of electricity[1] at ordinary temperature, but their resistance decreases as the temperature rises so that at 1,100° C. many may be classed as good conductors. Table 27 gives the data of Saklatwalla.[2]

Somerville[3] investigated the resistances of powdery ZnO, Fe_2O_3, CuO, Cu_2O, MgO, MnO_2 and Al_2O_3 up to 1,100° C. and found them to be very low when this temperature had been reached.

The oxides are insoluble in water. When heated, some are readily volatilized (As_2O_3, Sb_2O_3); others fuse at low temperatures (Cu_2O, PbO, Bi_2O_3); others are infusible in ordinary furnaces (ZnO, SnO_2, Fe_2O_3, Al_2O_3) and can be liquefied only at the temperature of the electric arc,[4] *e.g.*, FeO 1,419, Fe_3O_4 1,538 Fe_2O_3, 1,548, CaO 1,995, Al_2O_3 2,020, Cr_2O_3 2,050, UO_2 2,176° C.

Many oxides are decomposed when they are heated, *e.g.*, MnO_2 loses part of its O; Au_2O_3, Ag_2O, PtO, HgO all of it. Others take up O, *e.g.*, $Cu_2O+O=2CuO$; $2Fe_3O_4+O=3Fe_2O_3$; $PbO+O=PbO_2$. Some metals become oxidized

[1] Koenigsberger-Reichenheim, *Phys. Zt.*, 1906, VII, 570; *Electrician*, 1906, LVIII, 100.
[2] *Zt. Electrochem.*, 1907, XIII, 589.
[3] *Met. Chem. Eng.*, 1912, X, 422.
[4] Ruff, *Zt. angew. Chem.*, 1911, XXIV, 1134.

at ordinary temperature when exposed for some time to air, especially in the presence of moisture, *e.g.*, Fe, Cu, Pb, Zn, Mn, As.[1]

TABLE 27.—ELECTRIC RESISTANCE OF SOME METALLIC OXIDES

Temperature, deg. C.	Resistance ohms								
	Cr_2O_3	Fe_3O_4	SnO_2	NiO	CaO	Al_2O_3	SiO_2	MgO	ZrO
	At room temperature over 50,000								
400	6,000	11,750	900	3,000	......	.....	.....	..	.....
450	2,450	4,300	400	1,150	....	..	.....	......	......
500	1,250	2,450	235	490	.....	...	...	....	....
550	1,000	1,450	125	400	...	...	....	....	......
600	850	1,200	68	330	...	.....	....	.....	......
650	1,175	845	56	240	....	....	...	.....	....
700	1,010	710	47	195	....	......	....	....	......
750	950	510	42	121	...	....	......	...	....
800	690	357	37	220	...	....	......	......	....
850	668	290	32	280	....	.....	.....	......	....
900	520	210	28	190	..	.....	.....	......	....
950	395	162	25.5	81	.....	.....	.....	.....	......
1,000	345	127	24.0	115	......	....	.....	.....	....
1,050	335	117	23.0	93	....	......	....	......	....
1,100	330	105	22.25	45	....	......	....	....	....
Gas-blowpipe	....	....	....	....	550	650	590	600	580

Oxides are prepared (1) From metals and metallic compounds in the *dry* way by heating with access of air ($3Fe+2O_2=Fe_3O_4$; $4FeS+7O_2=2Fe_2O_3+4SO_2$); metals like Ag, Au, Pt, Ir remain unchanged. (2) From carbonates and most sulphates by simple calcining ($MCO_3=MO+CO_2$, $MSO_4=MO+SO_3$). (3) Fusion with oxidizing fluxes (PbO, K_2CO_3, $NaHSO_4$) or simply with access of air ($Pb+O=PbO$) causes the formation of oxides. (4) Heating in steam converts many metals into oxides; thus, metals having a strong affinity for O (Al, Mn, Zn, Fe, Cd, Ni) are oxidized at a red heat, and others with less affinity (Sb, W, Mo) require a temperature above a red heat, while metals like Pb, Bi, Cu are hardly attacked, and precious metals not at all.

In the *wet* way an oxide is formed by dissolving a metal in an oxygen-acid or by making it the positive pole of an electrolytic cell.

The decomposition of metallic oxides by heat alone is applicable to precious metals only. The ordinary mode of procedure in the dry way is to heat with

[1] Jordis-Rosenhaupt, *Zt. angew. Chem.*, 1908, xxi, 50.

reducing agents[1] (C, CO, H, C_xH_y, C N), certain metals with P, Si, S or some metallic P-ides, Si-ides and S-ides.

In the *wet* way oxides are decomposed by being precipitated from their solutions by means of compounds which are readily peroxidized ($FeSO_4$, $SnCl_2$, As_2O_3, SO_2, C_2O_3), by metals which are more electropositive, and by electrodeposition.

Electric and electrolytic smelting processes are employed in making special steels and in the reduction of alumina.

TABLE 28.—HEAT OF FORMATION OF OXIDES

Formula	Molecular weights	Molecular heat of formation	In dilute solution
(Mg,O)	24+16= 40	143,400	148,800
(Ba,O)	137+16=153	133,400	161,500
(Ca,O)	40+16= 56	131,500	149,600
(Sr,O)	87+16=103	131,200	158,400
(Al^2,O^3)	54+48=102	392,600	....
(TiO^2)	48+32= 80	218,500[2]	....
(Na^2,O)	46+16= 62	100,900	155,900
(K^2,O)	78+16= 94	98,200	165,200
(Si,O^2)	28+32= 60	180,000	....
(Mn,O)	55+16= 71	90,900	....
(B^2,O^3)	22+48= 70	272,600	270,000
(Zn,O)	65+16= 81	84,800	....
(Mn^3,O^4)	165+64=229	328,000	....
(P^2,O^5)	62+80=142	365,300	....
(Sn,O)	118+16=134	70,700	....
(Sn,O^2)	118+32=150	141,300	....
(CO,O)	28+16= 44	68,040[3]	73,940
(H^2,O)	2+16= 18	70,400 (solid) 69,000 (liquid) 58,060 (gas)	....
(Fe^3,O^4)	168+64=232	270,800	....
(Cd,O)	112+16=128	66,300	....
(Fe,O)	56+16= 72	65,700	....
(Fe^2,O^3)	112+48=160	195,600	....
(Co,O)	59+16= 75	64,100	....
(Mn,O^2)	55+32= 87	125,300	....
(Ni,O)	58.5+16= 74.5	61,500	....
(Sb^2,O^3)	240+48=288	166,900	....

[1] Fay-Seeker-Lane-Fergusion, *Polytech. Engineer* (Brooklyn), 1910, X, 72; *Eng. Min. J.*, 1911, XCI, 406; *J. Soc. Chem. Ind.*, 1910, XXIX, 877.

Greenwood, Slade, and Spring, *Tr. Brit. Chem. Soc.*, 1908, XCIII, 327, 1484, 1496, 2101; *Electrochem. Met. Ind.*, 1909, VII, 119.

[2] *Met. Chem. Eng.*, 1912, X, 255.

[3] Heat of combustion of 1 cbm CO=3062 Cal.; of 1 cbm H=2613 Cal.

TABLE 28.—HEAT OF FORMATION OF OXIDES—*Continued*

Formula	Molecular weights	Molecular heat of formation	In dilute solution
(As^2,O^3)	150+48=198	156,400	148,900
(Pb,O)	207+16=223	50,800	....
(C,O^2)	12+32= 44	97,200 (gas)	103,100
(Bi^2,O^3)	416+48=464	139,200	....
(Sb^2,O^5)	240+80=320	231,200	....
(As^2,O^5)	150+80=230	219,400	225,400
(Cu^2,O)	127.2+16=143.2	43,800	....
(Tl^2,O)	408+16=424	42,800	39,700
(Cu,O)	63.6+16= 79.6	37,700	....
(Ba,O^2)	137+32=169	145,500	....
(S,O^2)	32+32= 64	69,260 (gas)	77,600
(Pb,O^2)	207+32=239	63,400	....
(S,O^3)	32+48= 80	91,900	141,000
(Tl^2,O^3)	408+48=456	87,600	....
(C,O)	12+16= 28	29,160 (gas)	....
(Hg^2,O)	400+16=416	22,200	....
(Hg,O)	200+16=216	21,500	....
(Te,O^2)	125.5+32=157.5	....	78,300
(Pd,O)	106+16=122	21,000	....
(Pt,O)	195+16=211	17,000 (?)	....
(Ag^2,O)	216+16=232	7,000	....
(Au^2,O^3)	394+48=442	−11,500	....

TABLE 29.—HEAT OF FORMATION OF HYDROXIDES

Formula	Molecular weights	Molecular heat of formation	In dilute solution
(Li,O,H)	7+16+ 1= 24	112,300	118,100
(Mg,O^2,H^2)	24+32+ 2= 58	217,800	....
(Sr,O^2,H^2)	87+32+ 2=121	217,300	227,400
(Ca,O^2,H^2)	40+32+ 2= 74	215,600	219,500
(K,O,H)	39+16+ 1= 56	104,600	117,100
(Na,O,H)	23+16+ 1= 40	102,700	112,500
(N,H^5,O)	14+ 5+16= 35	88,800	90,000
(Al,O^3,H^3)	27+48+ 3= 78	301,300	....
		70,400 (solid)	....
(H,O,H)	1+16+ 1= 18	69,000 (liquid)	....
		58,060 (gas)	....
(Tl,O,H)	204+16+ 1=221	57,400	54,300
(Bi,O^3,H^3)	208+48+ 3=259	171,700	....
(Zn,O^2,H^2)	65+32+ 2= 99	83,500	....
(Te,O^2,H^2)	127+32+ 2=161	78,300	....
(Te,O^3,H^3)	127+48+ 3=178	....	99,500
(Se,O^2,H^2)	79+32+ 2=113	52,400	51,500
(Se,O^3,H^3)	79+48+ 3=130	....	79,300
(Tl,O^3,H^3)	204+48+ 3=255	43,800	....

60. Sulphides.—Metallic sulphides occur frequently native and are formed in many metallurgical processes. Sulphides of all the metals are known. The affinity of the different metals for S at ordinary temperatures is indicated by their heats of formation, Table 30. These are generally lower than those of the corresponding oxides. The affinity for S at ordinary temperature differs often from that in the fused state. Fournet[1] placed the leading metals according to their behavior in smelting in the following order:

Cu, Fe (Co, Ni), Sn, Zn, Pb, Ag (Hg, Au), As, Sb, in which Cu has the greatest and Sb the smallest affinity for S.

Schütz[2] has reinvestigated the affinity of S for the leading metals and placed the latter in the following order:

Mn, Cu, Ni, Fe, Sn, Zn and Pb.

The experiments of Schütz show that the reaction $MS+N \rightleftarrows M+NS$ is reversible.[3] This has been shown in a striking manner by Baykoff and Troutneff[4] in connection with $Cu_2S+Fe \leftrightarrows Cu_2+FeS$. Thus Gibb and Philp[5] in fusing Cu_2S with an excess of Fe obtained a button of Cu and a matte with Cu 60.6, Fe 17.8, S 21.6 per cent., the reaction taking place being $Cu_2S+Fe \rightarrow Cu_2+FeS$; while Baykoff and Troutneff in fusing FeS with an excess of Cu obtained a button (Cu 94.99, Fe 5.32, S n.d.) and a matte with Cu 60.7, Fe 17.6, S 21.6 per cent., the reaction taking place being $FeS+Cu_2 \rightarrow Cu_2S+Fe$. The compositions of the two mattes are identical. It appears therefore that the direction which a reversible reaction between two neighboring metals will take is governed by the preponderance of one over the other. The existence of the reactions $3Cu_2S+Fe=(Cu_2S)_2.FeS+2Cu$ and $4Cu+3FeS=(Cu_2S)_2.FeS+Fe$ has been proved by Juschkewitsch.[6]

Sulphides are classed in a manner similar to oxides as basic, acidic, per-, and indifferent. Basic sulphides, commonly monosulphides (CuS, FeS, ZnS, PbS, Ag_2S, HgS), when treated with an acid give a salt and H_2S. Acidic sulphides (As_2S_3, Sb_2S_3, Bi_2S_3) give in combination with basic sulphides, double sulphides ($Cu_2S.Fe_2S_3$; $4PbS.As_2S_3$; $Ag_2S.Sb_2S_3$; $3Cu_2S.4Bi_2S_3$). In these the sulphide of the metal which is more electropositive is the basic radicle; that which is more negative the acidic. As to the constitution of artificial multiple sulphides (matte, § 343), it is known that they are not homogeneous, but are composed of distinct mineral entities. Persulphides (FeS_2, MnS_2, CaS_5) are sulphides which contain more S than either basic or acidic sulphides. Indifferent sulphides (Cr_3S_4) are those which do not react as a basic or acidic sulphide, and which from their composition cannot be persulphides.

Metallic sulphides are solids; have different kinds of luster; show a considerable variety in color; have a specific gravity lower than the component metal;

[1] *Ann. Min.*, 1833, IV, 3, 225.
[2] *Metallurgie*, 1907, IV, 659, 694.
[3] Röntgen, *Metallurgie*, 1906, III, 486.
[4] *Rev. Mét.*, 1909, VI, 536.
[5] *Tr. A. I. M. E.*, 1906, XXXVI, 609.
[6] *Metallurgie*, 1912, X, 543.

are as a rule brittle and more readily fusible than the corresponding oxides; some are volatile at a low temperature (As_2S_3, Sb_2S_3, HgS, CdS), but most of them are stable at elevated temperatures and good conductors of electricity;[1] practically all are insoluble in H_2O.

Sulphur and the metals do not unite in the cold; in a few instances the two have been made to combine in a finely-divided intimate mixture by subjecting them to a high pressure (Spring).

Metallic sulphides are formed in the *dry* way:

(1) By heating metal with S. The union is accompanied by the evolution of heat and often of light (Cu, Fe, Pb, Sn . . .); some metals (Zn, Al) do not combine readily with S; Au alone remains unchanged.

(2) By heating metallic oxide with S, when part of the S passes off as SO_2, thus: $2HgO+3S=2HgS+SO_2$, or $6CuO+4FeS=3Cu_2S+4FeO+SO_2$. With oxides difficult of reduction (MnO_x), oxysulphides are often formed.

(3) By heating metallic sulphates with reducing agents: $PbSO_4+4C=PbS+4CO$. With metallic sulphates which are decomposed by heat into metallic oxide and SO_3 before the C begins to act, there may result metallic oxide (or metal), SO_2 and CO_2, *e.g.*, $2ZnSO_4+C=2ZnO+2SO_2+CO_2$.

(4) By heating a metal or metallic oxide in a stream of CS_2 : $4Cu+CS_2=2Cu_2S+C$; $2FeO+CS_2=2FeS+CO_2$.

(5) By the action of H_2S upon a metal or metallic oxide in the cold: $Cu_2+H_2S=Cu_2S+H_2$; $Ag_2O+H_2S=Ag_2S+H_2O$.

(6) By the action of an alkaline sulphide upon a metallic oxide: $Cu_2O+K_2S=Cu_2S+K_2O$.

In the *wet* way metallic sulphides are formed:

(1) By precipitating dissolved or suspended oxide with H_2S: $CuSO_4+H_2S=CuS+H_2SO_4$, or with alkaline sulphides: $FeSO_4+Na_2S=FeS+Na_2SO_4$.

(2) By the reduction of sulphates with organic substances.

Metallic sulphides are decomposed:

(1) By heat alone. Thus Au_2S_3 and PtS give up all their S; persulphides (FeS_2, MnS_2) and disulphides (CuS, SnS_2) lose part of their S being converted into basic sulphides.

(2) By heating in a current of H; this applies especially to Ag, Bi, Sn, Sb: $Ag_2S+2H=Ag_2+H_2S$.

(3) By heating with a metal or other element that has a stronger affinity for S (Fournet series), $HgS+Fe=Hg+FeS$; $3MoS_2+4Al=3Mo+2Al_2S_3$; $MoS_2+Si=Mo+SiS_2$.

(4) By heating with access of air (see Oxidizing Roasting, §173, Oxidizing Fusion, §215, and Converting, §234).

(5) By heating with metallic oxides or salts. $PbS+2PbO=Pb_3+2SO_2$; $Cu_2S+2Cu_2O=Cu_6+SO_2$; $PbSO_4+PbS=Pb_2+2SO_2$; $FeS+30PbO=Fe_3O_4$, Pb, PbO, SO_2; $CuS+25\ PbO=Cu_2O$, Pb, SO_2.

[1] Hittorf, *Ann. Phys.*, 1851, XXIV, 1.
Guinebaut, *Compt. Rend.*, 1902, CXXXIV, 134; *Zt. Electrochem.* 1902, VIII, 889.

(6) By heating to a high temperature (1500° C.) with CaO and C when the reaction $MS_2+2CaO+2C=M_2+2CaS+2CO$ may take place.[1]

(7) By heating in a current of steam[2] (oxidizing roasting, §173) $Ag_2S+H_2O=Ag_2O+H_2S$; $3Ag_2S+2H_2O=6Ag+2H_2S+SO_2$; $CaS+4H_2O=CaSO_4+8H$; $3FeS+4H_2O=Fe_3O_4+3H_2S+H_2$.

(8) By heating with alkali or alkaline earths; the decomposition is effected only in part, the alkali or alkaline earth being more or less converted into sulphide and sulphate: $7PbS+4K_2O=4Pb+3K_2PbS_2+K_2SO_4$ and $3K_2PbS_2+3Fe=3Pb+3K_2FeS_2$; $4HgS+4CaO=4Hg+3CaS+CaSO_4$.

(9) By heating in a current of Cl, when metallic chloride and SCl_2 are formed (see Formation of Chlorides, §68).

(10) Heating with C, CO, C_xH_y has next to no decomposing effect.

(11) In the wet way metallic sulphides are decomposed by treatment with acids (HNO_3, H_2SO_4, aq. reg.) and solutions of nitrates, sulphates and chlorides.[3] Schürmann[4] placed the following sixteen metallic sulphides in the order Pd, Hg, Ag, Cu, Bi, Cd, Sb, Sn, Pb, Zn, Ni, Co, Fe, As, Tl, Mn as regards the rate at which they are decomposed by the nitrates, sulphates and chlorides of the other metals. PdS is not decomposed by the salts of any of the other metals, while $PdCl_2$ converts the sulphides of the other metals into chlorides. The reverse is the case with MnS which is decomposed by the salts of the other metals, while $MnSO_4$ has no decomposing effect whatever.

(12) Electrolytic decomposition in the dry (Al_2S_3, Cu_2S) and the wet (Siemens-Halske, Hoepfner) way has been tried, but has not developed into a successful method.

TABLE 30.—HEAT OF FORMATION OF SULPHIDES

Formula	Molecular weights	Molecular heat of formation	In dilute solution
(Li^2,S)	14+32= 46	....	115,400
(K^2,S)	78+32=110	103,500	113,500
(Ba,S)	137+32=169	102,900	100,800
(Sr,S)	87+32=119	99,300	106,700
(Ca,S)	40+32= 72	94,300	100,600
(Na^2,S)	46+32= 78	89,300	104,300
(Mg,S)	24+32= 56	79,400	....
(K,S^2)	39+64=103	59,300	59,700
(Na,S^2)	23+64= 87	49,500	54,400
(Mn,S)	55+32= 87	45,600	....
(Zn,S)	65+32= 97	43,000	....
(Al^2,S^3)	54+96=150	126 400	....
(N,H^5,S)	14+5+32= 51	40,000	36,700
(Cd,S)	112+32=144	34 400	....

[1] Lehmer, *Metallurgie*, 1906, III, 550, 596.
[2] Gautier, *Bull. Soc. chim.*, 1906, XXXV, 934.
[3] Cammerer, Action of $FeCl_3$., *Berg. Hüttenm. Z.*, 1891, L, 201, 261, 282.
[4] Thesis, Tübingen, 1888.

TABLE 30.—HEAT OF FORMATION OF SULPHIDES—(*Continued*)

Formula	Molecular weights	Molecular heat of formation	In dilute solution
(B^2,S^3)	22+96=118	75 800	......
(Fe,S)	56+32= 88	24,000	......
(Co,S)	59+32= 91	21,900	......
(Tl^2,S)	204+32=236	21,600	......
(Cu^2,S)	127.2+32=159 2	20,300	......
(Pb,S)	207+32=239	20,200	......
(Si,S^2)	28+64= 92	40,000	......
(Ni,S)	58.5+32= 90.5	19,500	......
(Sb^2,S^3)	240+96=336	34,400	......
(Hg,S)	200+32=232	10,600	......
(Cu,S)	63.6+32= 95.6	10,100	......
(H^2,S)	2+32= 34	4,800 (gas)[1]	9,500
(Ag^2,S)	216+32=248	3,000	......
(C,S^2)	12+64= 76	−25,400 (gas)	......
		−19,000 (liquid)	......
(I,S)	127+32=159	9,000	......

61. **Selenides.**—Selenium is found in combination with Pb, Bi, Cu, Co, Ag and Hg, but selenides occur sparingly in nature.[2] However, Se can be made to combine with most metals. The heats of formation are slightly lower than those of the sulphides. Selenides are classed as basic and acidic as are the corresponding sulphides. Metallic selenides are all solid; have a sub-metallic luster when crystallized, and are dark-colored; they are readily fusible and the greater number of those examined is easily volatilized; all are insoluble in water and many insoluble in dilute acids.

Metallic selenides are prepared in the *dry* way as follows:

(1) By heating with Se or in Se vapor; the combination is often attended by the development of light (Mn,Ni,Co).

(2) By reducing selenites or selenates with C, H or Al.

In the *wet* way they are prepared by the action of H_2Se or an alkali selenide upon the salts of metals.

Metallic selenides are decomposed to a small extent by heating in a current of H; this is especially the case with Sn, Cu and Ag; acidic are converted into basic selenides ($CrSe_3 = CrSe + Se_2$; $SnSe_2 = SnSe + Se$). Heating with a metal that has a stronger affinity for Se causes decomposition as does heating with access of air (see Oxidizing Roasting, §173). Heating in Cl gives $SeCl_4$ and metallic chloride, and in HCl yields H_2Se and metallic chloride. The constitutions of $Cu-Cu_2Se$, $Ag-Ag_2Se$, and $Pb-PbSe$ have been investigated by

[1] Molecular heat of combustion of H_2S = 122,520 Cal., and heat of combustion of 1cbm. H_2S = 5513 Cal.

[2] Merz, *Oest. Zt. Berg. Hüttenw.*, 1904, LII, 59, 71, 86, 99; *Metallurgie*, 1904, I, 142, 163, 185.

Friedrich and Leroux,[1] that of Sb—Se by Pélabon,[2] and that of Sn—Se by Biltz and Mecklenburg.[3]

TABLE 31.—HEAT OF FORMATION OF SELENIDES

Formula	Molecular weights	Molecular heat of formation	In dilute solution
(Li²,Se)	14+79= 93	83,000	93,700
(K²,Se)	78+79=157	79,600	87,900
(Ba,Se)	137+79=216	69,900	....
(Sr,Se)	87+79=166	67,600	....
(Na²,Se)	46+79=125	60,900	78,600
(Ca,Se)	40+79=119	58,000	....
(Zn,Se)	65+79=144	30,300	....
(Cd,Se)	112+79=191	23,700	....
(Mn,Se)	55+79=134	22,440	....
(N,H⁵,Se)	14+5+79= 98	17,800	12,800
(Cu,Se)	63 6+79=142.6	17,300	....
(Pb,Se)	207+79=286	17,000	....
(Fe,Se)	56+79=135	15,200	....
(Ni,Se)	58.5+79=137.5	14,700	....
(Co,Se)	59+79=138	13,900	....
(Tl²,Se)	408+79=487	13,400	....
(Cu²,Se)	127.2+79=206.2	8,000	....
(Hg,Se)	200+79=279	6,300	....
(Ag²,Se)	216+79=295	2,000	....
(H²,Se)	2+79= 81	−25,100 (gas)	−15,800
(N,Se)	14+79= 93	−42,300	....

62. Tellurides.[4]—Tellurium is found in combination with the same metals as Se, but tellurides[5] occur more frequently than selenides; they form important gold ores. Te unites with metals in a manner similar to that of Se. The constitutions of Te-Pb and of Te-Sb alloys have been studied by Fay-Gillson[6] and Fay-Ashley; that of Te–Bi by Mönkemeyer;[7] of Te–Au by Rose[8] and Caste;[9] of Te–Ag and Au by Pellini-Quercigh;[10] of Te–Sn by Biltz–Mecklenburg;[11] of Te–Cd

[1] *Metallurgie*, 1908, v, 355.
[2] *Compt. Rend.*, 1911, CLIII, 343.
[3] *Zt. anorg. Chem.*, 1909, LXIV, 226.
[4] Tibbals, Jr., C. A., "A Study of Tellurides," *Bull.* 274, Univ. of Wisc., 1910.
[5] Kemp, *Min. Ind.*, 1897, VI, 295.
Sharwood, *Min. Sc. Press*, 1907, XCIV, 731.
Lehner, *Econ. Geol.*, 1909, VI, 544.
[6] *Trans. A. I. M. E.*, 1901, XXXI, 527, 544.
[7] *Zt. anorg. Chem.*, 1905, XLVI, 415.
[8] *Tr. Inst. Min. Met.*, 1907–08, XVII, 285.
[9] *Compt. Rend.*, 1911, CLII, 859.
[10] *J. Inst. Met.*, 1911, V, 317.
[11] *Zt. anorg. Chem.*, 1909, LXIV, 226.

by Kobayashe;[1] of Te–S by Chikashige,[2] and of Te–Zn by Kobayashe.[3] Metallic tellurides are all solid, they have a metallic luster, show various colors, are readily fusible, some are volatile; all are insoluble in H_2O; some are insoluble in dilute acids, and all are decomposed by HNO_3, *aq. reg.*, and Br-water.

Metallic tellurides are prepared in the same way as are the selenides. They are partly decomposed by heat alone (Au, Hg) or by heating in a current of H (Ag, Au, Cu, Zn), also by heating with access of air (see Oxidizing Roasting, §174).

TABLE 32.—HEAT OF FORMATION OF TELLURIDES

Formula	Molecular weights	Molecular heat of formation	In dilute solution
(Zn,Te)	65+126=191	31,000	....
(Cd,Te)	112+126=138	16,600	....
(Co,Te)	59+126=185	13,000	....
(Fe,Te)	56+126=182	12,000	....
(Ni,Te)	58.5+126=184.5	11,600	....
(Tl^2,Te)	408+126=534	10,600	....
(Cu^2,Te)	127.2+126=253.2	8,200	....
(Pb,Te)	207+126=333	6,200	....
(H^2,Te)	2+126=128	−34,900 (gas)	....

63. Nitrides.—Metallic nitrides (RN, R_2N, R_3N,R =monovalent metal) are of very little importance. They do not occur native, but may be prepared by heating a metal in a current of N (Ti) or NH_3;[4] a metallic oxide (HgO, CuO, TiO_2, Nb_2O_5) or chloride ($FeCl_2$, $CrCl_3$, $MoCl_5$, $TaCl_5$) in a current of NH_3; or an amide ($3\ Zn(NH_2)_2 = Zn_3N_2 + 4NH_3$). Most nitrides have a metallic aspect; some are decomposed by heat, others withstand very high temperatures. Up to this year the only nitride of metallurgical importance has been the refractory, bright-red crystalline cyanonitride of titanium (Ti_5CN_4) sometimes formed in the iron blast-furnace; the newcomer AlN[5] has a great future. For Ca_3N_2 consult Bredig-Fraenkel-Wilke[6] and Beck.[7]

[1] *Op. cit.*, 1910, LXIX, 1.
[2] *Op. cit.*, 1911, LXXII, 109.
[3] *J. Inst. Met.*, 1911, VI, 336.
[4] Fe, Co, Ni, Cu, Zn, (Beilley-Henderson, *Tr. Chem. Soc.*, 1901, LXXIX, 1245); Zn, Fe, Al, (White-Kirschbaum, *J. Am. Chem. Soc.*, 1906, XXVIII, 1343); Cr, Mo, W, Ti, Sn, Pb, Zn, Cd, Al, Pd, (Henderson-Galletly, *J. Soc. Chem. Ind.*, 1908, XXVII, 387).
[5] Serpek process: Richards, *Met. Chem. Eng.*, 1913, X, 113; Tucker, *ibid.*, 119.
[6] *Zt. Electrochem.*, 1907, XIII, 69, 605.
[7] *Metallurgie*, 1908, V, 504.

TABLE 33.—HEAT OF FORMATION OF NITRIDES

Formula	Molecular weights	Molecular heat of formation	In dilute solution
(H^3,N)	3+14= 17	12,200 (gas) 16,600 (liquid)	21,000 ....
(Ba^3,N^2)	411+28=439	149,400	....
(Li^3,N)	21+14= 35	49,500	....
(K,H^3,N)	39+3+14= 56	30,700	....

64. **Hydrides.**—Metallic hydrides[1] do not occur native and are of little importance. H can be made to unite with a few metals in the dry and in the wet way; it forms a true chemical compound, or is simply absorbed by the metal; Pd, *e.g.*, absorbs H at atmospheric pressure up to 115° C. and forms the compound Pd_2H; above 115° no more H is taken up. Pd_2H is also formed if acidulated water is electrolyzed with Pd serving as negative pole. Pd, Pt[2], Rh, Ru, Ir, Os, Fe, Ni, Co, Cu, Au, Ta and some other metals absorb considerable amounts of H under similar conditions; Ag, Pb, Hg, Mo, Cr, Zn, Bi, W, Va, Al do not. Besides Pd_2H there may be mentioned K_2H_4, Na_2H_4, Cu_4H_2, Li_2H_2, CaH_2, BaH_2; the existence of hydrides of Ag, Ni, Fe seems doubtful.

TABLE 34.—HEAT OF FORMATION OF METALLIC HYDRIDES

Formula	Molecular weights	Molecular heat of formation	In dilute solution
(Sr,H^2)	87+2= 89	38,400	....
(Ba,H^2)	137+2=139	37,500	....
(Pt^{10},H)	1,950+1=1,951	14,200	....
(Pd^{15},H)	1,590+1=1,591	4,600	....
(Si,H^4)	28+4= 32	−6,700 (gas)	....

65. **Phosphides.**—Excepting small amounts of (Ni_x and Fe_y) P in meteoric iron, metallic phosphides do not occur native. The affinity of metals for P is not so strong as that for O and for S. The heats of formation of Mn_3P_2 and FeP only have been determined so that the relative affinities of the leading metals for P are not known. The probable reason for this is that with a few metals only (Zn, Cd, Mn, Fe and perhaps Ni,[3] Co, Cr, Cu[4]) have phosphides of definite chemical composition been isolated; the product usually obtained is

[1] Moissan, *V Internat. Congr. Appl. Chem.*, Berlin, 1903, I, 78.
[2] Lucas, *Zt. Electrochem.*, 1905, XI, 182.
Pirani, *op. cit.*, 1905, XI, 555.
Fokin, *op. cit.*, 1906, XII, 749.
[3] Konstantinoff, *Zt. anorg. Chem.*, 1908, LX, 405.
[4] Heyn-Bauer, *Zt. anorg. Chem.*, 1907, LII, 129.
Huntington-Desch, *Tr. Faraday Soc.*, 1908, IV, 51.

a mixture difficult to separate. The constitutions of Fe–P[1] and Mn–P[2] only, have so far been investigated. Metallic phosphides are solid and brittle; have a metallic luster if the percentage of P is low; are mostly white to gray; readily fusible and insoluble in H_2O. The addition of a small amount of P to a metal causes this to become brittle. Thus steel is brittle if it contains over 0.1 per cent. P, while an addition of the same amount to Cu or some of its alloys does not affect its mechanical properties unfavorably. On the contrary it assists in the production of sound castings, as the presence of P hinders the oxidation of Cu, Sn, etc.

Phosphides are prepared: (1) By heating a metal, metallic oxide or chloride, with P. The union of some metals with P is attended by the development of light (Pt, Sn, Zn), of others only by heat (Ag, Au, Cu, Pb, Cd, W). In heating metallic oxides (ZnO, CdO) with P, some metallic phosphide is formed; with metallic chlorides, part of the P is volatilized as PCl_3.

(2) By heating in a brasqued crucible a mixture of metallic phosphate and C, or of P_2O_5, metallic oxide and C (Fe, Cr, Mo, W, Ti).

(3) By reducing metallic phosphate with H.

(4) By the action of PH_3 upon metal salts.

Phosphides are decomposed: (1) By heat alone; Ag, Au and Bi give up all the P, the former two with sprouting; Fe and Sn retain a considerable amount of P, Cu very little.

(2) By heating with access of air or oxidizing reagents, when phosphates are formed or a mixture of metal, metallic oxide and P_2O_5.

(3) By treating with HNO_3, Cl-water, $HClO_3$, with the formation of metallic phosphate, or with HCl with the formation H_3P.

TABLE 35.—HEAT OF FORMATION OF PHOSPHIDES

Formula	Molecular weights	Molecular heat of formation	In dilute solution
(Mn^3,P^2)	165+62=227	70,900	....
(H^3,P)	3+31= 34	4,900 (gas)	....
(Fe,P)	56+39= 95	nearly zero.	....

66. Arsenides. GENERAL PROPERTIES.—Pure metallic arsenides are not of frequent occurrence (arsenical nickel and cobalt ores); they are sometimes formed in smelting operations (arsenical speise, §348). In arsenical minerals As is often in part replaced by Sb and S; the same is generally the case with the smelter product, speise. Heats of formation of metallic arsenides have not been

[1] Stead, *J. I. and St. I.*, 1900, II, 60; *Metallographist*, 1901, IV, 89, 199, 332.
Saklatwalla, *J. I. and St. I.*, 1908, II, 93.
Gerke, *Metallurgie*, 1908, V, 604.
Konstantinoff, *Zt. anorg. Chem.*, 1910, LXVI, 209.

[2] Schemtschuschny-Efremow, *Zt. anorg. Chem.*, 1908, LVII, 241.

determined. In smelting arsenical nickel ores, it has been found that Ni h the greatest affinity for As, then follow Cu, Co, Fe in the order given. Tl first three form definite chemical compounds[1]: Ni_4As, Cu_3As, Co_4As. Tl constitution of Cu–As alloys has been studied by Hiorns[2], Friedrich[3] and Be gough-Hill[4]; that of Mn–As by Schoen[5]; of Fe–As by Hilpert-Dieckmann of Sn–As by Jolibois-Duprey[7]; of Hg–As by Dumesnil[8] and of Sb–As by Parr vano-Cesaris.[9]

The presence of a very small amount of As often has a decided effect upc the physical properties of a metal; it thus makes Au brittle, reduces the electr conductivity of Cu, and causes Pb to assume a spherical form in granulatir (shot).

PREPARATION OF ARSENIDES.—Metallic arsenides are prepared in the *dr* way:

(1) By strongly compressing the finely divided constituents.

(2) By heating metals with As, the union is accompanied by evolution heat and sometimes of light (Zn, Pt). Any excess of As is volatilized by tl rise of temperature due to the reaction.

(3) By reducing metallic arsenites or arsenates with C or KCN. A mi ture of As_2O_3 or As_2O_5 with a metal or metallic oxide and reducing agent giv a similar result.

(4) By heating some metals or metallic oxides in a current of AsH_3 whe metallic arsenide and H, or H_2O, are formed (Sn, Cu).

In the *wet* way some metallic arsenides are prepared by the action of AsI upon metal salts. Precious metals are reduced to the metallic state; MnO SnO- and FeO-salts are not acted upon, but most of the other metals a precipitated

DECOMPOSITION OF ARSENIDES.—Metallic arsenides are decomposed in tl *dry* way:

(1) By heat alone. Thus arsenides of Bi, Sn, Pt, Ag, Au retain only trac of As, while Ni, Cu, Co, Fe lose part of their As leaving behind definite, and Z Pb indefinite, chemical compounds.

(2) By heating with access of air (see Oxidizing Roasting, §174).

(3) By heating with metallic oxides or salts. In roast-smelting, the solid of the oxides formed changes As into volatile As_2O_3 or stable basic arsenate.

(4) By heating with a metal that has stronger affinity for As (see ord given above).

(5) By heating in a current of steam (see Oxidizing Roasting).

[1] Plattner-Cornwall, "Blow-pipe Analysis," Van Nostrand, New York, 1902, p. 424.
[2] *Electrochemist and Metallurgist*, 1903–04, III, 648.
[3] *Metallurgie*, 1905, II, 484.
[4] *J. Inst. Met.*, 1910, III, 34.
[5] *Metallurgie*, 1911, VIII, 737.
[6] *Ber. deutsch. chem. Ges.*, 1911, XLIV, 2378.
[7] *Compt. Rend.*, 1911, CLII, 1312.
[8] *Op. cit.*, 1911, CLII, 868.
[9] *Internat. Zt. Metallogr.*, 1912, II, 70.

(6) By fusing with alkali carbonate and nitrate which convert the As into alkali arsenite and arsenate soluble in H_2O, and the heavy metal into oxide.

(7) By fusing with alkali carbonate and S which convert the As into alkali sulpharsenide and the heavy metal into sulphide.

(8) By heating in a current of Cl or HCl, which converts the metal into chloride and the As into volatile $AsCl_3$ (see Chlorides, § 68).

(9) By heating with S which acts in a manner similar to that of Cl, only that As_xS_y is not as volatile as $AsCl_3$.

In the *wet* way some arsenides (Zn, Sn, Fe) are converted into chlorides or sulphates with evolution of AsH_3 by treatment with HCl or H_2SO_4; most arsenides, however, remain unattacked and require HNO_3 and aqua regia for their decomposition, and conversion into arsenates.

TABLE 36.—HEAT OF FORMATION OF ARSENIDES

Formula	Molecular weight	Molecular heat of formation	In dilute solution
(H^3,As)	3+75 = 78	−44,200 (gas)	....

67. Antimonides.—They occur native, as do arsenides, and are also formed in smelting operations (antimonial speise). In a general way they show the same metallurgical behavior as do arsenides. They are lustrous, hard and brittle. Little is known in a general way which might serve to differentiate them from arsenides. The affinity of the different metals for Sb is not the same as that for As. Thus Fe combines readily with As, but not with Sb, while Cu, Pb, Au, Pt have a greater affinity for Sb than for As; further, metallic antimonates as a rule are more difficult to decompose than are arsenates.

TABLE 37.—HEAT OF FORMATION OF ANTIMONIDES

Formula	Molecular weight	Molecular heat of formation	In dilute solution
(H^3,Sb)	3+120=123	−86,800 (gas)	....

68. Chlorides.—Metallic chlorides occur sparingly native (AgCl, HgCl), but metals are frequently converted into chlorides in order to render them amenable to wet and electrolytic processes. Chlorides of all the metals are known. Most chlorides are stronger compounds than the corresponding oxides or sulphides. Many chlorides are known as hydrochlorides ($SnCl_4.3H_2O$); some form double salts with alkali chlorides ($KAuCl_2$, Na_2HgCl_4, K_2PtCl_6), others acid chlorides with HCl (H_2CuCl_4, $HAuCl_4$, H_2CdCl_4); oxychlorides, *i.e.*, chlorides of the more electronegative metals combining with the oxides of the same metals ($2CuO.CuCl_2$, $3CuO.CuCl_2$, $SnCl_4.SnO_2$), are not uncommon; occasion-

ally sulphochlorides ($SnCl_4.2SCl_4$, $SbCl_5.SCl_4$) are encountered. At ordinary temperature most metallic chlorides are solid; some, however, are liquid ($SnCl_4$, $AsCl_3$, $SbCl_5$, $TiCl_4$); their colors vary greatly. Solid chlorides are more readily fusible[1] than the corresponding metals; all are easily volatilized excepting $MnCl_2$; some pass directly from the solid state to that of vapor without melting ($HgCl$, $FeCl_3$). AgCl is non-volatile at a red heat but is mechanically carried along by volatile chlorides. A perchloride as a rule is more readily volatilized than its lower chloride ($SnCl_4$–$SnCl_2$, Fe_2Cl_6–$FeCl_2$, $CuCl_2$–Cu_2Cl_2, $AuCl_3$–$AuCl$). Some chlorides are in part decomposed while being volatilized ($CuCl_2$, $AuCl_3$). Most chlorides are soluble in water; AgCl, Hg_2Cl_2, Cu_2Cl_2, AuCl, $PtCl_2$ are insoluble; $PbCl_2$ is sparingly soluble; $BiCl_3$, $SnCl_2$, $SbCl_3$ are more or less decomposed by H_2O forming basic salts ($BiOCl$, $SnOCl_2$, $SbOCl$). Heating with an acid body giving off O causes Cl to be liberated:

$$2NaCl+MnO_2+3H_2SO_4=MnSO_4+HNaSO_4+2H_2O+Cl_2.$$

FORMATION OF CHLORIDES.—Metallic chlorides are formed in the *dry* way

(1) By the action of Cl upon metals, metallic oxides, sulphides, arsenides and antimonides. The union of dry Cl with a finely divided metal forming anhydrous chloride is frequently accompanied by the emission of light; in the formation of chlorides of Pb, Ag, Au and Pt, heat only is developed. Metallic oxides which have to part with their O are not readily converted into chlorides as are the metals proper; chloridation is assisted by the presence of C which burning to CO or CO_2, aids in the decomposition of the oxide. The S, As, and Sb of metallic sulphides, arsenides and antimonides passes off as SCl_2, $AsCl_3$, $SbCl_3$.

(2) By the partial decomposition of higher chlorides. Thus $CuCl_2$, $FeCl_3$ are converted into Cu_2Cl_2 and $FeCl_2$ upon heating. Heating the lower chloride in a current of Cl restores the higher chloride.

(3) By the action of HCl gas upon metals, metallic oxides, sulphides, arsenides, sulphates, arsenates and antimonates. The metals, oxides, sulphides and arsenides form H, H_2O, H_2S, AsH_3; the sulphates, arsenates and antimonates, SCl_2, $AsCl_3$ and $SbCl_3$.

(4) By the interaction of metallic oxygen-salts and chlorides. Thus vapors of NaCl act upon metallic sulphates, arsenates and antimonates being converted into Na_2SO_4, etc., and changing the oxygen compounds into chlorides (see Chloridizing Roasting, §177).

In the *wet* way metallic chlorides are formed:

(1) By the action of HCl or aqua regia upon many metals, metallic oxides, sulphides and some arsenides.

(2) By the interaction of oxygen salts and chlorides:

$$Fe(Cu)SO_4+2NaCl=Fe(Cu)Cl_2+Na_2SO_4.$$

DECOMPOSITION OF CHLORIDES.—Metallic chlorides are decomposed in the *dry way:*

[1] Arndt-Kunze, *Zt. Electrochem.*, 1912, XVIII, 994.

(1) By heating in a current of H with the formation of HCl : $AgCl+H=Ag+HCl$. The decomposition is often imperfect.

(2) By heating in a current of steam. Many chlorides (Fe, Mn, Co, Ni, Zn, Bi, Cu, Sb . . .) are thus converted into oxides or basic chlorides. In the presence of C some chlorides are reduced to the metallic state:

$$2AgCl+H_2O+C=Ag_2+2HCl+CO.$$

(3) By heating in a current of steam in the presence of SiO_2 and O: $2NaCl+SiO_2+H_2O=Na_2SiO_3+HCl$ (see Chloridizing Roasting).

(4) By heating with S. Some chlorides are converted into sulphides with the liberation of SCl_2.

(5) By heating with a metal that has a stronger affinity for Cl:

$$AlCl_3+Na_3=Al+3NaCl.$$

(6) Heating with C does not decompose chlorides.

(7) By electrolytic reduction: NaCl, $ZnCl_2$, $MgCl_2$, $CaCl_2$ . . .

In the *wet* way metallic chlorides are decomposed:

(1) By treating with a metal that has a stronger affinity for Cl: $2AgCl+(Hg_2, Cu, Fe, Zn)=Ag_2+(Hg_2, Cu, Fe, Zn)Cl_2$. The decomposition may be perfect ($6HgCl_2+6Fe=6FeCl_2+6Hg$) or imperfect ($6HgCl_2+2Fe=3Hg_2Cl_2+Fe_2Cl_6$).

(2) By treating with alkali or alkaline earths or their carbonates:

$$CuCl_2+2NaOH=CuO+H_2O+2NaCl.$$

(3) By treating with substances that are readily chloridized ($2AuCl_3+6FeSO_4=2Au+Fe_2Cl_6+2Fe_2.(SO_4)_3$) or oxidized ($2AuCl_3+3C_2O_3+3H_2O=2Au+6HCl+6CO_2$).

(4) By electrolytic reduction: Cu_2Cl_2.

TABLE 38.—HEAT OF FORMATION OF CHLORIDES

Formula	Molecular weights	Molecular heat of formation	In dilute solution
(K,Cl)	39+ 35.5= 74.5	105,700	101,200
(Ba,Cl^2)	137+ 70 =207	197,100	198,300
(Be,Cl_2)	9+ 71 = 80	155,000	199,500
(Na,Cl)	23+ 35.5= 58.5	97,900	96,600
(Li,Cl)	7+ 35.5= 42.5	93,900	102,300
(Sr,Cl^2)	87+ 71 =158	184,700	195,850
(Ca,Cl^2)	40+ 71 =111	169,900	187,400
(N,H^4,Cl)	14+4+ 35.5= 63.5	76,800	72,800
(Mg,Cl^2)	24+ 71 = 95	151,200	187,100
(S,Cl_2)	28+142 =170	128,800 (gas)	....
(Al,Cl^3)	27+106.5=133.5	161,800	238,100
(Mn,Cl^2)	55+ 71 =126	112,000	128,000
(Zn,Cl^2)	65+ 71 =136	97,400	113,000

TABLE 38.—HEAT OF FORMATION OF CHLORIDES—(*Continued*)

Formula	Molecular weights	Molecular heat of formation	In dilute solution
(Tl,Cl)	204+ 35.5=239.5	48,600	38,400
(Cd,Cl^2)	112+ 71 =183	93,700	96,400
(Pb,Cl^2)	207+ 71 =278	83,900	77,900
(Fe,Cl^2)	56+ 71 =127	82,200	100,100
(Sn,Cl^2)	118+ 71 =189	80,900	....
(Co,Cl^2)	59+ 71 =130	76,700	95,000
(Ni,Cl^2)	58.5+ 71 =129.5	74,700	93,900
(Cu,Cl)	63.5+ 35.5= 99	35,400	....
(Sn,Cl^4)	118+142 =260	129,800 (liquid)	....
(Fe,Cl^3)	56+106.5=162.5	96,150	127,850
(Hg,Cl)	200+ 35 5=235 5	31,320	....
(Sb,Cl^3)	120+106.5=226.5	91,400	....
(Bi,Cl^3)	208+106.5=314.5	90,800	....
(B,Cl^3)	11+106.5=117.5	89,100 (gas)	....
(Ag,Cl)	108+ 35 5=143.5	29,000	....
(Hg,Cl^2)	200+ 71 =271	53,300	50,300
(Cu,Cl^2)	63.6+ 71 =134.6	51,400	62,500
(As,Cl^3)	75+106.5=181.5	71,500	....
(H,Cl)	1+ 35.5= 36.5	22,000	39,400
(Sb,Cl^5)	120+177.5=297.5	104,500 (liquid)	....
(Pd,Cl^2)	106+ 71 =177	40,500	....
(Pt,Cl^4)	195+142 =337	60,200	70,800
(Au,Cl^3)	197+106.5=303.5	22,800	27,200
(Au,Cl)	197+ 35.5=232.5	5,800	....

69. **Bromides.**—Bromides resemble chlorides. As seen from their heats of formation, they are weaker compounds than chlorides and therefore more readily decomposed. They occur sparingly native (AgBr) and are not often formed in metallurgical operations ($AuBr_3$, but not $PtBr_x$). At ordinary temperature they are solid and mostly white, they are all fusible and volatile; a higher bromide is more readily volatilized, as a rule, than a lower bromide; bromides are less soluble in H_2O than chlorides; some oxybromides are known.

FORMATION OF BROMIDES.—Metallic bromides are formed:

(1) By the action of Br or HBr upon a metal, metallic oxide or carbonate:

$$Ag_2O+HBr=2AgBr+H_2O.$$

(2) By the action of Br upon a metallic oxide in the presence of C:

$$Al_2O_3+6Br+3C=Al_2Br_6+3CO.$$

(3) By precipitating a metal salt with an alkali bromide.

DECOMPOSITION OF BROMIDES.—They are decomposed by similar means as are chlorides; Cl always decomposes a bromide.

TABLE 39.—HEAT OF FORMATION OF BROMIDES[1]

Formula	Molecular weights	Molecular heat of formation, liq. Br.	In dilute solution
(Al,Br_3)	$27+240=467$	120,600	207,500
(Zn,Br_2)	$65+160=325$	78,200	93,200
(Cd,Br_2)	$112+160=272$	76,200	77,200
(Pb,Br_2)	$207+160=367$	69,000	59,000
(Sn,Br_2)	$118+160=278$	63,000	....
(Cu,Br_2)	$63+80=143$	26,000	....
(Sn,Br_4)	$118+320=438$	101,400 solid 98,400 liquid	118,000
(Hg,Br)	$200+80=280$	24,500	....
(Ag,Br) cryst.	$108+80=188$	23,700	....
(Sb,Br_3)	$120+240=360$	64,900	....
(Cu,Br_2)	$63+160=223$	34,800	53,000
(Pt,Br_4)	$195+320=515$	42,400	52,200
(Au,Br_3)	$197+240=437$	12,100	8,400
(Au,Br_2)	$197+160=357$	1,000	....

70. Iodides.—Iodides are very similar to bromides, but are weaker compounds as seen by their lower heats of formation. They are as rare as bromides (AgI) and are seldom formed in metallurgical operations (ZnI_2, AgI). At ordinary temperature metallic iodides are solid; they show strong and varied colors, but are without luster; all are fusible and volatile, they are less volatile and more readily soluble in H_2O than the corresponding chlorides and bromides. Some iodides, *e.g.*, KI, ZnI_2, have the property of dissolving I with evolution of heat showing that a higher iodide is being formed ($KI+I_2=KI_3$). Many metallic iodides dissolve freely in solutions of alkali iodides.

FORMATION OF IODIDES.—Metallic iodides are formed and decomposed in a manner similar to that of bromides. Heating with access of air does not give oxyiodides. Bromine will always decompose an iodide.

TABLE 40.—HEAT OF FORMATION OF IODIDES[2]

Formula	Molecular weights	Molecular heat of formation, sol. I	In dilute solution
(Zn,I_2)	$65+254=319$	49,200	60,600
(Cd,I_2)	$112+254=366$	45,000	44,000
(Pb,I_2)	$207+254=461$	42,000	....
(Cu,I_2)	$63.5+127=318$	16,500	....
(Hg,I_2)	$200+127=454$	14,200	....
(Ag,I) crys.	$108+127=235$	14,300	....
(Hg,I_2) red	$200+254=454$	24,300	....
(Sb,I_3)	$120+381=501$	29,200	....
(Au,I)	$197+127=324$	$\mp 55,000$	....

[1] Ditte, A., "Leçons sur les Métaux," supplement, p. 35.
[2] *Op. cit.*, p. 37.

71. **Cyanides.**—The action of CN upon metals is very similar to that of Cl, Br and I, only it is even weaker than that of I. Cyanides do not occur native. They are extensively used in silver, gold and copper plating; they have become of metallurgical importance only since the application of alkaline cyanide solutions to the leaching of silver and gold ores. Cyanides have a strong tendency to combine with one another and form double salts. At ordinary temperature metallic cyanides are solid and show various colors ranging from the colorless alkali and alkaline earth cyanides to the brilliant colors of the double iron salts. Alkaline cyanides fuse readily at a red heat and remain unchanged as long as air or moisture is excluded. Alkali and alkaline-earth cyanides are soluble in H_2O, and the solutions give an alkaline reaction even in the presence of free HCN; the single heavy-metal cyanides are insoluble, excepting $Hg(CN)_2$.

FORMATION OF CYANIDES.—Chemical treatises give the different methods by means of which cyanides are prepared. In metallurgical operations the start is made with KCN or NaCN or with a mixture of the two. In dissolving metals (insoluble Sn, Hg, Pt) in dilute solutions of alkaline cyanides, the presence of O may be necessary (Cd, Ag, Au) or not (Zn, Fe, Ni, Co, Mn, Pb):

$$4KCN+2Au+H_2O+O=2KAu(CN)_2+2KOH;$$
$$4KCN+Zn+H_2O=K_2Zn(CN)_4+2KOH+H_2.$$

In passing, it may be noted that NaCN contains 53 per cent. CN and KCN only 40 per cent., that NaCN is the cheaper salt, and that it has proved itself in most instances to be as effective as KCN.

DECOMPOSITION OF CYANIDES.—Metallic cyanides are decomposed:

(1) By heat alone. This is the case with all heavy-metal cyanides; most of them are converted into N and a mixture of metal and C, a few (Pd, Hg, Ag) into CN and metal.

(2) By heating with access of air. Alkaline cyanides are first converted into cyanates: $KCN+O=KCNO$; this affinity for O makes them powerful reducing agents at comparatively low temperatures. Upon further heating, the cyanate breaks up: $2KCNO+3O=K_2CO_3+CO_2+2N$. Heavy-metal cyanides readily burn giving metal or metallic oxide, CO_2 and N.

(3) By boiling with water; alkali cyanides give NH_3 and alkali formate ($KCN+2H_2O=NH_3+HCO_2K$), heavy-metal cyanides give CO_2, CO, CN and NH_3, leaving as residue metal or metallic oxide, occasionally some C.

(4) By the action of S in the dry or in the wet way forming a sulphocyanide (rhodonite) soluble in H_2O, excepting the salts of Cu, Pb, Hg and Ag:

$$KCN+S=KCNS;$$
$$(Ag_2, Pb, Hg, Zn, Se, Te)(CN)_2+S_2=(Ag_2, Pb, Hg, Zn, Se, Te)(CN)_2S_2.$$

(5) By the action of Cl with the formation of metallic chloride, CNCl, CN, and a yellow oily substance not fully studied.

(6) By the action of acids. Dilute acids set free HCN : $KCN+HCl=KCl+HCN$. Alkaline cyanide solutions are decomposed even by CO_2: $2KCN+CO_2+H_2O=K_2CO_3+2HCN$. Strong acids carry the decomposition farther:

$$2MetCN + 3H_2SO_4 + 2H_2O = Met_2SO_4 + 2NH_4.H.SO_4 + 2CO;$$
$$MetCN + HNO_3 = MetNO_3,\ N,\ H.$$

(7) By wet electrolytic reduction.

DOUBLE CYANIDES.—Metallic cyanides insoluble in H_2O are dissolved by solutions of alkaline cyanides forming double salts which are almost all soluble in water. There are two kinds. In the first, the alkaline cyanide is decomposed with evolution of HCN and precipitation of the heavy-metal cyanide: $2KCN.AgCN + H_2SO_4 = K_2SO_4 + 2HCN + 2AgCN$ or $2KCN.AgCN + H_2S = 2KCN + 2HCN + Ag_2S$; further, the metallic cyanide first separated may be again decomposed by an excess of acid: $KCN.AgCN + 2HCl = AgCl + 2HCN + KCl$. The evolution of HCN makes this class of salts very poisonous. In the second class, of which $K_4Fe(CN)_6$ may be considered as typical, dilute acids have no decomposing effect and concentrated acids cause no precipitate to form, as the K of the salt is simply replaced by the H of the acid: $K_4Fe(CN)_6 + 2H_2SO_4 = H_4Fe(CN)_6 + 2K_2SO_4$. Some double cyanides occupy an intermediate position: *e.g.*, $Hg(CN)_2.2KCN$.

TABLE 41.—HEAT OF FORMATION OF CYANIDES

Formula	Molecular weights	Molecular heat of formation	In dilute solution
(Ca,C^2,N^2)	40+ 24+ 28= 92	.....	41,650
(K,C,N)	39+ 12+ 14= 65	33,450	30,250
(Na,C,N)	23+ 12+ 14= 49	25,950	25,450
(K,Ag,C^2,N^2)	39+108+ 24+ 28=199	13,700	5,350
(Fe^7,C^{18},N^{18})	392+216+252=860	−256,700	.....
(Zn,C^2,N^2)	65+ 24+ 28=117	− 24,550	.....
(Cd,C^2,N^2)	112+ 24+ 28=164	− 31,850	.....
(Cu,C,N)	63.6+ 12+ 14=89.6	− 20,375	.....
(Pd,C^2,N^2)	106+ 24+ 28=158	− 49,250	.....
(H,C,N)	1+ 12+ 14= 27	− 27,150 (gas)	−21,050
(Hg,C^2,N^2)	200+ 24+ 28=252	− 59,150	.....

TABLE 42.—HEAT OF FORMATION OF CYANATES

Formula	Molecular weights	Molecular heat of formation	In dilute solution
(K,C,N,O)	39+12+14+16= 81	105,850	100,650
(Na,C,N,O)	23+12+14+16= 65	105,050	100,250
(Ag,C,N,O)	108+12+14+16=150	26,450	.....

TABLE 43.—HEAT OF FORMATION OF FERROCYANIDES

Formula	Molecular weights	Molecular heat of formation	In dilute solution
(K^4,Fe,C^6,N^6)........	156+56+72+84=368	157,300	145,300
(H^4,Fe,C^6,N^6)........	4+56+72+84=216	−102,000	−101,500
(K^3,Fe,C^6,N^6)	117+56+72+84=329	129,600	100,800
(H^3,Fe,C^6,N^6)	3+56+72+84=215	....	−127,400

72. **Fluorides.**—Fluorides occur sparingly native (CaF_2; $3NaF.AlF_3$) and are rarely formed in metallurgical processes. F has a great affinity for metals; it readily combines with all of them, and often with the emission of light; it has a strong tendency to form double salts. At ordinary temperature all fluorides are solid excepting SiF_4; in color and luster they resemble the corresponding chlorides. The solid fluorides are all readily fusible; some are volatilized without decomposition (SbF_3, AsF_5, CrF_3, HgF_2, SnF_4, ZnF_2 . . .). The fluorides of the alkalis, of Sn and Ag are soluble in H_2O, the others are sparingly soluble or insoluble.

FORMATION OF FLUORIDES.—Metallic fluorides are formed:

(1) By the action of F upon a metal.

(2) By the action of HF upon a metal, metallic oxide or carbonate.

(3) By heating electronegative metals (Sb) with PbF_2 or HgF_2.

(4) Volatile fluorides are obtained by heating CaF_2, H_2SO_4 and a metallic oxide, or a mixture of CaF_2 and metallic oxide in a current of HCl.

(5) Insoluble fluorides (CaF_2) are obtained by precipitation.

DECOMPOSITION OF FLUORIDES.—Metallic fluorides are decomposed:

(1) By heat alone: AgF, AuF_3, PtF_4.

(2) By heating with access of air in the presence of moisture many fluorides are converted into oxides or oxyfluorides while HF is given off.

(3) By the action of Cl many fluorides are converted into chlorides.

(4) Gently warming with H_2SO_4 gives metallic sulphate and HF.

(5) Electrolytic smelting readily decomposes NaF, CaF_2, AlF_3; experiments with PbF_2, SnF_4, AgF have not been successful.

DOUBLE SALTS.—Double salts are of three kinds:

(1) A union of two separate salts, of which cryolite, $3NaF.AlF_3$, is the best known.

(2) A combination of a fluoride with HF to form an acid salt: KF.HF; $SnF_4.2HF$; $TiF_4.2HF$; $ZnF_4.2HF$; in general $H_2Met.F_6$.

(3) A combination of an acidic oxide dissolved in HF with a basic fluoride; K_3BiF_4; K_2SiF_6; $PbSiF_6$; K_2SnF_6; K_2SbF_5; $KAsF_6$; $NiSnF_6+6H_2O$. Heat generally splits them into component fluorides.

TABLE 44.—HEAT OF FORMATION OF FLUORIDES

Formula	Molecular weights	Molecular heat of formation	In dilute solution
(Sr,F^2)	87+38=125	224,020	....
(Ba,F^2)	137+38=175	224,000	221,500
(Li,F)	7+19= 26	....	116,880
(K,F)	39+19= 58	110,000	113,600
(Ca,F^2)	40+38= 78	216,450	....
(Mg,F^2)	24+38= 62	209,500	....
(Na,F)	23+19= 42	109,720	109,120
(N,H^4,F)	14+4+19= 37	101,250	99,750
(Al,F^3)	27+57= 84	....	275,220
(B,F^3)	11+57= 68	....	219,345
(Mn,F^2)	55+38= 93	....	153,310
(Zn,F^2)	65+38=103	....	138,220
(Si,F^4)	28+76=104	275,920 (gas)	....
(Fe,F^2)	56+38= 94	....	125,220
(Cd,F^2)	112+38=150	....	121,720
(Co,F^2)	59+38= 97	....	120,340
(Ni,F^2)	58.5+38= 96.5	....	118,980
(Fe,F^3)	56+57=113	....	164,940
(Tl,F)	204+19=223	....	54,405
(Pb,F^2)	207+38=245	101,600	....
(H,F)	1+19= 20	38,500 (gas)	50,300
(Sb,F^3)	120+57=177	....	136,680
(Cu,F^2)	63.6+38=101.6	....	88,160
(Ag,F)	108+19=127	22,070	25,470

73. Silicides.[1]—Silicon is not found in the free state, nor are silicides; both are artificial products.

[1] Vigouroux, E., "Le Silicium et Les Siliciures Métalliques," Gauthier-Villars, Paris, 1906 (reprints from *Compt. Rend.*, 1895, CXXI, 771; 1896, CXXII, 318; CXXIII, 115.)

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TABLE 45.—CHEMICAL ANALYSIS OF SILICON[1]

Si	98.10	95.30
Fe	1.20	1.06
Al	0.60	1.54
C	0.10	2.10

The importance of Si was confined to the metallurgy of iron and steel until electric smelting became important in metallurgical operations. Since then silicon-copper and other metallic silicides as well as silicon carbide (carborundum) have become industrially valuable. With the cheap production of Si by Tone this element and some of its compounds will be used more generally, especially as a reducing[2] and desulphurizing agent (alkali-earth silicides). Si melts at 1,430° C., has a specific gravity 2.34, a hardness 6–7, is oxidized by air and CO_2 in the neighborhood of its melting-point. It can be substituted for Al in the thermic reaction of the Goldschmidt process[3] as can some of its compounds, such as CaSi and MgSi. Vigouroux found that Si exists only in two forms (amorphous and crystalline) instead of three (amorphous, graphitoidal and crystalline) as has been held, and that both show the same chemical behavior. Moissan and Smiles[4] believe, however, that they have found an amorphous variety differing from that of Vigouroux.

Silicon metals fall into two classes: (1) Those forming chemical compounds (Mg, Ni, Co, Fe, Cr, Mn, Cu, Pt . . .) Si_2 or Met Si_2 in which the metal is bivalent. When they are decomposed by acid, the Si is oxidized and separates as SiO_2. To these metallic silicides may be added Acheson's carborundum,[5] SiC, used as a refractory material (§165) and a reducing agent. (2) Those which when fused dissolve Si and drop it again more or less upon cooling: Alkaline metals, Zn, Al, Pb, Sn, Sb, Bi, Ag, Au. When they are decomposed by acid the Si separates. Silicides often also dissolve Si in the same manner as do some metals.

Metallic silicides have a metallic luster, show a considerable variety of color, are hard and brittle, are more readily fusible than the component elements, and are stable at elevated temperatures.

FORMATION OF SILICIDES.—Metallic silicides are formed:

(1) By the direct union of Si and metal at an elevated temperature. This

[1] Tone, *Min. Ind.*, 1906, XV, 685.

[2] Neumann, *Stahl u. Eisen*, 1908, XXVIII, 356.

[3] Goldschmidt, *Electrochem. Met. Ind.*, 1908, VI, 244.
Watts-Breckenridge, *op. cit.*, 1908, VI, 237, 260.
Breckenridge, *Bull.* 438, University of Wisc., 1911.

[4] *Eng. Min. J.*, 1903, LXXV, 479.

[5] Fitzgerald, F. A. J., "Carborundum," Knapp, Halle, 1904.
Acheson, *J. Frankl. Inst.*, 1893, CXXXVI, 194, 279.
Mathews, *School Min. Quart.*, 1894–95, XVI, 73.
Blake, *Eng. and Min. J.*, 1893, LVI, 270, 320.
Fitzgerald, *J. Frankl. Inst.*, 1897, CXLIII, 81.

does not always require that the metal be melted; *e.g.*, $CrSi_2$ was prepared by Moissan by heating a mixture of the constituents to 1200° C. when they diffused into one another forming the compound. In the same manner he prepared $FeSi_2$, $NiSi_2$, $CoSi_2$.

(2) By the action of C at an elevated temperature upon metallic silicates or a mixture of SiO_2 and metal or metallic oxide; this is the process of the iron blast furnace or the electric furnace.

(3) By the action of Na at an elevated temperature upon Si-and metal-compounds: $MnSi_2$ from MnF_3, Na_2SiO_3, $3NaF.AlF_3$, Na; or $CuSi_2$ from K_2SiF_6, Cu, Na.

(4) By precipitation with SiH_4, *e.g.*, $CuSi_2$ from $CuSO_4$ or $AgSi_4$ and Ag from $AgNO_3$.

DECOMPOSITION OF SILICIDES.—Metallic silicides are decomposed:

(1) By an oxidizing fusion, *e.g.*, all processes for the production of steel or wrought iron from pig iron.

(2) By treating with acids; thus aqua regia and HF are effective, while HCl, HNO_3 and H_2SO_4 are not.

TABLE 46.—HEAT OF FORMATION OF SILICIDES

Formula	Molecular weights	Molecular heat of formation	In dilute solution
(Mn^7,Si^2)..................	385+56=441	47,400	....
(H^4,Si).....................	4+28= 32	−6,700 (gas)	....

74. Carbides.[1]—Metallic carbides do not occur native. Up to the time of the application of the electric arc to smelting, carburized iron alone was of industrial importance; since then many metal-carbon compounds have been prepared, and some have found application in the arts. A metal acts upon C in two different ways: (1) It forms a chemical compound with C; (2) it dissolves C when melted and releases it again upon cooling, either entirely or it retains a part in solid solution. The following are some of the chemical compounds known at present: M_3C(Fe,Mn); M_2C(Mo,W); M_3C_2(Cr); MC(Zr,Si,V); M_4C_3(Gl,Al); M_2C_3(Ur,Cr); MC_2(Ba,Sr,Ca,Li_2,Ce,La,Y,Th,Zr); MC_4(Cr); MC_{4x}(Ni,Co,Ta,Nb), readily carburized, but definite compounds have not yet been isolated. Besides these single carbides, binary carbides of Fe–Mn, Fe–Cr, Fe–Mo, Fe–W, Cr–W, and ternary of Fe–Cr–Mo, Fe–Cr–W have been isolated.[2] Of the cyanonitrides $Nb_5C_3N_2$, $TiCN_4$ and $Ta_{19}C_{18}N$

[1] Moissan, *Ann. chim. phys.*, 1896, IX, 247, 289; *Compt. Rend.*, 1896, CXXII, 274; *Min. Ind.*, 1896, V, 468, *et seq.*

Moissan, "The Electric Furnace."

Delépine, M., "Les Carbures Métalliques," Joanin & Co., Paris, 1904.

Pring, *Electrochem. Met. Ind.*, 1909, VII, 121.

[2] Delépine, *op. cit.*

may be mentioned. The metals dissolving and releasing C are Ag, (Zn, Cd), Rh,Pd,Ir,Pt; many carbides have a similar property. The metals which do not dissolve any C are Au.Bi.(Pb).Sn.(Al). Information about the metals placed in parentheses is doubtful. The solubility of C in a metal increases with the temperature, but is hindered by the dissociation of a possible carbide, if the temperature rises above a certain point. Thus Na_2C_2, CaC_2, Li_2C_2, Mn_3C, SiC are decomposed, if the heat is sufficiently raised. Metallic carbides have been divided into two classes: 1. Those that decompose water; 2. those that do not. To the former belong the alkali and alkali-earth carbides, also the carbides of Al, Ce, La, Y, Th, Mn, Ur; to the latter the carbides of O, S, Cl, Br, I, F, Si, B. The carbides of W, Va, Cr, Mo and Fe occupy an intermediary position; Fe_3C and Cr_2C_3, decomposed by dil. HCl, lean toward the water-decomposing carbides; W_2C and VC, attacked by HNO_3, and SiC with BC, not attacked by acids, toward the other end of the series.

Water at ordinary temperature acts upon carbides in three different ways: (1) $CaC_2+2H_2O=Ca(OH)_2+C_2H_2$ (acetylene). This group includes the alkali and alkali-earth carbides. (2) $Al_4C_3+12H_2O=4Al(OH)_3+3CH_4$ (methane), represented by Al_4C_3 and Be_2C. (3) Met. $C+H_2O=$ met. compound $+$ C_2H_2, CH_4, C_2H_4 (ethylene), H in varying proportions: Mn_3C, CeC_2, LaC_2, YC_2, ThC_2, Ur_2C_3. At elevated temperatures[1] some carbides are decomposed by H-compounds with the liberation of C and H, viz., $CaC_2+H_2O=CaO+2C+2H$; $CaC_2+2HCl=CaCl_2+2C+2H$; $CaC_2+2NH_3=CN_2Ca+C+6H$.

Omitting alkali and alkali-earth carbides, metallic carbides resemble in color the metal of the compound. They are crystalline, have a metallic luster, are very hard; their specific gravity is lower than that of the metals. Some are attracted by the magnet, others not; their melting-points are high, *e.g.*, Fe_3C 1,837°; Fe_2C 2,220°; UC 2,425°; V_4C_3 2,750° C.[2]

FORMATION OF CARBIDES.—Metallic carbides are formed:

(1) By the direct union of C and fused metal or metal brought to a cementation temperature.

(2) By the reduction of metallic oxide with C or carbonaceous matter in the absence or presence of fluxes.

DECOMPOSITION OF CARBIDES.—They are decomposed:

(1) By heating with access of air: conversion of pig iron into steel or wrought iron.

(2) By heating with substances which readily give off O (malleable castings).

(3) By heating with a metal or metallic compound that has a greater affinity for C, *e.g.*, Ca or CaO[3] for the decarburization of ferrochrome.

(4) By adding an element that decreases the solubility of C. Thus Si, Al and S lower the saturation point of Fe for C.

[1] Hahn-Streetz, *Metallurgie*, 1906, III, 727.

[2] Ruff, *Zt. angew. Chem.*, 1911, XXIV, 1134; *Metallurgie*, 1911, VIII, 456.

[3] Pratt, *Engineering*, 1907, LXXXIV, 326.

Perkin, *Tr. Faraday Soc.*, 1907, III, 179.

TABLE 47.—HEAT OF FORMATION OF CARBIDES

Formula	Molecular weights	Molecular heat of formation	In dilute solution
(Al^4,C^3)	108+36=144	232,000	
(Mn,C^2)	55+24= 79	114,400 (Ponthiere)	
(Mn,C^3)	55+36= 91	9,900 (Berthelot)	
(Fe^3,C)	168+12=180	8,460	
(Ca,C^2)	40+24= 64	−6,250	+131,500[1]
(Na,C)	23+12= 35	−4,400	
(Li,C)	7+12= 19	−5,750	
(N^2,C^2)	28+24= 52	−73,000 (gas)	−67,100
(Ag,C)	108+12=120	−43,575	

75. Metallic Oxygen-salts.[2]—Their general properties depend upon the character of both the acid and the base.

FORMATION OF OXY-SALTS.—They are formed: (1) By the oxidation of S-, Se-, Te-, P-, As-, Sb-, Si- . . . ides.

(2) By the action of an oxygen-acid upon a metal, metallic oxide, Se-, Te-, P-, As-, Sb-, Si-, ide.

DECOMPOSITION OF OXY-SALTS.—They are decomposed in the *dry* way:

(1) By heat alone; (2) by heating with C; (3) by heating with S- ides; (4) by heating with Cl.

In the *wet* way they are decomposed: (1) By the action of H_2S or an alkali sulphide. (2) By the action of compounds which are readily peroxidized.

In the *combined dry and wet way* they are decomposed: (1) By being brought into contact with an acid or a base that has a stronger affinity and thus forms a new compound, the acid or base of the new salt being set free. (2) By double decomposition. (3) By precipitation through a metal. (4) By electrolysis.

76. Carbonates ($Met.^{II}CO_3$).—Alkali carbonates are readily soluble in H_2O, and not decomposed by heat. Other carbonates are insoluble in H_2O, readily soluble in acids, easily decomposed by heat ($Met.CO_3=Met.\ O+CO_2$), by heating in a current of steam ($Met.\ CO_3+H_2O+heat=H_2Met.O_2+CO_2$), by treatment with stronger acids $Met.CO_3+2HCl=Met.Cl_2+CO_2+H_2O$, by heating stable carbonates with C: $Met.CO_3+C=Met.O+2CO$; $Met.CO_3+3C=Met.+3CO$.

Friedrich-Smith[3] determined the decomposition temperatures in an electrically heated tube closed at one end, *i.e.*, in the atmosphere of CO_2 set free by dissociation. Their temperatures are: cerussite, $PbCO_3$, 315° C.; smithsonite, $ZnCO_3$, 395°; siderite, $FeCO_3$, 380–415°; hydrocerussite, probably $2PbCO_3$, $Pb(OH)_2$, 430°; rhodocrosite, $MnCO_3$, 525°; magnesite, $MgCO_3$, 570°; dolomite, $CaMgC_2O_6$, 895°; aragonite, $CaCO_3$, 895°; calcite, $CaCO_3$, 895°; strontianite,

[1] Guntz-Bassett, *J. Chim. Phys.*, 1906, IV, 1; *Rev. Mét. Extr.*, 1906, III, 667.

[2] Ditte, A., "Etude Générale des Sels," Dunod, Paris, 1906, 2 vols.

[3] *Stahl-Eisen*, 1911, XXXI, 1909; *Metallurgie*, 1912, IX, 409.

$SrCO_3$, over 1,100° C. The data are higher than those that will be obtained if the decomposition is carried on in a current of air.

Some other dissociation temperatures at atmospheric pressure are subjoined:[1]

TABLE 48.—HEAT OF FORMATION OF CARBONATES

Formula	Molecular weights	Molecular heat of formation	In dilute solution
(Ba,C,O^3)	137+12+48=197	286,300	....
(K^2,C,O^3)	78+12+48=138	282,100	288,600
(Sr,C,O^3)	87+12+48=147	281,400	....
(Ca,C,O^3)	40+12+48=100	273,850	....
(Na^2,C,O^3)	46+12+48=106	273,700	270,300
(Mg,C,O^3)	24+12+48= 88	269,900	....
(Mn,C,O^3)	55+12+48=115	210,300	....
(Zn,C,O^3)	65+12+48=125	197,500	....
(Fe,C,O^3)	56+12+48=116	187,800	....
(Cd,C,O^3)	112+12+48=172	183,200	....
(Pb,C,O^3)	207+12+48=267	170,000	....
(Cu,C,O^3)	63.6+12+48=123.6	146,100	....
(Ag^2,C,O^3)	216+12+48=276	123,800	....

TABLE 49.—HEAT OF FORMATION OF BICARBONATES

Formula	Molecular weights	Molecular heat of formation	In dilute solution
(K,H,C,O^3)	39+1+12+48=100	233,300	228,000
(Na,H,C,O^3)	23+1+12+48= 84	227,000	222,700
(N,H^5,C,O^3)	14+5+12+48= 79	208,600	202,300

77. Sulphates ($Met.^{II}SO_4$).[2]—Most metallic sulphates are readily soluble in H_2O, some are only sparingly so (Ag_2SO_4), others are insoluble ($PbSO_4$). All metallic sulphates are completely decomposed by heat, with the exception of those of Pb and Bi, but the temperatures at which decompositions take place vary with the metals. Some metallic sulphates are converted directly into oxides, others first form basic salts. Alkali-earth sulphates show a behavior similar to

[1] $CaCO_3$: 812° (Le Chatelier, *Comp. Rend.*, 1883, CII, 1243); 825° (Brill, *Zt. anorg. Chem.*, 1905, XLV, 275); 908° (Riesenfeld, *J. chim. phys.*, 1909, VII, 561); 910° (Zavrieff, *Compt. Rend.*, 1907, CXLV, 428).

$MgCO_3$: 445° (Brill, *Zt. anorg. Chem.*, 1905, XLV, 283).

$SrCO_3$: 820° (Le Chatelier, *Bull. Soc. Chim.*, 1887, XLVII, 100.

$BaCO_3$: 1,352° (Finkelstein, *Ber. deutsch. chem. Ges.*, 1906, XLV, 275).

$NiCO_3$: >100°; $CdCO_3$: >120°; $ZnCO_3$: 137° (Doeltz-Graumann, *Metallurgie*, 1906, III, 473.)

Ag_2CO_3:>200°; $MnCO_3$: about 320° (Joulin, *Ann. chim. Phys.*, 1873, XXX, 278); $FeCO_3$: about 700° C.

[2] Hofman-Wanjukow, *Tr. A. I. M. E.*, 1913, XLIII, 523.

that of metallic sulphates, while alkali sulphates do not part with their sulphur trioxide even at elevated roasting-temperatures. Normal sulphates which do not form basic salts upon dissociation show both SO_2 and SO_3 in the gaseous product; normal sulphates which form basic salts upon dissociation evolve only SO_2 when they pass from the normal to the basic state; basic sulphates upon dissociation into MO set free only SO_3; metallic sulphates with bases M_2O_3 and M_2O do not form basic sulphates upon dissociation; the former set free SO_3, the latter SO_2.

The electric conductivity of $FeSO_4$ and $CuSO_4$ has been studied by Somerville.[1]

All sulphates are reduced to sulphides by heating with C at high temperatures: $Met.SO_4+2C=Met.S+2CO_2$; at a low temperature (excepting $PbSO_4$) the following reaction takes place: $2Met.SO_4+C=2Met.O+2SO_2+CO_2$. Heating with SiO_2 decomposes $PbSO_4$, viz., $2PbSO_4+SiO_2=Pb_2SiO_4+2SO_3$, while in wet processes H_2SO_4 decomposes silicates. Many sulphates are decomposed by heating in a stream of HCl, giving chlorides.

TABLE 50.—HEAT OF FORMATION OF SULPHATES

Formula	Molecular weights	Molecular heat of formation	In dilute solution
(K^2,S,O^4)	$78+32+64=174$	344,300	337,700
(Ba,S,O^4)	$137+32+64=233$	339,400	....
$(Li,^2,S,O^4)$	$14+32+64=110$	333,500	339,600
(Sr,S,O^4)	$87+32+64=183$	330,200	....
(Na^2,S,O^4)	$46+32+64=142$	328,100	328,500
(Ca,S,O^4)	$40+32+64=136$	317,400	321,800
(Mg,S,O^4)	$24+32+64=120$	300,900	321,100
(Al^2,S^3,O^{12})	$54+96+192=342$	....	879,700
(N^2,H^8,S,O^4)	$28+8+32+64=132$	283,500	281,100
(Mn,S,O^4)	$55+32+64=151$	249,400	263,200
(Zn,S,O^4)	$65+32+64=161$	229,600	248,000
(Fe,S,O^4)	$56+32+64=152$	....	234,900
(Co,S,O^4)	$59+32+64=155$	....	228,900
(Ni,S,O^4)	$58.5+32+64=154.5$	....	228,700
(Fe^2,S^3,O^{12})	$112+96+192=400$	....	650,500
(Tl^2,S,O^4)	$408+32+64=504$	221,800	213,500
(Cd,S,O^4)	$112+32+64=208$	219,900	231,600
(Pb,S,O^4)	$207+32+64=303$	215,700	....
(II^2,S,O^4)	$2+32+64=98$	192,200	210,200
(Cu,S,O^4)	$63.6+32+64=159.6$	181,700	197,500
(Hg^2,S,O^4)	$400+32+64=496$	175,000	....
(Ag^2,S,O^1)	$216+32+64=312$	167,100	162,600
(Hg,S,O^4)	$200+32+64=296$	165,100	....

[1] *Met. Chem. Eng.*, 1912, x, 682.

TABLE 51.—HEAT OF FORMATION OF BI-SULPHATES

Formula	Molecular weights	Molecular heat of formation	In dilute solution
(K,H,S,O^4)	$39+1+32+64=136$	276,100	272,900
(Na,H,S,O^4)	$23+1+32+64=120$	269,100	268,300
(N,H^5,S,O^4)	$14+5+32+64=115$	244,600	245,100
(H,H,S,O^4)	$1+1+32+64=98$	192,200	210,200

TABLE 52.—DEHYDRATION OF METALLIC SULPHATES

	Temperature of beginning of dehydration	Product formed	Remarks
	Degrees C.		
$FeSO_4+7H_2O$	21	$FeSO_4+4H_2O$	Slight apple-green.
$FeSO_4+4H_2O$	80	$FeSO_4+H_2O$	White.
$FeSO_4+H_2O$	406	$Fe_2O_3+2SO_3$	Yellowish-brown.
$Al_2(SO_4)_3+16H_2O$	51	$Al_2(SO_4)_3+13H_2O$	White.
$Al_2(SO_4)_3+13H_2O$	82	$Al_2(SO_4)_3+10H_2O$	White.
$Al_2(SO_4)_3+10H_2O$	97	$Al_2(SO_4)_3+7H_2O$	White.
$Al_2(SO_4)_3+7H_2O$	109	$Al_2(SO_4)_3+4H_2O$	White.
$Al_2(SO_4)_3+4H_2O$	180	$Al_2(SO_4)_3+H_2O$	White.
$Al_2(SO_4)_3+H_2O$	316	$Al_2(SO_4)_3$	White.
$CuSO_4+5H_2O$	27	$CuSO_4+3H_2O$	Sky-blue.
$CuSO_4+3H_2O$	93	$CuSO_4+H_2O$	Pale blue.
$CuSO_4+H_2O$	155	$CuSO_4$	White.
$MnSO_4+5H_2O$	25	$MnSO_4+2H_2O$	Pale peachblossom.
$MnSO_4+2H_2O$	60	$MnSO_4+H_2O$	Paler than preceding.
$MnSO_4+H_2O$	152	$MnSO_4$	Paler than preceding.
$ZnSO_4+7H_2O$	25	$ZnSO_4+6H_2O$	White.
$ZnSO_4+6H_2O$	28	$ZnSO_4+2H_2O$	White, granular.
$ZnSO_4+2H_2O$	115	$ZnSO_4+H_2O$	White.
$ZnSO_4+H_2O$	225	$ZnSO_4$	White.
$NiSO_4+7H_2O$	40	$NiSO_4+4H_2O$	Green.
$NiSO_4+4H_2O$	106	$NiSO_4+H_2O$	Yellow.
$NiSO_4+H_2O$	279	$NiSO_4$	Orange colored.
$CoSO_4+7H_2O$	19	$CoSO_4+4H_2O$	Rose.
$CoSO_4+4H_2O$	58	$CoSO_4+H_2O$	Lilac.
$CoSO_4+H_2O$	276	$CoSO_4$	Lilac.
$CdSO_4+\frac{8}{3}H_2O$	30	$CdSO_4+2H_2O$	White.
$CdSO_4+2H_2O$	41	$CdSO_4+H_2O$	White.
$CdSO_4+H_2O$	170	$CdSO_4$	White.
$MgSO_4+7H_2O$	19	$MgSO_4+6H_2O$	White.
$MgSO_4+6H_2O$	38	$MgSO_4+2H_2O$	White.
$MgSO_4+2H_2O$	112	$MgSO_4+H_2O$	White.
$MgSO_4+H_2O$	203	$MgSO_4$	White.
$CaSO_4+2H_2O$	38	$CaSO_4+H_2O$	White.
$2CaSO_4+2H_2O$	80	$2CaSO_4+H_2O$	White.
$2CaSO_4+H_2O$	149	$2CaSO_4$	White.

TABLE 53.—DESULPHATIZATION OF ANHYDROUS METALLIC SULPHATES

Metallic sulphates	Temperature of beginning of decomposition	Temperature of energetic decomposition	Products of decomposition	Remarks
	Degrees C.	Degrees C.		
$FeSO_4$	167	480	$Fe_2O_3.2SO_3$	Yellow-brown.
$Fe_2O_3.2SO_3$	492	560	Fe_2O_3	Red.
$Bi_2(SO_4)_3$	570	639	$5Bi_2O_3.4(SO_3)_3$	White.
$Al_2(SO_4)_3$	590	639	Al_2O_3	White.
$PbSO_4$	637	705	$6PbO.5SO_3$	White.
$CuSO_4$	653	670	$2CuO.SO_3$	Orange-color.
$MnSO_4$	699	790	Mn_3O_4	Dark red to black.
$ZnSO_4$	702	720	$3ZnO.2SO_3$	White, cold and hot.
$2CuO.SO_3$	702	736	CuO	Black.
$NiSO_4$	702	764	NiO	Brownish-green.
$CoSO_4$	720	770	CoO	Brown to black.
$3ZnO.2SO_3$	755	767	ZnO	Hot yellow, cold white.
$CdSO_4$	827	846	$5CdO.SO_3$	White.
$5Bi_2O_3.4(SO_3)_3$	870	890	Bi_2O_3(?)	Yellow.
$5CdO.SO_3$	878	890	CdO	Black.
$MgSO_4$	890	972	MgO	White.
Ag_2SO_4	917	925	Ag	Silver white.
$6PbO.5SO_3$	952	962	$2PbO.SO_3$(?)	White to yellow.
$CaSO_4$	1,200	....	CaO	White.
$BaSO_4$	1,510	....	BaO	White.

78. Sulphites ($Met.^{II}SO_3$).—These are mostly insoluble in H_2O; on exposure to moist air they are slowly oxidized to sulphates; oxidizing agents (Cl, HNO_3) do it more quickly. Sulphites are decomposed at a red heat forming either sulphide and sulphate ($4Met.\ SO_3 = Met.\ S + 3MSO_4$) or oxide and SO_2, viz., $MSO_3 = MO + SO_2$; heating with C reduces them to sulphide or oxide; they are decomposed by all acids excepting CO_2. Sulphite solutions reacting with S, H_2S, alkali hydrosulphides form hyposulphites; upon heating with SO_2 they give sulphates and hyposulphites.

TABLE 54.—HEAT OF FORMATION OF SULPHITES

Molecular formula	Molecular weights	Molecular heat of formation	In dilute solutions
K_2,S,O_3	$32+48+78=158$	....	272,600
Na_2,S,O_3	$32+48+46=126$	....	261,000

79. Hyposulphites ($Met.^{II}S_2O_3$).—Hyposulphites of the alkalis and alkali earths are readily soluble in H_2O; they are effective solvents for some halides ($AgCl$, $AgBr$, AgI); from these solutions alkali and alkali-earth sulphides pre-

cipitate metallic sulphides. Hyposulphites have a tendency to form double salts; heat decomposes hyposulphites: $4Na_2S_2O_3=3Na_2SO_4+Na_2S_5$; boiling of aqueous solutions evolves H_2S and forms sulphate, or sulphide if the metal in combination with the $H_2S_2O_3$ is precipitated by the H_2S. Hyposulphites are converted into tetrathionates ($H_2S_4O_6$) by I, into sulphates by HNO_3, HCl, $KClO_3$; a mineral acid, gives S and SO_2.

80. **Selenites (Met.$^{II}SeO_3$) and Selenates (Met.$^{II}SeO_4$).**—Selenites are insoluble or sparingly soluble in H_2O; are difficult to decompose by heat alone, are reduced by C to selenides or converted into metallic oxide and Se; they are decomposed by sulphuric and arsenic acids. Acid and basic salts are known.

Selenates resemble sulphates very much in their general properties; most of them are, however, insoluble in H_2O; they can stand a red heat without being decomposed; they are reduced to selenides by heating with C or Na_2CO_3, or by heating in a current of H.

81. **Tellurites (Met. $^{II}TeO_3$) and Tellurates (Met.$^{II}TeO_4$).**—These resemble very much selenites and selenates; tellurites are mostly fusible; tellurates when heated give off O and are changed into tellurites.

82. **Phosphates (Met.$^{II}_3$ P_2O_8).**[1]—Phosphates are as a rule insoluble in H_2O and not decomposed by heat. Many are fusible; most of them are reduced to phosphides by heating with C; they are not decomposed by heating with SiO_2.

TABLE 55.—HEAT OF FORMATION OF PHOSPHATES

Formula	Molecular weights	Molecular heat of formation	In dilute solution
(Ca^3,P^2,O^8)...................	120+62+128=310	919,200	....
(Mg^3,P^2,O^8)...................	72+62+128=262	910,600	....
(Na^3,P,O^4)...................	69+31+ 64=164	452,400	....

83. **Arsenates (Met.$^{II}_3$ As_2O_8) and Arsenites (Met.$^{II}As_2O_4$).**—Arsenates are isomorphous with phosphates; they are insoluble in H_2O, but more or less soluble in acids; many are fusible; they are incompletely, if at all, decomposed by heat alone; heating with C reduces them. Arsenites are very unstable compounds. They are insoluble in H_2O; are decomposed by heating, thus: Met. As_2O_4=Met. $O+As_2O_3$; $5As_2O_3=3As_2O_5+As_2$; are converted into arsenates by roasting; are reduced to arsenides by heating with C.

84. **Antimonates (Met.$^{II}Sb_2O_6$) and Antimonites (Met.$^{II}_3$ Sb_2O_6).**—These resemble and are similar to, the corresponding arsenic compounds.

85. **Nitrates (Met.$^{II}N_2O_6$).**—All are soluble in H_2O, and as a rule insoluble in conc. HNO_3; all are decomposed by heating at comparatively low temperatures as well as by warming with H_2SO_4.

[1] Basic Bessemer Slag: Matweieff., *Rev. Mét.*, 1910, VII, 848.
Blome, *Metallurgie*, 1910, VII, 659.
Kroll, *J. I. and St. I.*, 1911, II, 126.
Steinweg, *Metallurgie*, 1912, IX, 28.

TABLE 56.—HEAT OF FORMATION OF NITRATES

Formula	Molecular weights	Molecular heat of formation	In dilute solution
(K,N,O^3)	$39+14+48=101$	119,000	110,700
(Na,N,O^3)	$23+14+48=85$	110,700	106,000
(Zn,N^2,O^6)	$65+28+96=187$	.....	131,700
(Pb,N^2,O^6)	$207+28+96=331$	105,400	98,200
(Cu,N^2,O^6)	$63.5+28+96=187.5$	.....	81,300
(H,N,O^3)	$1+14+48=63$	34,400 (gas)	48,800
(Ag,N,O^3)	$108+14+48=170$	28,700	23,000

86. Silicates (Met. Si_xO_y) (See Slags, §191).—Metallic silicates are insoluble in H_2O, and melt at high temperatures. The slagged metal cannot be reduced by CO; it requires contact with solid C, and the reduction is imperfect unless another body takes the place of the metal. Some silicates are completely decomposed by strong acids, others are not. Liquid slags that have been chilled in water, having retained their character of solutions and become solid solutions, are more readily acted upon by acids than slags which have been allowed to cool slowly, giving the mineralogical entities the necessary time to form and to crystallize.

TABLE 57.—HEAT OF FORMATION OF SILICATES

Starting from	g-cal. developed per g-mol.	g-cal. developed per 1 g. silicate formed	Starting from	g-cal developed per g-mol.	g-cal. developed per 1 g. silicate formed	Reference
(FeO,SiO_2)	10,600	80	(Fe,Si,O_3)	254,600	1,929	Le Chatelier, *Compt. Rend.*, 1895, CXX, 623.
(MnO,SiO_2)	5,400	41	(Mn,Si,O_3)	276,300	2,109	Le Chatelier, *Compt. Rend.*, 1896, CXXII, 80.
(BaO,SiO_2)	14,700	69	$(Ba,SiO,_3)$	328,100	1,540	Tschernobaeff, *Rev. Mét.*, 1905, II, 729; *Electro-chem. and Met. Ind.*, 1906, IV, 72.
(CaO,SiO_2)	17,850	154	$(Ca,Si,O)_3$	329,350	2,839	
$(2CaO,SiO_2)$	28,300	165	(Ca_2,Si,O_4)	471,300	2,740	
$(3CaO,SiO_2)$	28,550	125	(Ca_3,Si,O_5)	603,050	2,645	
(SrO,SiO_2)	17,900	110	(Sr,Si,O_3)	329,100	2,019	
$(Al_2O_3,S2iO_2)$	14,900	67	(Al_2,Si_2,O_7)	767,500	3,457	
$(3CaO,Al_2O_3,2SiO_2)$	33,500	86	(Ca_3,Al_2,Si_2,O_{10})	1195,550	3,065	
$(2H_2O,Al_2O_3,2SiO_2)$	43,800	170	(H_4,Al_2,Si_2,O_9)	927,420	3,595	
(Li_2O,SiO_2)	65,100	720	(Li_2Si,O_3)	347,100	3,856	
(Na_2O,SiO_2)	45,200	370	(Na_2,Si,O_3)	326,100	2,673	
(CaO,Al_2O_3)	450	3	(Ca,Al_2O_4)	524,550	3,220	
$(2CaO,Al_2O_3)$	3,300	15	(Ca_2,Al_2O_5)	658,900	3,079	
$(3CaO,Al_2O_3)$	2,950	11	$(Ca_3,Al_2O_6,)$	789,050	2,922	
(SiO_2 35.5, FeO 39.7 MnO 1.0, CaO 11.4, MgO 2.7, Al_2O_3 9.2, Cu 0.42, S 0.42).	....	133	....	....	....	Richards, *Electrochem. and Met. Ind.*, 1907, V, 266.
$(2FeO,SiO_2)$ (FeO 70.80, SiO_2 29.20).	22,236	109	(Fe_2, Si, O_4)	333,636	1,637	Hofman-Wen. *Tr. A. I. M. E.*, 1910, XLI, 495.
(FeO 57.58, CaO 12.00, SiO_2 30.42).	....	140	....	....	....	Hofman-Wen.
(FeO 40.30, CaO 28.00, SiO_2 31.70).	....	193	....	....	....	Hofman-Wen.

86a. Heats of Formation and Calorific Values of Hydrocarbons.—In order to bring together in this chapter the leading calorific values, the heat formations of hydrocarbons and their calorific powers are given in Table 58.

TABLE 58.—HEAT OF FORMATION OF HYDROCARBONS

Name	Formula	Molecular weights	Molecular heat of formation.	Heat of combustion, Molecular (Cal.)	Heat of combustion, 1 cbm., (Cal.)
Methane (marsh gas)......	(C,H^4)	12+ 4= 16	22,250 (gas)	191,070	8,598
Ethane (ethylene hydride)	(C^2,H^6)	24+ 6= 30	26,650 (gas)	341,930	15,387
Propane (propylene hydride).	(C^3,H^8)	36+ 8= 44	33,850 (gas)	489,900	22,050
Ethylene (olefiant gas). .	(C^2,H^4)	24+ 4= 28	− 11,250 (gas)	321,770	14,480
Propylene...	(C^3,H^6)	36+ 6= 42	− 6,050 (gas)	471,830	21,232
Toluene..........	(C^7,H^8)	84+ 8= 92	5,650 (liquid)	906,990	....
Benzene....	(C^6,H^6)	72+ 6= 78	− 750 (liquid)	758,130	... .
			− 7,950 (gas)	765,330	34,440
Turpentine..	(C^{10},H^{16})	120+16=136	7,550 (liquid)	1,428,930	....
			− 1,850 (gas)	1,438,330	64,725
Naphthaline.	(C^{10},H^8)	120+ 8=128	− 19,450 (solid)	1,223,690	....
			− 24,050 (liquid)	1,228,290	55,273
Anthracene....	(C^{14},H^{10})	168+10=178	− 39,050 (solid)	1,690,150	. .....
Acetylene.......	(C^2,H^2)	24+ 2= 26	− 54,750 (gas)	305,270	16,437
Methyl-alcohol (wood spirit).	(C,H^4,O)	12+4+16= 32	65,050 (liquid)	148,270	... ...
			56,650 (gas)	156,670	7,050
Ethyl-alcohol (alcohol)....	(C^2,H^6,O)	24+6+16= 46	73,250 (liquid)	295,330	....
			63,150 (gas)	305,430	13,744
Acetone.................	(C^3,H^6,O)	36+6+16= 58	69,650 (liquid)	396,130	....
			62,150 (gas)	403,630	18,163

CHAPTER V

ORES

87. Ores in General.—An ore[1] is a metal-bearing substance from which a metal, alloy or metallic compound can be extracted at a profit. A metal-bearing substance usually consists of one or more economic minerals and gangue (waste minerals); in many instances the term stands for one or more intermediary products of metallurgical operations which have to be retreated.

The metals in the economic minerals occur, (1) in the native state (Au.Ag Cu.Hg.Pt . . .); (2) as oxides (Fe.Mn.Sn.Cu.Al . . .); (3) as sulphides including selenides, tellurides, arsenides, antimonides (Cu.Pb.Ag.Zn.Ni.Co. Bi . . .); (4) as haloids (Ag . . .); (5) as oxygen salts: carbonates (Cu.Pb. Zn . . .), silicates (Cu.Zn . . .), etc. Thus ores are spoken of as native, oxide, sulphide, etc., ores. Ores are named according to their leading useful metals: Fe, Cu, Pb, Ag, Cu . . . ores; Cu–Pb, Ag–Au, Ag–Pb, Ag–Cu . . . ores; according to the amounts of these useful metals they are termed as rich and poor, first and second class ores. The character of the gangue serves for classifying ores as siliceous, argillaceous, calcareous, ferruginous, or more broadly as acid and basic (fluxing) ores. In regard to the treatment, ores are spoken of as smelting and milling (*i.e.*, amalgamating and leaching) ores; the latter are often termed dry ores on account of the scarcity of base metal present. The sizes of the single pieces furnishes the distinctions of coarse, medium-size and fine ores; the last two may be concentrates and slimes from dressing works.

88. Valuation of Ores.[2]—In valuing an ore, the smelter reckons the price per ton as the difference between the market value of the amount of metal which he expects to recover and the cost of treatment plus the profit which he desires to make. The value V of a ton of ore to the smelter may be expressed in a general way by $V=\frac{a-y}{100}\times m-(T+P)$, in which a is the assay value in per cent., y the metal yield in per cent., m the market price of a ton of metal, T the total cost of treatment, and P the profit. With complex ores, the value $\frac{a-y}{100}\times m$ has to be determined for each useful metal. In the United States there is no generally accepted standard for valuing ores; it varies with the metals and the customs of mining districts and smelting centers; whatever the differences as to detail, the general principle given above remains unchanged.

[1] Kemp, *J. Canad. Min. Inst.*, 1909, XII, 356.

[2] Kursch, P., "Die Untersuchung und Bewertung von Erzlagerstätten," Enke, Stuttgart, 1911; Kreutz, *Glückauf*, 1905, XLI, 1077, 1626.

CHAPTER VI

FUEL[1]

89. Combustion.—The term combustion denotes an oxidation of sufficient rapidity to be attended by an evolution of heat and light. Fuel may be defined as any substance which, being rapidly burned in air, evolves heat capable of being applied to industrial purposes. The common heat-producing elements are C and H. In certain processes, such as converting, pyritic smelting, roasting, the oxidation of Si, P, S and of metal, furnishes all or part of the required heat. For combustion to begin and to continue, the combustible must be brought to the kindling temperature (ignition-point). This varies with the chemical constitution and porosity of the combustible and the pressure of the atmosphere. Massive carbon does not ignite nor burn as readily as when it is finely divided (spontaneous combustion); at a great altitude combustion progresses more slowly than at sea-level.[2]

TABLE 59.—KINDLING TEMPERATURES OF FUELS

Solid	Deg. C.	Gaseous	Deg. C.[4]	Deg. C.[5]
Dry peat	225	Hydrogen	585	580–590
Bituminous coal	326	Carbon monoxide, moist	651	644–658
Pine wood	395	Ethylene	543	542–547
Charcoal, made at 350° C.	360	Acetylene	429	406–440
Charcoal, made at 1,250° C.	650	Hydrogen sulphide	364	....
Anthracite	700	Methane	650–750	650–750
Coke	700	Ethane	520–630	....
Mine timbers[3]	200–400	Benzene	....	406–440
Bituminous coal[3]	400	Illuminating gas	....	580–590
		Water gas	....	644–658
		Enriched producer gas	....	644–658

[1] Works on General Metallurgy, given p. 2; further:

Percy, J., "Metallurgy, Introduction, Refractory Materials, and Fuel," Murray, London, 1875.

Groves, C. E., and Thorp, W., "Chemical Technology," Vol. 1, chapter on "Fuel and Its Applications," by E. J. Mills and F. J. Rowan, Churchill, London, 1889.

Sexton, H. A., "Fuel and Refractory Materials," Blackie & Son, London, 1910.

Fischer, F., "Chemische Technologie der Brennstoffe," Vieweg, Brunswick, 1897–1901.

Colomer, F., and Lordier, C., "Combustibles Industriels," Dunod, Paris, 1906.

Von Jüptner, H., and Nagel, O., "Heat Energy and Fuels," McGraw-Hill Book Co., New York, 1908.

[2] Palmer, *Eng. and Min. J.*, 1906, LXXXI, 134.

[3] Hall, *Tr. Inst. Min. Eng.*, 1908–09, XXXVI, 2.

[4] Average temperatures of Dixon-Coward, *Proc. Chem. Soc.*, 1909, XXVI, 67.

[5] Grebel, *Gén. Civ.*, 1912, LXI, 256; *Eng. Min. J.*, 1912, XCIV, 447.

When a fuel is once ignited combustion will continue if the heat evolved is sufficient to keep the temperature of the substance above the ignition-point; if not, combustion will cease as soon as the external source of heat which brought the fuel to the kindling temperature is withdrawn. Combustion can be perfect (complete) or imperfect (incomplete). It is perfect when the constituents of a fuel are brought to the highest state of oxidation or, what is the same, to the state evolving the largest amount of heat, viz., C to CO_2, H_2 to H_2O. It is imperfect when the constituents of a fuel are brought only to a lower state of oxidation. If part of the fuel remains unchanged, the combustion is insufficient.

The amount of air theoretically required for perfect combustion is found by a simple calculation: $C+O_2=CO_2$ or $12+2\times16=44$; 1 kg. of C. requires 2.667 kg. of O. In the same manner 1 kg. of H requires 8 kg. of O. As 1 kg. of O corresponds to 4.33 kg. of air (air = O 23.10 per cent. + N 76.90 per cent. weight, or O : N = 1 : 3.33), 1 kg. of C will require $2.667\times4.33=11.548$ kg. of air; and 1 kg. of H, $8\times4.33=34.640$ kg. Hence the weight of air theoretically necessary for the perfect combustion of 1 kg. of fuel $=\dfrac{11.548\,C+34.640\,H}{100}$ kg., in which C = per cent. carbon and H = per cent. available H.

If the volume of air necessary is desired instead of the weight, it may be found from the latter by $\dfrac{\text{kg. air}}{1.2936}\times\dfrac{760(1+0.00366t)}{b}$ cbm., in which 1.2936 kg. stands for 1 cbm. air at 0° C., and 760 mm. pressure; $1+0.00366$ for expansion coefficient of gas; *b* for the barometric pressure in mm., and *t* for the temperature in ° C.

A simpler way is to calculate the volume directly. Starting with the fact that 0.54 is the weight of C in 1 cbm. CO_2 at 0° C. and 760 mm. pressure, we have $\dfrac{\text{kg. C}}{0.54}=$ cbm. CO_2. From Table 64 it is seen that the volume of CO_2 found is equal to the volume of O required. Therefore, dividing the above result by 0.208, the amount of O in normal air, will give directly the volume of air under standard conditions, *i.e.*, 0° C. and 760 mm. pressure.
In actual work as much as twice the theoretical amount of air is required.

Let a fuel have the composition: C 76.1 per cent., H 5.1 per cent., O 8.8 per cent., H_2O 5.0 per cent., ash 5.0 per cent. = 100 per cent. The available H is found by $16\ O:2H=8.8:x$, $x=1.1$, $\therefore\ 5.1-1.1=4.0$ available H. According to the formula given above there are required $\dfrac{11.548\times76.1+34.640\times4.0}{100}=$ 10.17 kg. air; according to Table 60: $11.555\times0.761+34.664\times0.04=10.18$ kg., or at 0° C. and 760 mm., 7.86 cbm.

The conditions necessary for perfect combustion are: (1) that air in sufficient quantity be brought into intimate contact with the fuel; (2) that it be given sufficient time to act, and (3) that the temperature most suitable for the combustion be maintained throughout. As an intimate mixing of gaseous fuel and air is more readily obtained than of solid fuel, the excess air necessary can be smaller with gaseous fuel. The mixing is better accomplished if gas and air

travel at a different speed and meet at an angle than if they have the same velocity and come in contact as parallel sheets. The excess air required by a solid fuel depends largely upon its porosity. The more porous the fuel, the greater will have to be the excess in order to burn it completely in a given time, and this on account of the many points of attack; conversely, with a given amount of air, a porous or finely crushed fuel will give more CO than will a dense fuel or a fuel in lump form, and this on account of the lack of O. Caking and splintering or crumbling of solid fuels which hinder free access of air tend toward imperfect combustion.

TABLE 60.—OXYGEN AND AIR REQUIRED FOR PERFECT COMBUSTION[1]

1 Kg.	Requires kg.		Product of combustion		Nitrogen taken from air, kg.
	Oxygen	Dry air	Composition	Kg.	
C	1.333	5.777	CO	2.333	4.444
C	2.667	11.555	CO_2	3.667	8.888
CO	0.571	2.472	CO_2	1.571	1.901
H	8.000	34.664	H_2O	9.000	26.664
CH_4	4.000	17.332	CO_2,H_2O	2.750, 2.250	13.332
C_2H_4	3.429	14.858	CO_2,H_2O	3.143, 1.286	11.429
Fe	0.286	1.238	FeO	1.286	0.952
Fe	0.429	1.857	Fe_2O_3	1.439	1.428
Si	1.143	4.953	SiO_2	2.143	3.810
P	1.290	5.586	P_2O_5	2.290	4.296
Mn	0.291	1.261	MnO	1.291	0.970
S	1.000	4.333	SO_2	2.000	3.333

With ordinary fuels, *i.e.*, those in which C and H are the leading heat-producing constituents, CO_2 and H_2O are the products of perfect combustion. If these are brought to a high heat they are liable to be split (dissociated) into their constituents. Deville[2] taught that the dissociation of H_2O began at 1,000° C. and was complete at 2,500°. This was accepted until Mallard and LeChatelier[3] found that dissociation was hardly perceptible at 3,480° C. Deville[4] also believed that the dissociation of CO_2 began at 1,200° C. and was complete at 2,000°. Mallard and LeChatelier found that it did not even begin at 2,000° and that only 30 per cent. was thus split up at 3,200° C. According to Langer-Meyer,[5] 2CO is only slightly decomposed into C and CO_2 at 1,700° C.

[1] From Jüptner, H. von, "Die Untersuchung von Feuerungsanlagen," Hartleben, Vienna, 1891, p. 69; Mn→MnO_2 omitted, S→SO_2 added, some data corrected.

[2] *Compt. Rend.*, 1863, LVI, 195, 392.

[3] *Ann. Mines*, 1883, IV, 379.

[4] *Op. cit.*, p. 729.

[5] "Pyrotechnische Untersuchungen," Vieweg, Brunswick, 1885, p. 61.

Dissociation can therefore be neglected as far as carbon-heated furnaces are concerned; it has to be considered with electrical furnaces.

Combustion of a fuel is usually attended by the emission of light. This may be a simple glow, as is the case with charcoal or coke free from gas, or the fuel may burn with a more or less luminous flame, which will be the case when the fuel is a gas or a solid which evolves combustible gases or vapors upon being heated. If it is a gas, the flame may be very long; if a solid giving off gases or vapors, it will be comparatively short. According to the amount of combustible gas evolved, solid fuels are sometimes spoken of as long- and short-flame fuels.

It is generally stated that the flame of fuel may be oxidizing, neutral or reducing according to the relative proportions of gas (vapor) and air used. An oxidizing flame is readily obtained, a neutral flame has only an imaginary existence, and in large-scale work the reducing flame can only be approximated in a carbon-heated furnace. A reducing atmosphere is readily maintained in an electric furnace. If in burning carbonized fuel a flame is occasionally seen, this can be due to two causes: either the C is burning with an insufficient amount of air to CO, and this burns with a flame to CO_2; or CO_2 formed is being reduced to CO by coming in contact with incandescent C and then burns, forming again CO_2.

The heat generated by the combustion of a fuel may be transmitted by contact or by radiation. The former will be mainly the case with fuels that are free from gas (charcoal, coke), the latter with flaming fuels, although not exclusively so. F. Siemens[1] maintained that the only proper method of heating with a flame was by radiation.[2] But it is a fact of every day experience that contact of a flame with a solid is harmful only when this has a cooling effect; otherwise it is advantageous, as, *e.g.*, with the perforated fire-wall of a boiler. In order to furnish the flame free space in which to develop without obstruction, furnaces are now constructed with larger combustion chambers than was common in order to assist a thorough mixing of gas and air, and to give the products of combustion a better chance to act upon the charges (old and new copper reverberatory smelting furnaces).

90. Heating Effect of Fuels.—In determining the effect a fuel produces in combustion a distinction is made between the power and the intensity. The *calorific power* is the amount of heat a given weight of fuel evolves in perfect combustion. It is independent of the time taken. The *calorific intensity* or *pyrometric effect* is the degree of heat evolved by a given weight of fuel in perfect combustion in air at 0° C. and 760 mm. pressure. This theoretical maximum temperature is inversely proportional to the time occupied in producing it.

91. Calorific Power.[3]—There exist two units for measuring heat values.

[1] *J. I. and St. Inst.*, 1884, II, 434.

[2] Reverberatory Furnace of the Kedabeg Copper Works, *Min. Ind.*, 1897, VI, 246; *Tr. Inst. Min. Met.*, 1904–05, XIV, 497.

[3] Poole, H., "The Calorific Power of Fuels," Wiley, New York, 1900.

Mahler, P., "Études sur les Combustibles Solides, Liquides et Gazeux, Mesure de leur Pouvoir Calorifique, Béranger, Paris, 1903.

Glikin, W., "Kalorimetrische Methoden," Bornträger, Berlin, 1911.

The majority of metallurgists use the metric unit or gram-calorie (cal.), which is the amount of heat required to raise the temperature of 1 g. of H_2O 1° C. If for practical purposes the kg. is substituted for the g., we have the kilogram-calorie (Cal.). In order to obtain a figure 1,000 times as large as the gram-calorie, Berthelot expresses the values in Cal., using, however, the gram as the basis for the laboratory experiment. Ostwald does away with the decimal point of Berthelot by using a unit 100 times as large as the gram-calorie and calls it *K*. Thus 12 g. C. develop 97,200 cal. = 97.2 Cal. (Berthelot) = 972 *K*. (Ostwald); 12 kg. C. = 97,200 Cal. In metallurgical treatises the heat evolved is often referred to the unit weight (1 g. of C. instead of to the combining weight (12 g. of C.). Thus the calorific power of C. has been and is still frequently given as 97,200 ÷ 12 = 8,100. Using the combining weights, however, generally facilitates calculations. The calorific power of 1 g. of C. in its different allotropic states (diamond, graphite, charcoal) shows a range of from 7,770 to 8,080 cal.[1] Grüner[2] states that the figure 9,000 for fuels gives results which accord best with experimental facts. The figure 8,100 of J. W. Richards[3] is used here; in most metallurgical treatises 8,080 is the common number.

Mechanical engineers use the British Thermal Unit (B. T. U.) which is the amount of heat needed to raise the temperature of 1 lb. H_2O 1° F. 1 cal. = 3.968 B. T. U., and 1 B. T. U. = 0.252 cal.

In the table of heat of formation of oxides (p. 72, 73) the calorific power of CO and CO_2 are given as 29,160 and 97,200. It seems strange at first that C burning to CO should evolve less than 1/3 the heat than when burning with twice the amount of O to CO_2. The probable explanation is that heat is rendered latent by the passage of C from the solid state to that of vapor.

The calorific power of a fuel can be found either by calculation or by experiment in the laboratory. Mechanical engineers often express the heating power of a fuel in terms of evaporating power, *i.e.*, the pounds of H_2O a fuel can evaporate in a suitably constructed boiler.[4] The results obtained will always be low and can have only a relative value, as they depend upon the heating plant, the condition of the boiler and the manner of firing. It seems more advantageous to make a number of small laboratory tests and average the results; discrepancies found will be due to the inaccuracy of the samples used.

92. Calculation of the Calorific Power of Solid and Liquid Fuels.—This method is based upon the supposition that the calorific power of a fuel is equal to

[1] Favre and Silbermann, *Ann. chim. phys.*, 1852, XXXIV, 357.

[2] Grüner, "Traité de Métallurgie Générale," I, p. 52.

[3] "Metallurgical Calculations."

[4] Kent, W., *Tr. A. I. M. E.*, 1885–86, XIV, 727; "Steam Boiler Economy," Wiley, New York, 1901.

Poole, *op. cit.*, p. 109.

Fischer, F., "Chemische Technologie der Brennstoffe," 1897, I, 130, 161, 382.

Kerl-Stohmann, "Handbuch der chemisch-technischen Chemie, Vieweg, Brunswick, 1893, IV, 272.

U. S. Geol. Survey, *Bulletins* 261 (Parker), and 290 (Holmes); Professional Paper No. 48 (Parker-Holmes-Campbell).

Campbell, *Econ. Geol.*, 1907, II, 285; *Mines and Minerals*, 1908, XXVIII, 312.

the sum of the calorific powers of its components in perfect combustion. The results obtained are likely to be too high, as the method takes no account of the amounts of heat necessary to decompose the fuel and to gasify the constituents. Dulong first proposed this method; his formula for a fuel consisting of C, H and O is:

$$\text{C. P.} = \frac{8{,}100\,\text{C} + 34{,}500\left(\text{H} - \frac{\text{O}}{8}\right)}{100}$$

in which C, H and O are given in per cent. It will be noticed that only the available H is considered, *i.e.*, that which is in excess of the proportion 1H : 8O, the ratio in which the two elements combine to form water. Although the O of a fuel is not known to be present in this combination, as far as practical purposes are concerned, it may be considered as virtually existing in this form. And only the available H forms a source of heat.

If the S of a fuel is to be taken into consideration as a source of heat, the Dulong formula, supposing the S to be burnt to SO_2 and some to SO_3 will be changed to

$$\text{C. P.} = \frac{8{,}100\,\text{C} + 34{,}500\left(\text{H} - \frac{\text{O}}{8}\right) + 2{,}250\,\text{S}^{1}}{100}$$

In the preceding formulæ, the water formed in combustion is assumed to be liquid water, *i.e.*, water below 100° C.; if it remains uncondensed, as is the case in practically all metallurgical operations, its latent heat of vaporization must be included in the formula. This will then be changed to

$$\text{C. P.} = \frac{8{,}100\,\text{C} + 34{,}500\left(\text{H} - \frac{\text{O}}{8}\right) + 2{,}250\,\text{S} - 537\,\text{H}_2\text{O}}{100}$$

Taking as an example a bituminous coal[2] from Palestine, O., of the composition: C 73.64, H 5.06, O 9.47, N 1.24, S 2.34, ash 8.25 per cent., its calorific power with liquid water will be 7,350 cal., and with steam at 100° C. only 7,105 cal., as 5.06 H give $5.06 \times 9 = 45.54$ H_2O, which, multiplied with the heat of vaporization at 100° C. and divided by 100, gives in round figures 245 cal., and $7{,}350 - 245 = 7{,}105$. The commercial formula for bituminous coal adopted by German engineers is:

$$\text{C. P.} = \frac{8{,}100\,\text{C} + 29{,}000\left(\text{H} - \frac{\text{O}}{8}\right) + 2{,}500\,\text{S} - 600\,\text{W}}{100}$$

in which H is assumed to be burnt to steam of 100° C., and the value W to represent the percentage of moisture.

In order to do away with the necessity of making an ultimate analysis of a fuel, Gmelin[3] devised the following empirical formula for calculating the

[1] Mahler's total acidity figure: Lord-Haas, *Trans. A. I. M. E.*, 1897, xxvii, 262, 264.

[2] Lord-Haas, *Trans. A. I. M. E.*, 1897, xxvii, 266, bituminous coal No. 40.

[3] *Oest. Zt. Berg-Hüttenw.*, 1886, xxxiv, 365.

calorific power of solid fuels which requires only the determination of the percentages of hygroscopic water and ash: C. P. $=(100-(w+a))\times 80-C\times 6\times w$.

Per cent. water......	<3	3-4.5	4.5-8.5	8.5-12	12-20	20-28	>28-
C.............. ..	-4	6	12	10	8	6	4

In it $w=$ per cent. H_2O, $a=$ per cent. ash, C= constant varying with percentage of H_2O. The formula is based upon the observed fact that in a solid fuel a relation exists between the percentages of O and H_2O, and that both depend to some extent upon its porosity. If, therefore, the percentage of H_2O is determined, the figure may be used to judge of the percentage of O, and to thus approximate the calorific power. The calorific power of the above bituminous coal of Palestine, Ohio, would be $[(100-(2.45+8.25)]80-(-4\times 6\times 2.45)=7{,}202$ cal. Comparative figures are given in Table 62.

Jüptner[1] proposed a formula for solid fuels based upon the ratio Vol. H-C and Fixed C, and the amounts of O required for their complete combustion.

Goutal[2] calculates the calorific power of 1 kg. of coal according to the formula:

$$\text{C. P.}=82\ C+aV'$$

in which *C* is the percentage of fixed C, *V* the volatile hydrocarbon, and *a* a factor depending upon the percentage of volatile hydrocarbon *V'* of the pure coal free from ash and water, *i.e.*, of F.C.+V.H-C. The values for *a* with coals in which *V'* is below 40 per cent. are given in Table 61.

TABLE 61.—GOUTAL'S FACTORS FOR CALCULATION OF CALORIFIC POWER

V'	a	V'	a	V'	a	V'	a
Per cent.	cal.	Per cent.	cal.	Per cent.	cal.	Per cent.	cal.
1-5	100	14	120	23	105	32	97
5	145	15	117	24	104	33	96
6	142	16	115	25	103	34	95
7	139	17	113	26	102	35	94
8	136	18	112	27	101	36	91
9	133	19	110	28	100	37	88
10	130	20	109	29	99	38	85
11	127	21	108	30	98	39	82
12	124	22	107	31	97	40	80
13	122	....	....	....	....	....	....

[1] *Op. cit.*, 1893, xxxiv, 420.

Ahrens, "Sammlung chemischer und chemisch-technischer Vorträge," Enke, Stuttgart, 1898, II, 423; *Feuerungstechnik*, 1912, I, 120, 140.

[2] *Electrochem. Metal. Ind.*, 1907, V, 145.

Mahler, P., "Etude sur les combustibles, etc.," Baudry, Paris, 1903, p. 83; also Damour, E., and Queneau, A. L., "Industrial Furnaces, etc.," Eng. and Min. J., New York, 1906, p. 221.

A coal contains F.C. 86.70, V. H–C. 10.05, ash 1.45, H_2O 1.80 per cent., then V.H–C 10.05 per cent., calculated upon the basis of pure fuel is:

$$V' = \frac{V\times 100}{C+V} = \frac{10.05\times 100}{86.70\times 10.5} = 10.4$$

The factor in Table 61 nearest to 10.4 being 129, the C.P. $=82\times 86.7+129\times 10.05=8,406$ cal. Direct experiment in the bomb gave 8,404 cal. The data assume the water vapor to have been condensed; for the practical C.P. with non-condensed vapor the necessary correction will have to be made.

Lenoble[1] simplifies the formula of Goutal by eliminating the factor a, and gives it the form:

$$\text{C. P.} = 82C + 73.66V + \frac{98\ C\times V}{C+7V}\ \text{cal.}$$

He finds that values higher than 8,700 cal. cannot be obtained and proposes

$$\text{C. P.} = 87.4\ (100-K)\ \text{cal.}$$

in which K is the sum of the percentages of H_2O (drying 2 hrs. at 105° C.) and of ash of the coal.

92a. Determination of the Calorific Power of Solid and Liquid Fuels.—The simplest way of approximately determining the calorific power of a solid fuel is by the assay method of Berthier.[2] It is based on the theory of Welter that the amount of heat evolved in perfect combustion is proportional to the quantity of O consumed, and Berthier proposed to oxidize the fuel at the expense of the O of PbO and to weigh the resulting lead button. Welter's theory, however, is not correct, as 1 kg. of O combining with C to form CO_2 evolves $8,100\times 3/8 = 3,037$ cal., and 1 kg. of O combining with H_2 to form gaseous H_2O evolves $29,030 : 8 = 3,630$ cal. Further, in the equations: $2PbO+C=Pb_2+CO_2$ and $PbO+H_2=Pb+H_2O$, three kg. of C reduce the same amount of lead as does 1 kg. of H. As the combustion of C forms the basis of calculation, the values obtained by the Berthier method will be too low; they will be the less accurate the higher the percentage of available H.

Mode of Operation.—One g. of dried and finely pulverized fuel (wood is rasped) is mixed with 40 g. PbO (passed through bolting cloth), charged in a deep close-grained clay crucible, covered with 30 g. PbO and 0.25 in. powdered glass. The crucible, which should not be more than half full, is covered with a lid, luted, heated slowly in a pot- or muffle-furnace until boiling ceases, brought to a bright-red, kept there 45 min., removed from the furnace, tapped and allowed to cool. When cold, it is broken, the lead-button freed from slag, which is to be examined for shots of lead, and weighed. Weights of duplicate assays ought to agree within 0.1 g.

Forchhammer[3] proposed to replace PbO by Pb_2OCl_2 as being more readily fusible and therefore active at a lower temperature than PbO, and also less

[1] *Bull. Soc. Chim.*, 1907, I, 111–114; *Jl. Gasbeleucht.*, 1907, XLVIII, 1006; *Chem. Centralblatt*, 1907, I, 1150.

[2] "Traité des Éssais," Thomine, Paris, 1834, I, 229.

[3] *Berg. Hüttenm. Z.*, 1846, V, 465.

corrosive. It is prepared by fusing 3 pt. red PbO and 1 pt. $PbCl_2$ in a clay crucible and finely pulverizing the product.

According to $2PbO+C=Pb_2+CO_2$, 1 g. C reduces 34 g. Pb. As C : C in fuel $=34$: a, $\frac{a}{34}$ gives the relation of the fuel in question to pure C. Let *e.g.*, the weight of the lead button from 1 g. fuel be 30 g., then $30 : 34 = 0.882$; or the fuel contains 88.2 per cent. C., and its calorific power is $8{,}100 \times 0.882 = 7{,}144$ cal. Comparative results are given in Table 62. For additional data consult papers by Carnot,[1] John-Fullon,[2] Noyes-MacTaggart-Craver,[3] John-Eichleiter,[4] Kerr,[5] and Mills-Rowan.[6]

The calorimetric method of experimentally determining the calorific power is the most reliable. It consists in burning a known weight of fuel in O,

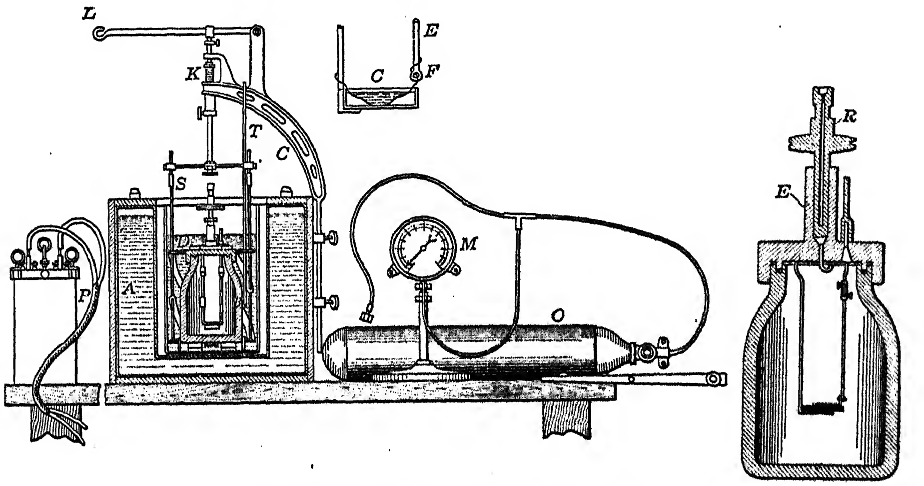

FIG. 46.—The Mahler calorimetric bomb.

cooling the products of combustion in a known weight of water and measuring the rise of temperature.

MAHLER BOMB CALORIMETER.—Of the many calorimeters that have been devised, that of Berthelot and Mahler[7] is the simplest. It is shown in figure 46. *A* is a non-conducting jacket of thin sheet brass, 9 1/16×5 7/8 in.; it rests upon a wooden support and is placed in a double-walled insulating vessel, 7 1/16

[1] *Ann. Mines*, 1879, XVI, 426.

[2] *J. I. and St. Inst.*, 1891, I, 217; *Berg. Hüttenm. Z.*, 1893, LII, 89.

[3] *J. Am. Chem. Soc.*, 1895, XVII, 847.

[4] *Berg. Hüttenm. Z.*, 1895, LIV, 371.

[5] *Tr. Am. I. Mech. Eng.*, 1900, XXI, 304.

[6] "Fuel," p. 713.

[7] Mahler, *Bull. Soc. d'Enc.*, 1891, VI, 704; full abstract *Coll. Guard.*, 1892, LXII, 147, 203, 237, 285, 331.

Thwaite, *J. I. and St. Inst.*, 1892, I, 189.

Kent, *Min. Ind.*, 1892, I, 97.

Lürmann, *Stahl u. Eisen*, 1893, XIII, 52.

Struthers, *Sch. Min. Quart.*, 1894–95, XVI, 201.

in. high, with felt covering (3/8 in.) and a space, 2 3/8 in., containing water at room temperature. *B* is the combustion chamber or bomb, nickel-plated on the outside, enameled on the inside, 34 cu. in. capacity, 5 7/8×3 15/16 in., thickness of wall 5/16 in., weight 4 kg. or 8.8 lb. *C*=platinum tray, 1/8 in. diam. and 3/16 in. deep, suspended by platinum rod, 1/16 in. diam., to receive the fuel to be tested. *D*=calorimeter vessel filled with water. *E*=isolated electrode. *F*=ignition wire, iron No. 26 to 30 B.W.G. or fine platinum. *G*= agitator stand. *K*=agitating mechanism. *L*=lever of agitator. *M*=pressure gauge. *O*=oxygen holder with 1,200 liters O at 120 atm. pressure, good for 100 tests. *P*=bichromate battery of 2 amp. at 12 volts. *S*=agitator. *T*=thermometer reading to 0.01° C.

The mode of operating is as follows: The bomb having been washed out, but not necessarily dried, is ready to receive its charge. Clamp the platinum tray with its pan *c* and the electrode rod *e*; connect the two by an iron or platinum ignition wire; pour into the tray pulverized fuel, using with bituminous coal slightly over 1 g. If the fuel is too coarse, it agglomerates, if too fine it dusts. See that the ignition wire dips into the fuel. Transfer the top with its charge to the bomb, tighten the screw with a wrench making certain that the lead washer *P* is clean and the thread greased. The next step is to fill with O; close the bomb with the needle-valve *R*; connect its upper end by means of the flexible copper tube with the oxygen tank and the gauge; test for leakage by admitting O slowly until the gauge shows a pressure of 10 atm. and close the valve. Any drop in the gauge will be proof of escaping gas. Open again the valve *R*, turn oxygen on slowly until the gauge reads 20 to 25 atm.,[1] and close the valve. Before placing the bomb in the calorimeter, fill the outer jacket with water of room temperature, thoroughly dry the inner side, put the calorimeter in position and the bomb in its place, adjust the stirrer, add 2.2 kg. (=4.84 lb.) water of room temperature and insert the thermometer. Everything is now ready for the oxidation of the fuel. Start the stirrer, work it continuously and make observations every 15 sec. During the first 5 to 10 min. of stirring, the temperatures become equalized and conditions normal. Now close the circuit; the coal ignites, the combustion is instantaneous and perfect; the temperature rises to a maximum in about 3 min. and then sinks again; continue the observations for 5 to 10 minutes more, if it is desirable to make the cooling corrections, *i.e.*, corrections for the loss of heat of the calorimeter by radiation and evaporation during the test.[2] This is unnecessary for technical valuations, if in determining the water-equivalent (481 g. in Mahler's bomb) enough naphthalene has been used to cause the same rise within 1° C. as in burning 1 g. coal. With this supposition the following simplified formula can be used to compute the heat value:

$$nH = W(t_2 - t_1) + K(t_2 - t_1) \text{ or}$$

$$H = \frac{1}{n}(W+K)(t_2 - t_1)$$

in which n=g. combustible, H=heat of combustion sought, W=weight of

[1] Lord-Haas (*Tr. A. I. M. E.*, 1897, XXVII, 261) found 15 to 17 atm. sufficient for 1 g. coal.

[2] Hofman-Wen., *Tr. A. I. M. E.*, 1910, XLI, 506.

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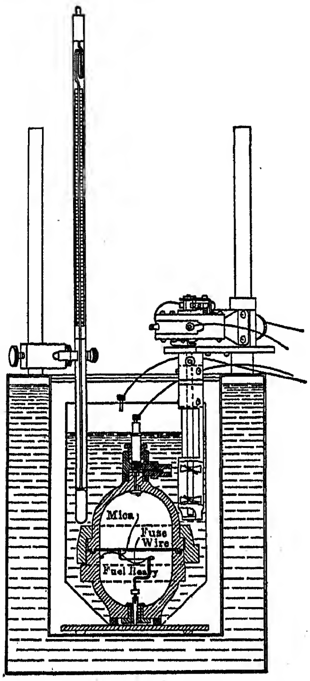
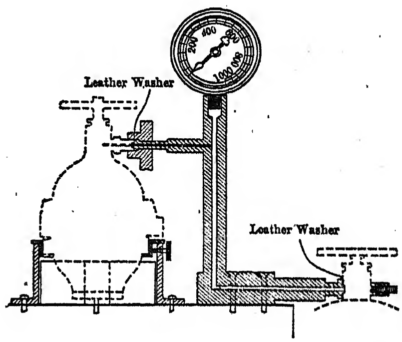
water in calorimeter, K = water equivalent of bomb (calorimeter + stirrer + thermometer, etc.), t_2 = maximum temperature, t_1 initial temperature. The figure obtained includes the heat evolved by the formation of HNO_3 and H_2SO_4 (and of Fe_2O_3 if iron wire was used), a factor usually neglected. The acids are washed out after the close of the experiment; if they are to be determined a definite volume of H_2O has to be used. The HNO_3 is titrated with NH_3 and the H_2SO_4 precipitated with $BaCl_2$.

The water equivalent of the calorimeter (the heat absorbed by the apparatus, expressed in cal.) is ascertained either by calculation[1] or by experiment.

In making a determination either 1 g. naphthalene (9,692 cal.) or carbonized sugar (8,120+ cal.) is burned,[2] the difference between the theoretical and the actual figure noted, and divided by the difference between maximum and initial temperatures. In the above formula $\frac{H}{t_2-t_1}=K$.

THE EMERSON CALORIMETER.[3]—The Mahler





FIGS. 47 and 48.—Bomb of the Emerson fuel-calorimeter.

bomb has several disadvantages: cost ($300), peeling of enamel, extra expense if platinum-lined, awkwardness of handling and of securing of top with charge in tray, complex and feeble construction of stirring mechanism. These are avoided by the Emerson fuel calorimeter, the bomb of which is shown in Figs. 47 and 48. The bomb (inner height 5 in., diam. 3 in., thickness 3/8 in., weight 3,694 g., sp. heat 0.118), is of steel and lined with sheet metal spun in to fit, or with a double-process high-temperature porcelain; it costs $150.

The joint, an enlarged plumber's union, is in the middle and furnishes easy access to all parts; the lower rim has a groove to receive the lead washer, the

[1] Multiplying the weights of the several parts of the calorimeter with their specific heats, and adding the sums to the weight of water used (Hofman-Wen, *op. cit.*, 5071).

[2] Cane sugar from the Bureau of Standards, Washington, D. C., is probably the most desirable substance for standardization.

[3] *J. Ind. Eng. Chem.*, 1909, 1, 1; Emerson Apparatus Co., 251 Causeway St., Boston, Mass.

upper a rib. The platinum or nickel tray (1 in. diam. and 0.25 in. deep) rests on an isolated loop of nickel wire which enters at the bottom of the bomb and forms the positive pole; on one side is a screw-eye, the negative pole to which is attached the platinum fuse-wire passing through the fuel charge. The brass stirrer (spec. heat 0.094) is a vertical rod with 3 propeller blades rotated in a cylinder by a small electric motor. The admission of O takes place over an inverted cup instead of through a bent tube.

Example.—The calorimeter holds 1,900 g. water; 0.8666 g. coal are to be tested.

Readings of Thermometer

Time	Temperature	Time	Temperature	Time	Temperature
0	20.348	30	21.000	11	23.182
30	20.350	6	22.600	30	23.178
1	20 352	30	22.900	12	23.174
30	20.356	7	23.100	30	23.170
2	20.358	30	23.150	13	23.166
30	20.360	8	23.194	30	23.162
3	20.362	30	23.196 Max. temp.	14	23.158
30	20.364	9	23.196	30	23.154
4	20.368	30	23.194	15	23.150
30	20.374	10	23.194		
5	20.376 Firing temp.	30	23.190		

(1) The apparent rise in temperature is $23.196-20.348=2.848°$.

(2) The cooling correction is computed as follows: Rate of change of temperature the first 5 min. before firing, $20.376-20.348=0.028$, and this divided by 5 min. gives $R_1=0.0056$ p. min. Rate of change of temperature during the last 5 min. gives $23.194-23.150=0.044$, and divided by 5 gives $R_2=0.0088$ p. min. The difference $0.0088-0.0056=0.0032$, multiplied by 3.5 min. (diff. firing and maximum temp.) gives 0.006 (additive) as the total cooling correction, or the total corrected rise of temperature, as 2.826°. This for 1 g. fuel = 3.261°. The water equivalent of bomb, calorimeter can, stirrer, etc., = 496. The calorific power of the fuel is therefore $(1{,}900+496)\times 3.261=7{,}794$ cal.

The numerical examples Nos. 1–24 of calorific values given in Table 62 have been selected from Jüptner's table.[1] The original does not contain the data obtained with the Dulong formula. As most of the examples chosen by Jüptner are those given by Lord and Haas[2] who used

$$\frac{8{,}080C+34{,}462(H-\frac{1}{8}O)+2{,}250\ S}{100}$$

to calculate the calorific power; the other missing calculations have been made on the same basis. Examples Nos. 25–29 are by Sherman and Amend.[3]

[1] Ahrens, "*Sammlung chem. und chem. techn. Vorträge.*"

[2] *Tr. A. I. M. E.*, 1897, XXVII, 259.

[3] *Sch. Mines Quart.*, 1911, XXXIII, 30.

TABLE 62.—CALORIFIC POWER OF SOME SOLID FUELS

	Name or locality	Analysis									Calorific power			
		Ultimate							Proximate					
		C	H	O	N	S	H_2O	Ash	V. H–C	F.C.	Dulong formula	Gmelin formula	Berthier method	Calorimetric method
1	Anthracite, Pa.	86.46	1.995	1.449	0 75	.....	3 45	5.90	2.75	87.90	7,608	7,128	7,425	7,484
2	Cardiff, Wales	88.98	4.54	1.	50	0.84	3.01	1.13	14.22	81.64	8,707	7,560	8,243	9,033
3	Pocahontas, W. Va.	85 40	4.39	3.94	0 85	0 62	0.85	4.80	18 60	77.75	8,258	7,718	7,845	8,281
4	St. Étienne, France	84.60	4.772	4.592	0 84	.....	1.25	4.00	19.75	75.00	8,282	7,610	7,449	8,391
5	Wigan, England	76.49	4.96	8.46	1.44	1.07	4.84	2.75	28.31	64.10	7,552	7,218	7,128	7,572
6	Blanzy, France	79.39	4 97	8 72	1.13	.....	3.90	1.90	30.10	64.10	7,751	7,395	7,353	7,866
7	Salineville, O.	71.13	4.95	9.93	1.23	1.86	3.15	10.90	35.00	50.95	7,068	6,763	6,586	7,032
8	Beaver Creek, Pa.	74.60	5.06	8.23	1.40	1.96	1.50	8.75	34.33	55.42	7,460	7,216	6,459	7,360
9	Hocking Valley, O.	66.50	5.16	15.57	1.43	1.67	6.65	9.67	34.14	49.54	6,520	6,215	6,152	6,496
10	Clinton, Pa.	73.57	5.14	10.14	1.24	1.86	2.55	8.05	35.60	53.80	7,320	7,213	6,886	7,300
11	Carnegie, Pa.	76.57	5.13	8.82	1.64	1.76	1.07	6.08	37.79	55.06	7,614	7,453	7,133	7,765
12	Thacker, W. Va.	78.40	5.19	7.56	1.40	1.40	1.35	6.05	36.35	56.25	7,831	7,440	7,364	7,867
13	Turtle Creek, Pa.	74.48	5.05	8.39	1.37	1.66	1.75	9.05	36.20	53.00	7,433	7,178	6,988	7,394
14	Carnegie, Pa.	73.50	5.19	8.08	1.44	2.54	1.08	9.25	37.67	52.00	7,436	7,199	6,952	7,396
15	Steubenville, O.	71.40	4.62	10.68	1.20	3.00	2.40	9.10	39.20	49 30	6,970	7,137	6,889	7,113
16	New Pittsburg, Ind.	62.88	5.07	13.06	1.01	7.46	6.83	13.31	39.92	39 93	7,732	5,924	5,914	6,175
17	Lignite, Russia	62.79	5.56	31.39	0.26	.....	.....	.....	66.40	33 60	5,875	.....	5,471	5,875
8	Peat, compressed	56.02	5.57	34.60	1.11	.....	11 90	2.51	58.58	27 01	4,954	6,131	4,828	5,125
9	Albertite	84.91	8.67	3.87	2.55	.....	.....	.....	57.10	42.90	9,676	.....	8,845	9,433
20	Gum	44.44	6.17	49.39	.....	.....	.....	.....	75 58	24.42	3,590	.....	3,591	4,464

TABLE 62.—CALORIFIC POWER OF SOME SOLID FUELS—(*Continued*)

	Name or locality	Analysis									Calorific power			
		Ultimate							Proximate					
		C	H	O	N	S	H_2O	Ash	V. H-C	F.C.	Dulong formula	Gmelin formula	Berthier method	Calori-metric method
21	Cellulose	44.44	6.17	49.39	......	......	......	......	88.70	11.30	3,590	......	3,591	4,452
22	Boghead coal, Russia	77.38	10.14	18.82	0.66	......	......	......	80.10	19.90	9,104	......	8,140	9,104
23	Boghead coal, Russia	70.00	8.56	20.59	0 85	......	......	......	86.40	13.60	8,545	......	7,107	8,595
24	Starch	44.44	6.17	49.39	......	......	......	......	93.27	6 73	3,590	......	3,591	4,479
25	Chestnut	50.28	5.58	43 21	0.10	0 03	......	0.80	......	......	4,125	......	......	4,632
26	Hemlock tan	53.74	5.66	39.05	0.24	0 04	......	1.27	......	......	4,612	......	......	5,106
27	Oak tan	49.51	5.53	39.24	0.40	0 05	......	5.27	......	......	4,217	......	......	4,798
28	Bagasse	49.04	5.96	42.58	0.31	0.07	......	2.04	......	......	4,185	......	......	4,625
29	Oil cake	48.20	6.47	38.02	4.24	0.25	......	2.82	......	......	4,594	......	......	4,953

93. Calculation of the Calorific Power of Gaseous Fuel.—The calorific power of a gaseous fuel is equal the sum of the heats of combustion of its constituents. Let w, w', w'' be the weights of the different gases and c, c', c'' their calorific powers, then the calorific power of the mixture:

$$\text{C. P.} = wc + w'c' + w''c'' + \ldots$$

The composition by weight of a producer gas, made from bituminous coal in a furnace run hot is: N, 67 per cent.; CO_2, 7 per cent.; CO, 21 per cent.; H, 1 per cent.; C_2H_4, 3 per cent.; CH_4, 1 per cent.; total, 100 per cent. Then its C. P. will be 1,265 Cal. supposing that the products of combustion pass off at 100° C.

Component	Weight per unit w	C. P., Cal.	w × Cal.
N.	0.67	....	....
CO_2	0.07	....	....
CO	0.21	2,430	510
H	0.01	29,030	290
C_2H_4	0.03	11,492	345
CH_4	0.01	12,000	120
Cal. pow.	1.00	....	1,265

The composition of a gas is, however, more frequently given in per cent. by volume than in per cent. by weight, and calculations are often simplified by adhering to volumes instead of first changing them into weights. If this, however, is to be done, the corresponding weights are found by multiplying the volume by the weight of 1 liter of each gas:

$$\text{Per cent. vol.} \times \text{wt. of 1 liter} = \text{gr. gas}$$

and then calculating the percentage. Weights of gases and vapors are given in Table 63.

Another and more simple method is that of J. W. Richards[1] who starts from the basis that the weight of 1 liter C, H or O in a gas containing 1 atom of these elements in its molecular formula is 0.54, 0.045 and 0.72 g. For a gas with 2 atoms of C, H or O in its molecule, the respective weights are twice the above.

If the composition is given in per cent. by weight, and it is desired to calculate the calorific power by volume, the weights can be readily changed into volumes by: liter gas = gr. gas : wt. of 1 liter.

Taking the above producer gas, the composition of which was given in per cent. by weight, its calorific power will be that given in the following tabulation:

[1] *J. Frankl. Inst.*, 1901, CLII, 109; "Metallurgical Calculations," McGraw-Hill Book Co., New York, 1906, I, p. 3.

Table 63.—Weights of Gases and Vapors at 0° C. and 760 mm. Pressure

Gas or vapor	Sp. gr., air = 1	Wt. of 1 liter, sea-level, 45° lat., ing. or 1 cbm. in kg. or 1 cft. in oz. tr.
C	....	1.073
C_2H_2	1.160	1.162
C_6H_6	1.339	3.486
C_2H_4	0.571	1.251
CH_4	0.552	0.715
Illuminating gas	....	0.520
C_2N_2	1.799	2.326
CO	0.967	1.250
Water gas	....	0.860
CO_2	1.520	1.965
O_2	1.111	1.430
H_2S	1.117	1.522
SO_2	2.213	2.861
N_2	0.967	1.254
H_2O	0.622	0.804
H_2	0.069	0.0896
Air	1.000	1.293

Gas	Weight per unit, g.		Weight of 1 liter, g.		Volume, liter	C.P. 1 liter. cal.	C.P., 0.8881 lit., gas, cal.	0.8881 lit. ×1.126 = 1 liter, p.c. by vol.
N	0.67	Divided by	1.254	Equals	0.5343	......	....	0.602
CO_2	0.07		1.965		0.0357	......	....	0.040
CO	0.21		1.250		0.1680	3,062	514	0.189
H	0.01		0.089		0.1123	2,613	293	0.126
C_2H_4	0.03		1.251		0.0239	14,480	346	0.027
CH_4	0.01		0.715		0.0139	8,598	119	0.016
Total	1.00		....		0.8881	....	1,272	1.000

94. Determination of the Calorific Power of Gaseous Fuel.—Of the different apparatus for determining the calorific power of gaseous fuels the Junkers Gas Calorimeter[1] is used more than any other. It is of the continuous type; a measured volume of gas is burned for a given time, and the rise in temperature of a measured or weighed quantity of flowing cooling water noted. Fig. 49 is a vertical section of the calorimeter and Fig. 50 an elevation showing

[1] *Verh. Ver. Beförd. Gewerbefl.*, 1894, LXXIII, 175; *Z. Ver. deutsch. Ing.*, 1895, XXXIX, 564; *Berg. Hüttenm. Z.*, 1894, LIII, 154; *Engineering*, 1895, LIX, 574; *Catalogue*, Eimer & Amend, New York City; modification by Sargent, *Iron Trade Rev.*, 1907, XL, 78.

the arrangement of the parts necessary for a test. In Fig. 49 the gas is supplied through 22 to the Bunsen burner 27; it burns in the combustion chamber 28, which is a cylindrical copper water-jacket 15, the water-space being traversed by a number of copper tubes 30. The products of combustion descend in 30 and pass off through pipe 32 with damper 33, having given off their heat to the cooling water. In order to prevent loss of heat by radiation, the water-jacket 15 is enclosed by an air-jacket 36 of nickel-plated copper. The cooling water is kept under a uniform pressure by the two overflows 5 and 21. The water is admitted at 1, descends through 6, the flow being regulated by the dial-cock 9 having a pointer (*e*, Fig. 50), enters the water-jacket at 13, is evenly distributed by baffle-plates 14, rises in 15 and 16, leaves through 18 and runs out at 21. The temperatures of the cold-water inflow and the warm-water outflow are taken by thermometers 12 and 43, which permit readings to 0.1° C. Any condensed water is drawn off at 35. Fig. 50 shows the meter with gas-inlet *g*, the pressure regulator 12, the calorimeter *d* with small *a* (100 c.c.) and large (2,000 c.c.) graduate *b*, and waste-water pipe *e* with funnel. The large graduate is replaced by a tin box on platform-scales if the water is to be weighed. The parts of the apparatus are placed close together so as to allow one person to make the necessary observations; it is, however, preferable to have two persons for a test. The large dial of the gas meter reads in liters; one revolution of its pointer indicates 3 liters. In the latest meter the two small dials in the larger face have been replaced by a single one which records, by a division, every revolution of the large pointer. The pressure regulator 2 corrects any irregularities in the flow of gas.

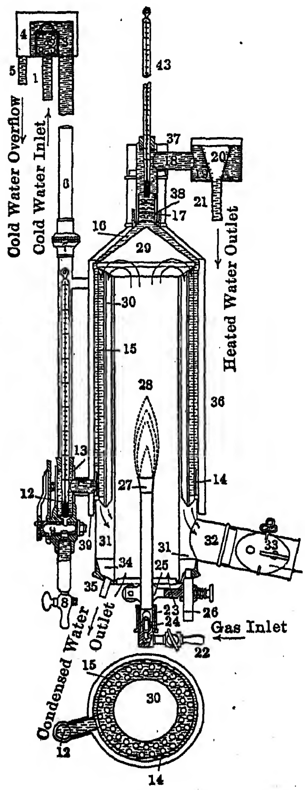


FIG. 49.—The Junkers calorimeter.

The water-supply for the calorimeter must be able to furnish about 3 liters per minute. In starting, the cold-water supply is connected with the inlet, and the cold- and hot-water overflows are turned into the waste-water pipe. Cold water is turned on, the dial-cock *e* opened enough to allow about 2 liters of water to pass through in 1.5 min., and the gas ignited after the calorimeter is filled. The temperature of the overflow water rises until in about 5 min., it becomes constant or approximately so. The difference between inlet and outlet waters should be about 20° C., and the temperature of the outlet water should not exceed 35° C., as otherwise there is liability to loss by vaporization.

The temperature of the cooling water having also become constant, the hot-water tube is shifted over into the 2-liter graduate as soon as the pointer of the

meter has come to the zero-point. Readings of both thermometers are taken every time 0.5 liter of cooling water has been collected in the graduate. Minute-readings are sufficient with runs lasting from 5 to 8 min. when the overflow water is weighed. As soon as the 2-liter graduate is filled the gas is shut off. Enough gas must have been burned to give from 1,000 to 1,500 cal. per hour, *i.e.*, 0.5 cbm. for illuminating gas, 1 cbm. for producer gas. If H = C. P. of

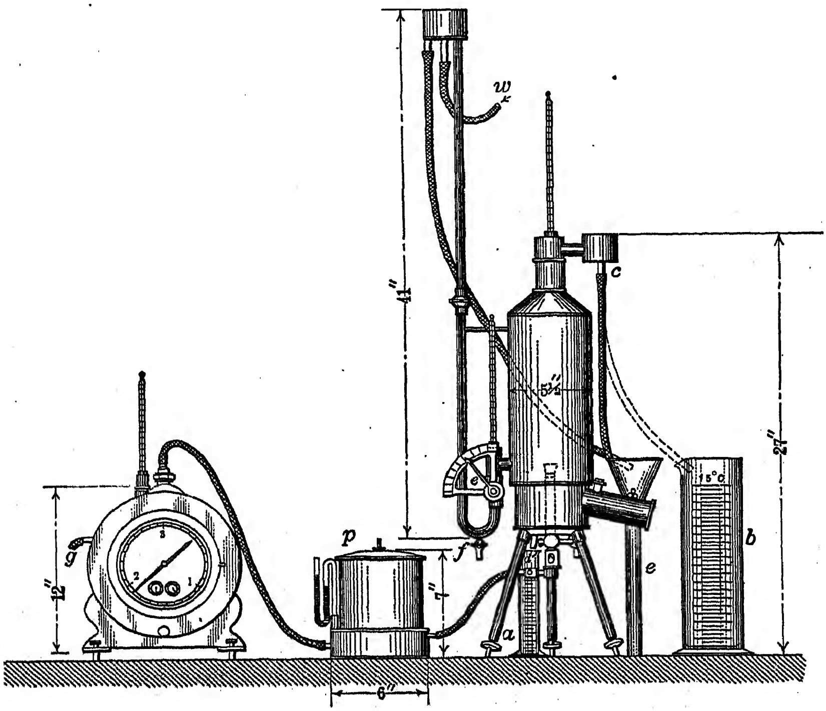


FIG. 50.—The Junkers calorimeter.

1 liter gas in cal.; W=liters or kg. water passing per min.; t=average temp. of inlet water; T=average temp. of outlet water; G=liters gas burned; d=g. water condensed per min.; then:

$$H=\frac{W(T-t)-537d}{G}.$$

As the water formed in combustion is not condensed in metallurgical operation, but passes off as vapor, the latent heat (537 cal.) of the condensed water (d) has to be deducted from the total heat of combustion measured by the calorimeter.

The Junkers calorimeter can also be used for determining the calorific power of oil or gasolene. In this case the gas-meter and regulator are replaced by a lamp, which rests on platform scales while burning, and thus allows noting the loss in weight.

EXAMPLE

	Run I	Run II	Run III
Weight of water per minute..............	2,011 g.	2,014 g.	2,013 g.
Weight of drip per minute....	3.21 g.	3.00 g.	2.71 g.
Liters of gas...............	3.26	3.13	2.97
Initial temperature of water.	17.21 °C.	17.21 °C.	17.44° C.
Final temperature of water.	27.28° C.	26.84° C.	26.56° C.
Gram-calories per liter of gas	5,683	5,682	5,692
B.T.U. per cu. ft. of gas......	639.3	639.2	640.4

95. **Calorific Intensity.**[1]—The unit serving for measuring the calorific intensity is the degree Centigrade or Fahrenheit scale. The former will be used exclusively. The calorific intensity of a substance can be ascertained by calculation or determined by measurement.

96. **Calculation of Calorific Intensity.**[2]—This is based upon the assumption that all the heat generated in perfect combustion is taken up by the products of combustion, *i.e.*, none is lost by radiation or conduction. The temperature reached will, therefore, depend upon the calorific power and the weights and specific heats of the products of combustion. Let C be the calorific power of a fuel, T the rise in temperature, w, w', w'' the weights of the products of combustion and s s' s'' their specific heats; then:

$$C = T(w\,s + w'\,s' + w''\,s'' + \ldots), \text{ and } T = \frac{C}{w\,s + w'\,s' + w''\,s'' + \ldots}$$

for a fuel at 0° C. and 760 mm. pressure. The formula shows that the calorific intensity is proportional to the calorific power and indirectly proportional to the weights of the products of combustion and their specific heats. The calorific intensity can be raised by superheating fuel and air; by depressing the weights of the products of combustion, *i.e.*, using the minimum excess air, or possibly diminishing the N in the air; by excluding moisture from the fuel, and lastly by diminishing the percentage of ash in the fuel (coal-washing). The above formula cannot be used conveniently for the calculation of the calorific intensity, as the specific heats of the products of combustion are not constant, but increase with the temperature. Thus the specific heat at $t°$ of N, O, H, CO $= 0.303 + 0.000054\ t$; of $CO_2 = 0.37 + 0.00044\ t$; of $H_2O = 0.34 + 0.00030\ t$. The mean specific heat between 0° and $t°$ is, of course, the constant plus the mean of the factor, *e.g.*, of N $= 0.303 + 0.000027\ t$; and the mean specific heat between t and t_1, *e.g.*, of N $= 0.303 + 0.000027\ (t_1 + t)$.

For the formula of $T = \dfrac{C}{w\,s + w'\,s' + w''\,s''}$ given, one can substitute the

[1] Flame temperatures: Harker, *J. Soc. Chem. Ind.*, 1912, XXXI, 307.

[2] Ehrenwerth, *Metallurgie*, 1909, VI, 305.

quantity of heat needed to raise 1 cbm. (at 760 mm.) of products of combustion from 0° to t°, that is, the mean specific heats of the various constituents of the product of combustion. These are: $0.303+0.000027\ t$ for N, O, H and CO; $0.37+0.00022\ t$ for CO_2; and $0.34+0.00015\ t$ for H_2O.

EXAMPLE FOR SOLID FUEL.—What is the calorific intensity of carbon burning with the theoretical amount of pure oxygen? One kilogram of C will develop on combustion 8,100 Cal., and this heat will be absorbed by the CO_2 formed by the combustion. According to the equation $C+O_2=CO_2$, 12 kg. C require one molecular volume of O_2, and form one molecular volume of CO_2. Then 1 kg. C will require one-twelfth of one molecular volume of O_2, or 1.86 cbm., and will form an equal volume of CO_2. The heat required to raise this volume of CO_2 one degree is $1.86\ (0.37+0.00022\ t)$ Cal., or for the entire temperature rise of t°, $1.86\ (0.37+0.00022\ t)\ t$ Cal. This value must be equal to the total amount of heat available from the combustion, *i.e.*, 8,100 Cal. Equating the two gives $1.86\ (0.37+0.00022 \cdot t^2)=0.688\ t+0.000409\ t^2=8{,}100$; and solving for t, gives $t=3{,}689°$.

TABLE 64.—RELATIONS OF VOLUMES OF COMPONENTS AND PRODUCTS OF COMBUSTION

In order to form 2 vol.	There are required, vol.		
	O	H	N
CO_2	2		
CO	1		
CH_4		4	
H_2O	1	2	
NH_3		3	1
C_6H_6		6	

If 1 kg. C is burned with the theoretical amount of air, instead of with pure oxygen, the products of combustion are $1.86\ CO_2+7.086\ N_2$. (The composition of air is O_2 20.8 per cent., and N_2 79.2 per cent. by volume; the ratio, $O_2 : N_2=1 : 3.81$.) This changes the equation above to $1.86(0.37+0.00022\ t^2)+7.086(9.303\ t+0.000027\ t^2)=2.836\ t+0.00060\ t^2=8{,}100$. Solving for t, gives us $t=2{,}005°$.

Experience has shown that with solid fuel, air to the extent of at least 50 per cent. more than is theoretically required must frequently be supplied. Supposing this to be the case, the products of combustion from 1 kg. C will be $1.86\ CO_2+7.086\ N_2+4.473$ excess air. Since the mean specific heat of air is the same as that of N_2, the equation becomes $1.86(0.37\,t+0.00022\,t^2)+11.559\ (0.303\ t+0.000027\ t^2)=4.191\ t+0.000721\ t^2=8{,}100$. Solving for t, gives $t=1{,}530°$.

The difference between 2,005° and 1,530° shows how essential it is to control the admission of air in firing.

In the preceding calculations it has been assumed that the C, O, and air were at 0° C. A simple means of obtaining a higher temperature is to preheat

the fuel, the air, or both. Natural gas being decomposed by heat cannot be superheated. Supposing the producer gas given on p. 118 is to be burned with 25 per cent. excess air, preheated to 500° C., what will be the calorific intensity?

EXAMPLE

1 cbm. gas	Needs cbm O(1)	And produces cbm. CO2(1)	H2O(1)	N(1)
N, 0.602				0.602
CO_2, 0.040		0.040		
CO, 0.189	0.095	0.189		
H, 0.126	0.063		0.126	
C_2H_4, 0.027	0.081	0.054	0.054	
CH_4, 0.016	0.032	0.016	0.032	
1.000	0.271	0.299	0.212	0.602
Corresponding air	1.303			1.032
Excess air, 25 per cent	0.3257			1.032
Total	1.6287 air	0.299	0.212	1.9597

The calorific power of 0.8881 cbm. producer gas, p. 119, was calculated to be 1,272 Cal., hence the calorific power of 1 cbm. will be 1,272 : 0.8881 = 1,432 Cal. Supposing the gas and air to be at 0° C., then

$$\begin{aligned} N+C &= 1.9597(0.303\,t+0.000027\,t^2) = 0.594\,t+0.000053\,t^2 \\ H_2O &= 0.212\ (0.34\,t+0.00015\,t^2) = 0.072\,t+0.000032\,t^2 \\ CO_2 &= 0.299\ (0.37\,t+0.00022\,t^2) = 0.111\,t+0.000066\,t^2 \\ \text{Total} & \qquad 0.777\,t+0.000151\,t^2 = 1,432 \\ & \qquad t = 1,440^\circ \text{ C.} \end{aligned}$$

It has been calculated above that theoretically there are required 1.303 cbm. of air to burn the gas. To this has to be added 25 per cent. excess, or 0.3257 cbm., making a total of 1.6287 cbm. The quantity of heat gained by heating this air to 500° C. is: $1.6287(0.303\times500+0.000027\times500^2) = 257.7$ Cal. This makes the total available heat 1,432+257.7 = 1,689.7 Cal.; and the equation becomes $0.777\,t+0.000151\,t^2 = 1,689.7$, from which $t = 1,647^\circ$ C. The calorific intensity then has been raised from 1,440° to 1,647° by the preheating.

97. Determination of the Calorific Intensity (Pyrometry).[2]—Almost every

[1] Table 64.

[2] Burgess, G. K. and Le Chatelier, H. "The Measurement of High Temperatures," Wiley, New York, 1912.

Barus, C. "On the Thermoelectric Measurement of High Temperatures," *Bull.* 54, U. S. Geological Survey, Washington, 1889.

Report on Pyrometers Suitable for Metallurgical Work, *J. I. and St. I.*, 1904, 1, 98.

Waidner, C. W., Methods of Pyrometry, *Proc. Eng. Soc., West. Pa.*, 1904, xx, 313.

thermal property of matter has been suggested as a basis for the construction of pyrometers. Here only the pyrometers that have proved serviceable for metallurgical work will be reviewed, and that briefly.

1. Expansion Pyrometers.—Many pyrometers have been devised which are based upon the linear dilatation of one or two solids. They have been made principally of metal and of carbon. They all suffer from the fact that the expansion is not uniform, and that after repeated exposure to heat they do not contract to their original lengths. Pyrometers of this class were extensively used for the measurement of the hot-blast of iron blast-furnaces, but they have had to yield to apparatus based upon other principles. The leading instruments of these so-called mechanical pyrometers are those of Gauntlett[1] (a copper tube enclosed in an iron tube, the two connected at their lower ends), Steinle and Hartung[2] (iron and gas-carbon rods), and Brown[3] (a light strip of platinum).

The usual representative for the dilatation of liquids is the mercury thermometer which serves to measure temperatures up to 360° C., the boiling-point of mercury. In order to permit measurement of higher temperatures of furnace or boiler flues, the vacuum has been filled with CO_2 or N. With borosilicate glass(Jena 59^{III}), temperatures up to 550° C. can be measured. With quartz tubes filled at 60 atmospheres pressure, 750° C. is the present limit. An "Industrial Thermometer" of the Hohmann and Maurer Mfg.Co., Rochester, N. Y., is shown in Fig. 51.

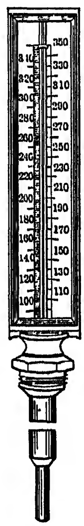


Fig. 51.—The Hohmann and Maurer industrial thermometer.

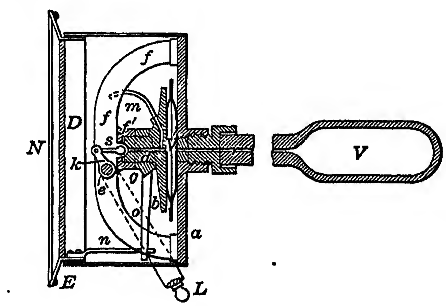


Fig. 52.—The Wiborgh air pyrometer.

As regarding principle involved, pyrometers based upon the expansion of gases are most satisfactory, since the volume occupied by the more stable gases is proportional to the absolute temperature and the coefficient of expansion is large. One of the difficulties of applying the principle lies in the fact that the expansion of the enclosing vessel has to be considered and that the material of the vessel is likely to be brittle and to fuse (glass, porcelain), or to be perme-

[1] Dingler, *Pol. J.*, 1860, CLVII, 259.

[2] Bolz, C. H., "Die Pyrometer," Springer, Berlin, 1888.

[3] *J. Frankl. Inst.*, 1894, CXXXVII, 480; *Bull. Iron and Steel Assoc.*, 1894, XVIII, 113.

able to gases (Pt,Fe). The only air pyrometer that has found technical application, and this more in Europe than the United States, is the constant-volume pyrometer of Wiborgh.[1] In it a known volume of air is forced into a porcelain bulb that has been raised to the temperature to be measured, and then the corresponding rise in pressure noted. The aneroid form is shown in Figs. 52 and 53. The porcelain tube *V*, which is wound with asbestos to prevent its cracking when exposed suddenly to high temperatures and ends in a capillary

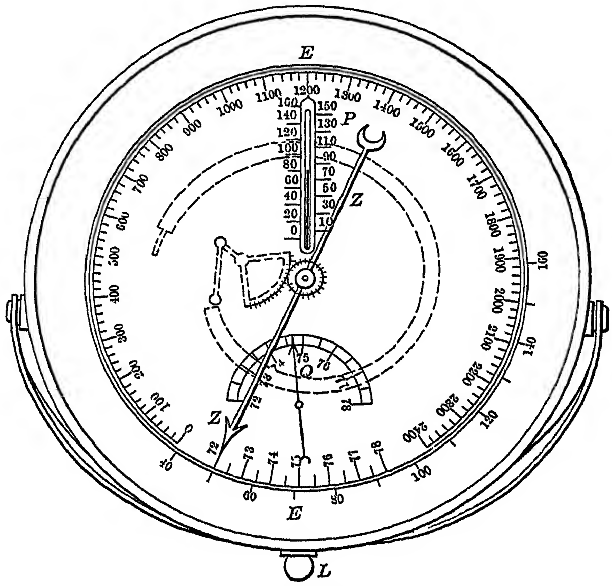


FIG. 53.—The Wiborgh air pyrometer.

tube *r*, is screwed into the bottom-plate *a* of the circular brass case holding the measuring apparatus. To *a* is fastened an elastic lenticular metal vessel *V′* which can be completely compressed by the metal plate *b*, but will resume its original shape when the pressure is released. The capillary tube *r* joins *V* and *V′* and, passing through plate *b* and its stem *d*, connects them with the outside air. Tube *r* is also connected by means of a capillary lead tube *m* with a manometer spring (dotted line in Fig. 53) which through a toothed sector actuates the hand *Z* indicating the temperature on the dial. To the plate *a* is screwed an iron bridge for supporting the axle *e*, the ends of which pass through the bearings in the sides of the metal box. The ends are joined by a fork-shaped lever *g* provided with a small handle *L*. Near the center, the axle has a short arm *k* actuating the small rod *s*. When vessel *V′* is to be compressed, handle *L* is raised; this turns the axle, causes rod *s* to descend, closes the capillary tube, and forces all the air out of *V′* into *V*. The small amount of residual air in the lead pipe *m* can be neglected. When the pressure is released, arm *L* is forced back into the

[1] Trotz, *Tr. A. I. M. E.*, 1892–93, XXI, 592.
Wiborgh, *Stahl u. Eisen*, 1891, XI, 913.

position represented in Fig. 52 by a spiral wound around the axle and attached to the bridge *f* (not shown). This is essential, as vessels *V* and *V′* should communicate with the outside air when no observations of temperature are being made.

In order to permit making the necessary correction of volume of *V′* for the varying pressure and temperature of the outside air, the stem *d* is enclosed in a ring *g* which when turned raises or lowers the plate *b* and thus increases or decreases the volume of *V′*. To the ring *g* is attached the arm *o* clasped by the two-prong fork *n* projecting downward from the ring *E* which carries the glass *N*. The dial-plate *D* (fastened to the bridge in a manner not seen in the figure) shows the hand *z* and the temperature scale 0–2,400° F., the aneroid barometer *Q* with its scale, 72–78 dcm., and a thermometer *P* with the Fahrenheit scale. On the movable ring *E* lastly, between *N* and *D*, is drawn a temperature scale (40 to 160° F.) to make the necessary temperature correction. When an observation is to be made, the porcelain tube *V* is exposed to the heat, the temperature of the air is read on the thermometer *P*, the degrees F. are then found on the temperature scale of ring *E*, and this is turned until the temperature degree coincides with the number corresponding to the pressure number seen on the aneroid barometer *Q*. When this is done, the thumb is placed on the dial-glass, and the handle *L* raised slowly with the forefinger as far as it will go; it is held in this position until the hand *z* comes to a standstill, which takes only a few minutes.

The theory of the instrument is discussed in the references.

The pyrometer is convenient for measuring temperatures up to 1,400° C., and the temperatures indicated agree[1] with those obtained with the Le Chatelier thermoelectric pyrometer (page 145), but it suffers from the defect of having a porcelain tube which is fragile and likely to crack.

2. Fusion Pyrometers.—These are simple and convenient as long as the materials used have fixed melting-points. Metals, alloys, salts and mixtures have been and are still used to some extent. In taking a temperature, several substances are placed in one or more vessels of refractory material (bone-ash, clay, iron), to hold and protect them from the flame, and exposed to the heat; the temperature sought is that at which fusion ceases. These pyrometers are suitable for intermittent heatings which demand a constantly rising temperature up to a certain maximum, when the source of heat is shut off and the furnace allowed to cool down slowly. They cannot indicate any decrease in temperature. Base metals which were used once quite frequently have the disadvantage of becoming oxidized. Precious metals (Pt,Pd,Au,Ag) give only a small range of temperature. Alloys which are not eutectics or solid solutions have no definite melting-points. This throws out a large number of mixtures which are frequently recommended. Base-metal alloys are likely to become oxidized, precious-metal alloys are satisfactory within certain limits. Erhard and Schertel[2] made up the list given in Table 65. Ag and Au form an unbroken series of solid

[1] Blass, *Stahl u. Eisen*, 1892, XII, 893.

[2] *Freiberg Jahrb.*, 1897, 154.

solutions, but Au and Pt do not. Seger[1] found that Au–Pt mixtures were satisfactory up to 1,200° C., but that alloys with over 15 per cent. Pt did not remain homogeneous; further Pt is likely to be affected by C and Si.

TABLE 65.—ERHARD AND SCHERTEL FUSION PYROMETERS

		Deg. C.			Deg. C.
100Ag	....	954	60Au	40Pt	1,320
80Ag	20Au	975	55Au	45Pt	1,350
60Ag	40Au	995	50Au	50Pt	1,385
40Ag	60Au	1,020	45Au	55Pt	1,420
20Ag	80Au	1,045	40Au	60Pt	1,460
....	100Au	1,075	35Au	65Pt	1,495
95Au	5Pt	1,100	30Au	70Pt	1,535
90Au	10Pt	1,130	25Au	75Pt	1,570
85Au	15Pt	1,160	20Au	80Pt	1,610
80Au	20Pt	1,190	15Au	85Pt	1,650
75Au	25Pt	1,220	10Au	90Pt	1,690
70Au	30Pt	1,255	5Au	95Pt	1,730
65Au	35Pt	1,285	....	100Pt	1,775[2]

Many metallic salts are readily fused without being decomposed. They would be used more generally as pyrometers were it not for the difficulty of finding a vessel to hold them which is not attacked and which is sufficiently cheap. A few salts[3] with their melting-points and some mixtures[4] are collected in Table 66 with a view to recording the melting-points rather than of recommending them as pyrometers.

TABLE 66.—METALLIC SALTS AS FUSION PYROMETERS

Salt	Melting-point, deg. C.	Salt	Melting-point, deg. C.
K_2SO_4	1,070	KBr	730
$BaCl_2$	955	KI	682
Na_2SO_4	865	$5.8KCl+4.2NaCl$	655
$5K_2SO_4+5Na_2SO_4$	850	$3NaCl+7KBr$	625
$3K_2SO_4+7Na_2SO_4$	830	$Ba(NO_3)_2$	600
$2K_2SO_4+8Na_2SO_4$	825	$5KCl+5K_2CO_3$	580
Na_2CO_3	810	$3Na_2CO_3+3K_2CO_3+2NaCl+2KCl$	560
NaCl	800	$Ca(NO_3)_2$	550
KCl	775	$3K_2SO_4+3Na_2SO_4+2NaCl+2KCl$	520

[1] *Thonind. Z.*, 1885, IX, 104, 121; *Berg. Hüttenm. Z.*, 1885, XLIV, 303.
[2] 1,755° C. is now held to be the correct figure.
[3] Grenet, *Rev. Mét.*, 1910, VII, 485; *Metallurgie*, 1910, VII, 721.
[4] Brearley-Moorwood, *J. I. and St. I.*, 1907, I, 268.

SEGER CONES.[1]—These are the most important fusion pyrometers, or rather pyroscopes. In order to obtain fusion pyrometers for temperatures at which the Au-Pt alloys of Erhard and Schertel showed segregation, the late Seger used mixtures employed in the glazing of porcelain. The most fusible is made up of $\left.\begin{matrix} 0.3\ K_2O \\ 0.7\ CaO \end{matrix}\right\} 0.5Al_2O_3.4SiO_2$. He increased its fusibility by a partial substitution of Fe_2O_3 for Al_2O_3 and obtained a mixture $\left\{\begin{matrix} 0.3\ K_2O \\ 0.7\ CaO \end{matrix}\right\} \left\{\begin{matrix} 0.3Al_2O_3 \\ 0.2Fe_2O_3 \end{matrix}\right\} 4SiO_2$ which melted at the same temperature as the alloy Au 90 : Pt 10, or at 1,150° C. This forms No. 1 of his series. He then diminished the fusibility of his mixture, first by increasing the ratio of $Al_2O_3:Fe_2O_3$ and then by dropping the Fe_2O_3 altogether. After this he increased the ratio of $SiO_2:Al_2O_3$ until mixture No. 27 was reached, which melted at about 1,670° C. With mixtures Nos. 28 to 36, melting at temperatures ranging from 1,690 to 1,850° C., the bases K_2O and CaO were discarded, and the ratio $Al_2O_3:SiO_2$ slowly increased. These mixtures proved so satisfactory that Cramer and Hecht prepared others for lower temperatures, ranging from 1,150° C. to the melting-point of Ag, 950° C., by replacing part of the SiO_2 with B_2O_3, and numbered them 010 to 01; finally, for mixtures below the melting-point of Ag down to 590° C. Hecht replaced the CaO by PbO, and the K_2O by Na_2O; he lastly dropped the Fe_2O_3 and decreased the Al_2O_3, and numbered these cones 022 to 011.

TABLE 67.—SEGER CONES, ORIGINAL SERIES

No. of cone	Composition			Melting-point
				Deg. C.
022	$\left\{\begin{matrix} 0.5\ Na_2O \\ 0.5\ PbO \end{matrix}\right\}$	....	$\left\{\begin{matrix} 2.0\ SiO_2 \\ 1.0\ B_2O_3 \end{matrix}\right\}$	590
021	$\left\{\begin{matrix} 0.5\ Na_2O \\ 0.5\ PbO \end{matrix}\right\}$	$0.10\ Al_2O_3$	$\left\{\begin{matrix} 2.2\ SiO_2 \\ 1.0\ B_2O_3 \end{matrix}\right\}$	620
020	$\left\{\begin{matrix} 0.5\ Na_2O \\ 0.5\ PbO \end{matrix}\right\}$	$0.20\ Al_2O_3$	$\left\{\begin{matrix} 2.4\ SiO_2 \\ 1.0\ B_2O_3 \end{matrix}\right\}$	650
019	$\left\{\begin{matrix} 0.5\ Na_2O \\ 0.5\ PbO \end{matrix}\right\}$	$0.30\ Al_2O_3$	$\left\{\begin{matrix} 2.6\ SiO_2 \\ 1\ 0\ B_2O_3 \end{matrix}\right\}$	680
018	$\left\{\begin{matrix} 0.5\ Na_2O \\ 0.5\ PbO \end{matrix}\right\}$	$0.40\ Al_2O_3$	$\left\{\begin{matrix} 2.8\ SiO_2 \\ 1.0\ B_2O_3 \end{matrix}\right\}$	710
017	$\left\{\begin{matrix} 0.5\ Na_2O \\ 0.5\ PbO \end{matrix}\right\}$	$0.50\ Al_2O_3$	$\left\{\begin{matrix} 3.0\ SiO_2 \\ 1.0\ B_2O_3 \end{matrix}\right\}$	740
016	$\left\{\begin{matrix} 0.5\ Na_2O \\ 0.5\ PbO \end{matrix}\right\}$	$0.55\ Al_2O_3$	$\left\{\begin{matrix} 3.1\ SiO_2 \\ 1.0\ B_2O_3 \end{matrix}\right\}$	770

[1] Hofman, *Tr. A. I. M. E.*, 1894, XXIV, 54; 1895, XXV, 8; 1899, XXIX, 686.
Gorton, *Tr. Am. Ceram. Soc.*, 1900, II, 60.
Ashley, *op. cit.*, 1906, VIII, 158.
Seger-Cramer, *Thonind. Z.*, 1904, XXVIII, 1315.
Simonis, *op. cit.*, 1908, XXXII, 1764.
Hoffmann, *op. cit.*, 1909, XXXIII, 1577; *Stahl-Eisen*, 1909, XXIX, 440.
Ricke, *Thonind. Z.*, 1911, XXXV, 1751.
Geijsbeck, *Tr. Am. Ceram. Soc.*, 1912, XIV, 849.

TABLE 67.—SEGER CONES, ORIGINAL SERIES—(*Continued*)

No. of cone	Composition			Melting-point
				Deg. C.
015	0.5 Na_2O 0.5 PbO	0.60 Al_2O_3	3.2 SiO_2 1.0 B_2O_3	800
014	0.5 Na_2O 0.5 PbO	0.65 Al_2O_3	3.3 SiO_2 1.0 B_2O_3	830
013	0.5 Na_2O 0.5 PbO	0.70 Al_2O_3	3.4 SiO_2 1.0 B_2O_3	860
012	0.5 Na_2O 0.5 PbO	0.75 Al_2O_3	3.5 SiO_2 1.0 B_2O_3	890
011	0.5 Na_2O 0.5 PbO	0.80 Al_2O_3	3.6 SiO_2 1.0 B_2O_3	920
010	0.3 K_2O 0.7 CaO	0.20 Fe_2O_3 0.30 Al_2O_3	3.50 SiO_2 0.50 B_2O_3	950
09	0.3 K_2O 0.7 CaO	0.20 Fe_2O_3 0.30 Al_2O_3	3.55 SiO_2 0.45 B_2O_3	970
08	0.3 K_2O 0.7 CaO	0.20 Fe_2O_3 0.30 Al_2O_3	3.60 SiO_2 0.40 B_2O_3	990
07	0.3 K_2O 0.7 CaO	0.20 Fe_2O_3 0.30 Al_2O_3	3.65 SiO_2 0.35 B_2O_3	1,010
06	0.3 K_2O 0.7 CaO	0.20 Fe_2O_3 0.30 Al_2O_3	3.70 SiO_2 0.30 B_2O_3	1,030
05	0.3 K_2O 0.7 CaO	0.20 Fe_2O_3 0.30 Al_2O_3	3.75 SiO_2 0.25 B_2O_3	1,050
04	0.3 K_2O 0.7 CaO	0.20 Fe_2O_3 0.30 Al_2O_3	3.80 SiO_2 0.20 B_2O_3	1,070
03	0.3 K_2O 0.7 CaO	0.20 Fe_2O_3 0.30 Al_2O_3	3.85 SiO_2 0.15 B_2O_3	1,090
02	0.3 K_2O 0.7 CaO	0.20 Fe_2O_3 0.30 Al_2O_3	3.90 SiO_2 0.10 B_2O_3	1,110
01	0.3 K_2O 0.7 CaO	0.20 Fe_2O_3 0.30 Al_2O_3	3.95 SiO_2 0.05 B_2O_3	1,130
1	0.3 K_2O 0.7 CaO	0.20 Fe_2O_3 0.30 Al_2O_3	4 SiO_2	1,150
2	0.3 K_2O 0.7 CaO	0.10 Fe_2O_3 0.40 Al_2O_3	4 SiO_2	1,170
3	0.3 K_2O 0.7 CaO	0.05 Fe_2O_3 0.45 Al_2O_3	4 SiO_2	1,190
4	0.3 K_2O 0.7 CaO	0.5 Al_2O_3	4 SiO_2	1,210
5	0.3 K_2O 0.7 CaO	0.5 Al_2O_3	5 SiO_2	1,230
6	0.3 K_2O 0.7 CaO	0.6 Al_2O_3	6 SiO_2	1,250
7	0.3 K_2O 0.7 CaO	0.7 Al_2O_3	7 SiO_2	1,270
8	0.3 K_2O 0.7 CaO	0.8 Al_2O_3	8 SiO_2	1,290

TABLE 67.—SEGER CONES, ORIGINAL SERIES—(*Continued*)

No. of cone	Composition					Melting-point
						Deg. C.
9	0.3 K_2O, 0.7 CaO	0.9	Al_2O_3	9	SiO_2	1,310
10	0.3 K_2O, 0.7 CaO	1.0	Al_2O_3	10	SiO_2	1,330
11	0.3 K_2O, 0.7 CaO	1.2	Al_2O_3	12	SiO_2	1,350
12	0.3 K_2O, 0.7 CaO	1.4	Al_2O_3	14	SiO_2	1,370
13	0.3 K_2O, 0.7 CaO	1.6	Al_2O_3	16	SiO_2	1,390
14	0.3 K_2O, 0.7 CaO	1.8	Al_2O_3	18	SiO_2	1,410
15	0.3 K_2O, 0.7 CaO	2.1	Al_2O_3	21	SiO_2	1,430
16	0.3 K_2O, 0.7 CaO	2.4	Al_2O_3	24	SiO_2	1,450
17	0.3 K_2O, 0.7 CaO	2.7	Al_2O_3	27	SiO_2	1,470
18	0.3 K_2O, 0.7 CaO	3.1	Al_2O_3	31	SiO_2	1,490
19	0.3 K_2O, 0.7 CaO	3.5	Al_2O_3	35	SiO_2	1,510
20	0.3 K_2O, 0.7 CaO	3.9	Al_2O_3	39	SiO_2	1,530
21	0.3 K_2O, 0.7 CaO	4.4	Al_2O_3	44	SiO_2	1,550
22	0.3 K_2O, 0.7 CaO	4.9	Al_2O_3	49	SiO_2	1,570
23	0.3 K_2O, 0.7 CaO	5.4	Al_2O_3	54	SiO_2	1,590
24	0.3 K_2O, 0.7 CaO	6.0	Al_2O_3	60	SiO_2	1,610
25	0.3 K_2O, 0.7 CaO	6.6	Al_2O_3	66	SiO_2	1,630
26	0.3 K_2O, 0.7 CaO	7.2	Al_2O_3	72	SiO_2	1,650
27	0.3 K_2O, 0.7 CaO	20	Al_2O_3	200	SiO_2	1,670
28	....	1	Al_2O_3	10	SiO_2	1,690
29	....	1	Al_2O_3	8	SiO_2	1,710
30	....	1	Al_2O_3	6	SiO_2	1,730
31	....	1	Al_2O_3	5	SiO_2	1,750
32	....	1	Al_2O_3	4	SiO_2	1,770

TABLE 67.—SEGER CONES, ORIGINAL SERIES—(*Continued*)

No. of cone	Composition			Melting-point
				Deg. C.
33	....	1 Al_2O_3	3 SiO_2	1,790
34	....	1 Al_2O_3	2.5 SiO_2	1,810
35	....	1 Al_2O_3	2 SiO_2	1,830
36	....	1 Al_2O_3	1.5 SiO_2	1,850
37	....	1 Al_2O_3	1.33 SiO_2	1,870
38	....	1 Al_2O_3	1 SiO_2	1,890
39	....	1 Al_2O_3	0.66 SiO_2	1,910
40	....	1 Al_2O_3	0.40 SiO_2	1,930
41	....	1 Al_2O_3	0 13 SiO_2	1,950
42	....	Al_2O_3	. ... .	1,970

The original series did not meet all the requirements in practical use. Investigations with a view to increasing the sharpness of the melting-points have resulted in supplanting the original series by a new one, in which PbO and Fe_2O_3 have been omitted as fluxes. The new series is given in Table 68. The compositions are given in the paper by Simonis.

TABLE 68.—SEGER CONES, NEW SERIES

Cone number	Melting-point, deg. C.	Cone number	Melting-point, deg. C.	Cone number	Melting-point, deg. C.	Cone number	Melting-point, deg. C.
022	600	07*a*	960	9	1,280	29	1,650
021	650	06*a*	980	10	1,300	30	1,670
020	670	05*a*	1,000	11	1,320	31	1,690
019	690	04*a*	1,020	12	1,350	32	1,710
018	710	03*a*	1,040	13	1,380	33	1,730
017	730	02*a*	1,060	14	1,410	34	1,750
016	750	01*a*	1,080	15	1,435	35	1,770
015*a*	790	1*a*	1,100	16	1,460	36	1,790
014*a*	815	2*a*	1,120	17	1,480	37	1,825
013*a*	835	3*a*	1,140	18	1,500	38	1,850
012*a*	855	4*a*	1,160	19	1,520	39	1,880
011*a*	880	5*a*	1,180	20[1]	1,530	40	1,920
010*a*	900	6*a*	1,200	26	1,580	41	1,960
09*a*	920	7	1,230	27	1,610	42	2,000
08*a*	940	8	1,250	28	1,630		

[1] It will be noticed that numbers 21–25 have been dropped, the reason being that their melting-points lay too close together.

Seger cones are three-sided pyramids; Nos. 022–25 are 3 in. high with a base 5/8 in. wide (Fig. 54), Nos. 26–42 are 25/32 in. high with a base 3/8 in. wide. Each cone has its number stamped on the side which usually remains uppermost when the cone bends over in melting. The temperature indicated for each cone in Tables 67 and 68 is only approximate; it is supposed to have been reached when the apex is half bent over. While a Seger cone is not an accurate pyrometer,[1] it is a most valuable instrument for controlling the working of an intermittent furnace in which mechanical mixtures are to be converted into chemical compounds by heating, as the success of such an operation[2] is not only a function of temperature, but depends also upon the manner of firing, the time given, and the form of the furnace. The Seger cone, being exposed to the same influences as the substance under treatment, shows by its fusion that sufficient heat and time have been given to complete the process.

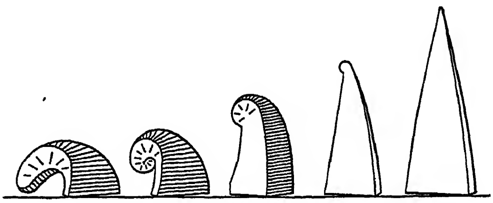


FIG. 54.—Large-size Seger cones.

3. Calorimetric Pyrometers.—The principle of measurement involved is that of the method of mixtures: A small body (the "heat carrier" or "pyrometric substance," a cylinder or ball of Fe, Cu, Ni or Pt) heated to the temperature to be measured is mixed with a large body (water) in such proportions that the rise in temperature of the latter can be measured with the mercury thermometer.

Let W be the weight of the heat carrier; w the weight of the water in the calorimeter; s the specific heat of the calorimeter; t_1 the initial and t_2 the final temperature of the water; T the unknown temperature; s_T the mean specific heat of the heat carrier between 0 and $T°$, and s_t that between 0 and $t_2°$; and finally c the weight of the calorimeter. The heat lost by the heat carrier $Ws_T(T-t_2)$ is equal to the heat taken up by the water $(t_2-t_1)\times(w+cs)$, whence

$$Ts_T = \frac{(t_2-t_1)(w-cs)}{W} + t_2 s_t$$

The values of the right side of the equation are found in the experiment, those of the left side have been determined by Violle and others for Fe, Cu, Ni and Pt over a wide range of temperature. By interpolation, the value of T may thus be calculated.

In the "Siemens Water-pyrometer" the ratio $\frac{w-cs}{W}$ is so chosen as to be equal to 50; the multiplication of (t_2-t_1) by 50 is indicated on a sliding scale so that the temperature T is obtained by adding this product to t_2. The calorim-

[1] *Thonind. Z.*, 1907, xxxi, 1366 (Rothe), 1404 and 1416 (Loeser).

[2] Ebeling, *Thonind. Z.*; 1895, xix, 803.

Seger-Cramer, *op. cit.*, 1893, xvii, 1344, with illustration of tray used for placing large-size cones in furnace.

eter shown in Figs. 55–56 consists of a cylindrical copper vessel *a* with handle containing a second small copper vessel *b*, 4¼ in. in diam. and 10.5 in. high, of a capacity of a little more than 1 pint water, surrounded by a layer of felt *c*. The mercury thermometer *d*, fixed close to the wall, has its lower end protected by a perforated tube *e*; on the upper side is a sliding brass scale *f*, 6 3/4 in. long, with pointer; its high-temperature markings are in the same degrees as those of the thermometer, but are adjusted to the metal to be used as the heat carrier and the increase of its specific heat with the temperature. In making a measurement, the calorimeter is filled with exactly 1 pint (0.568 liter) of clear water of about 5° below room temperature (distilled water has the advantage in that it makes it easier to keep the vessel clean), and left to itself to allow the thermometer to attain the temperature of the water. Then the pointer of the scale *f* is set opposite the degree indicated by the thermometer. In the mean time the metal cylinder *g* (Fe 112, Ni 117, Cu 137, Pt 402.6 g) has been placed in the furnace, the temperature of which is to be measured, and kept there for about 20 min.; it is removed quickly with cup-shaped tongs which have been heated to the temperature of the cylinder and dropped into the vessel, care being taken not to splash over any water. The temperature of the water rises

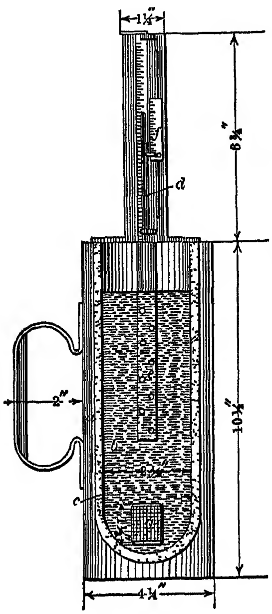


FIG. 55.

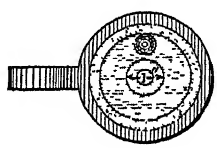


FIG. 56.

FIGS. 55 and 56.—The Siemens calorimeter.

and when equilibrium has been reached, the degrees of the thermometer and of the brass scale opposite are read; the sum of the two gives the temperature of the furnace. Cylinders of Fe and Cu can be used up to 1,000° C., of Ni up to 1,400° C., and of Pt up to 1,500° C. Cylinders of Cu scale easily, those of Fe become oxidized although less readily than those of Cu; both have to be weighed at intervals to ascertain the loss in weight by scaling. A table accompanies the apparatus showing the percentages by which the readings on the brass scale have to be increased before they may be added to the degrees of the thermometer.

This pyrometer was used quite extensively until within a comparatively short time; at present its use is restricted to plants which make measurements at infrequent intervals and desire an apparatus that is low in price. The method is reliable if sufficient care is taken; it is most valuable in ascertaining the actual temperatures of large pieces of metal not too unwieldy to be

dropped in a vessel filled with water. In such cases a mechanical stirrer will be necessary.

EXAMPLE.—Determination of the melting-point of aluminum.

Weight of platinum ball 179.3 gms.
Weight of can and stirrer 268 gms.
Weight of water 1,200 gms.
Rate of temperature change before mixture +.005.
Rate of temperature change after maximum temperature —.005.
Cooling correction 0.00.
Corrected rise of temperature 3.35.
Ts_t 23.70.
Temperature corresponding, 660° C.

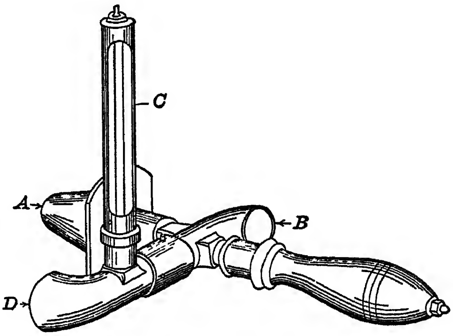


FIG. 57.—The Hobson hot-blast pyrometer.

The method of mixtures has been applied to determine the temperature of the hot-blast of an iron blast-furnace by mixing it with sufficient air of atmospheric temperature to permit measuring with the mercury thermometer.

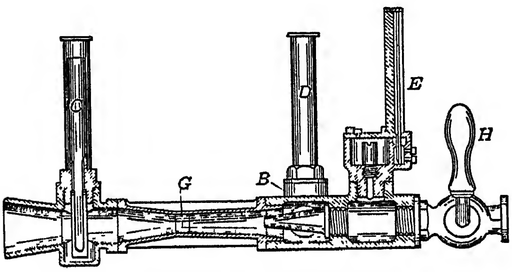


FIG. 58.—The Krupp hot-blast pyrometer.

The first apparatus of this kind is that of Hobson,[1] shown in Fig. 57, in which B represents the entrance of air from the room, A that of the hot-blast, and D the exit of the mixture; C is the mercury thermometer. The apparatus has been improved upon by Krupp.[2] The hot-blast entering at A is throttled down by means of cock H to a constant pressure indicated by the gauge E. In passing through nozzle F it sucks in cold air at B, the temperature of which is measured with thermometer D. The air-mixture streaming through G passes the bulb of the thermometer C which measures its temperature.

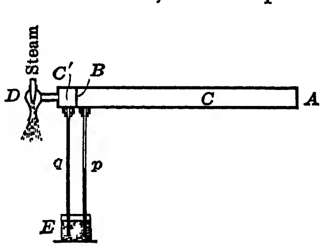


FIG. 59.—The Uehling-Steinbart pyrometer.

The temperature of the hot-blast is found by $T-t=c(t'-t)$ and $T=c(t'-t)+t$; in which T = temperature of hot-blast, c = constant of instrument found by measuring the true temperature of the hot-blast with a reliable type of pyrometer; t' = temperature of air-mixture; t = temperature of cold air sucked in. By using a movable scale and placing its zero at the temperature of the cold air sucked in, the temperature of the hot-blast can be read off directly.

[1] Dingler, *Pol. J.*, 1876, CCXXII, 46.
[2] Bergen, *J. I. and St. I.*, 1886, I, 207.

4. VISCOSITY PYROMETERS.—The viscosity of gases[1] was little used for temperature measurement until the researches of Barus[2] had shown in 1889 that with a gas flowing through a capillary tube or fine orifice the amount entering in a given time under definite pressure depended upon the temperature.

The Uehling and Steinbart pyrometer,[3] shown in Figs. 59 and 60, and extensively used for measuring the temperatures of hot-blast stoves and waste gases with iron blast-furnaces, is based on the viscosity of air flowing through fine orifices. Supposing (Fig. 59) air to be sucked into the chamber C, through the orifice A, and out of it through a second orifice B, into the chamber C', and thence discharged through D, the tension in the two chambers will be a function of the temperature. If the orifice A be connected with the hot-blast pipe and the air cooled down to a fixed temperature before it passes through B, less air will be sucked in at A than is drawn off through B, giving a different tension in chambers C and C', which can be measured by the arms p and q of the manometer. If the suction is a constant and the temperature at B is fixed, any change in the temperature of the hot blast will cause a corresponding change of tension in the chamber to be read off on the manometer. Fig. 60 represents diagrammatically the different parts of the instrument. Into the chamber M, the temperature of which is to be measured, reaches the fire tube f. This consists of a platinum tube d enclosing a smaller one e having the small orifice A. Both tubes are brazed into drawn copper tubes, c and f, surrounded by a water-jacket with in- and out-flows, y and z. The air, sucked in by the aspirator 6, enters through the filter I, a pipe filled with cotton, to be purified, travels through the regulating cock 4 and pipe b into the annular space between f and c and their continuations d and e. Here it becomes superheated and enters at A the pipe e, which forms one end of the chamber C (Fig. 59), and travels through fgh (with branches q' and S leading to the manometer q and the recording gauge L) and the coil i to the opposite end at B. This end is kept at the constant temperature of 100° C. by the aspirator D exhausting into the chamber G, from which steam and water escape through the pipe t at atmospheric pressure. In order to insure constant suction, the aspirator 6 draws

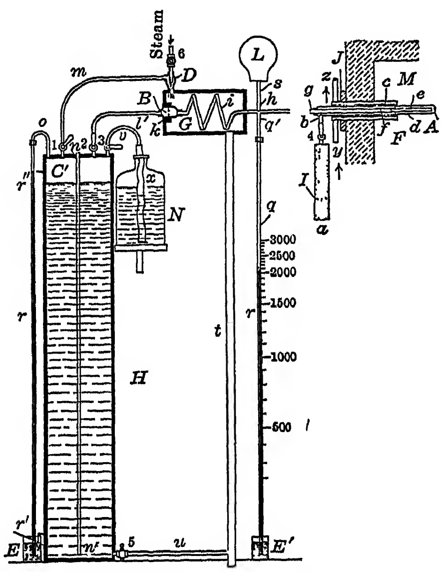


FIG. 60.—The Uehling-Steinbart pyrometer.

[1] Holman, *Proc. Am. Ac. Arts and Sc.*, 1886, XXI, 1.

[2] *Bull.*, U. S. Geol. Survey, No. 54, pp. 46, 239.

[3] *Iron Age*, 1894, LIII, 365; 1895, LV, 434; 1896, LVIII, 814; *Stahl u. Eisen*, 1894, XIV, 389; 1899, XIX, 431; *Eng. Min. J.*, 1895, LIX, 249.

the air through pipe *m*, let into the top of a Mariotte flask *C′* (with tube, *nn′*); into it the air from *i* discharges through cock 2. The suction is regulated by cock 1 so that the water in the manometer *r* shall be at the level of the pointer *r″*. Any evaporated water is replenished from the flask *N* by opening the cock 3.

Figure 61 is one form of an automatic record of the blast temperature of a blast-furnace.[1] Denton and Jacobus[2] tested three pyrometers for temperatures

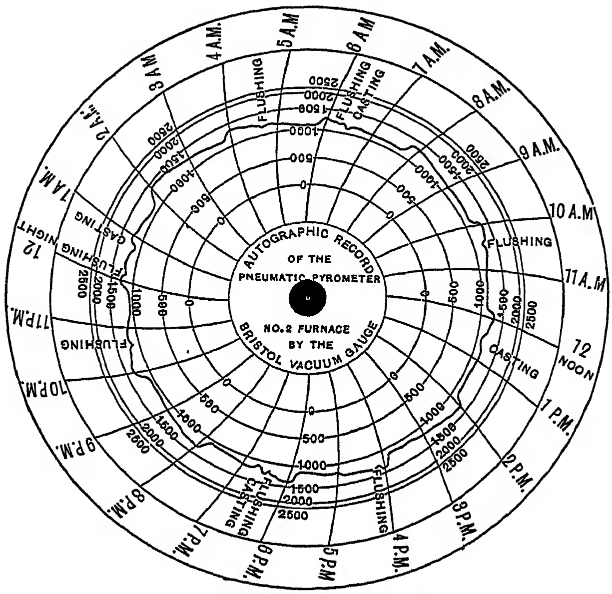


FIG. 61.—Autographic record with Uehling-Steinbart pyrometer of hot-blast of iron blast-furnace.

ranging from 10 to 1,482° C. and found the greatest variation near 1,482° C. to be within 3 per cent. and less for lower temperatures. The readings of a single instrument varied less than 1 per cent.; they were practically the same whether the distance from furnace to reading tube was 12 or 132 ft.

5. OPTICAL PYROMETERS.[3]—Starting from the lowest visible red, temperatures of incandescent bodies can be approximately judged by the eye. Until within a few years the scale given by Pouillet[4] has been the one generally used. He assigned a name to intervals of 100° C. from dull-redness to dazzling white. The scale has been modified by White-Taylor[5] and by Howe,[6] whose notations conform better to the color impressions produced on the eyes of experts by temperatures measured with the LeChatelier thermoelectric pyrom-

[1] *Iron Age*, 1896, LVIII, 814; 1907, LXXIX, 137.

[2] *Eng. Min. J.*, *loc. cit.*

[3] Waidner, C. W., and Burgess, G. K., "Optical Pyrometry," *Bull.* 2, Dept. of Commerce and Labor, Bureau of Standards, Washington, 1905.

[4] *Compt. rend.*, 1836, III, 784.

[5] *Tr. Am. I. Mech. Eng.*, 1900, XXI, 627; *Metallographist*, 1900, III, 41.

[6] *Eng. Min. J.*, 1900, LXIX, 75; *Metallographist*, 1900, III, 43.

TABLE 69.—COLOR SCALES.

White and Taylor		Pouillet		Howe	
Name of color	Deg. C.	Name of color	Deg. C.	Name of color	Deg. C.
....	...	....	....	Lowest visible red in the dark.	470
....	....	Incipient redness.	525	Lowest visible red in daylight.	475
Dark red, blood red, low red.	566	Dark red........	700	Dull red	550–625
Dark cherry red.. .	635	Incipient cherry red.	800		
Cherry, full red.....	746	Cherry red	900	Full cherry......	700
Light cherry, bright cherry, light red.	843	Light cherry red	1,000	Light red........	850
Orange............	899	Dark orange.....	1,100	....	....
Light orange........	941	Light orange....	1,200	....	....
Yellow...........	996	....	....	Full yellow	950–1,000
Light yellow.......	1,079	....	....	Light yellow.....	1,050
White......... ...	1,205	White..........	1,300	White..........	1,150
....	....	Brilliant white...	1,400	....	....
....	....	Dazzling white.	1,500–1,600	....	....

eter. Table 69 by Howe brings together the three color-scales and emphasizes the differences of the Pouillet scale and those of the other two observers which agree comparatively well. Such estimates of temperature cannot correspond to definite degrees; they cover only a certain range of temperature with each observation, as they depend upon individual judgment, and this varies with the susceptibility of the eye, with the time or the brightness of the day, with the degree of illumination of the locality in which the observation is made, and with the radiation of the heated body itself. In certain industries, *e.g.*, in the tempering of steel, individual judgment has been trained to be accurate within 10° C.[1]

A pyrometer which assists the eye in estimating temperatures above visible red is the Mesuré and Nouel Pyrometric Telescope,[2] Figs. 62 and 63. This is a telescope-shaped instrument 5 in. long and 1 in. diam.; Fig. 62 gives the eye-piece and graduated scale, 2 3/4 in. diam.; Fig. 63 is a longitudinal section.

[1] Landis, *Iron Age*, 1896, LVII, 1020.
[2] Howe, *Eng. and Min. J.*, 1890, XLIX, 637; *Berg-Hüttenm. Z.*, 1890, XLIX, 457.
LeChatelier, *Compt. rend.*, 1892, CXIV, 470.
Roberts-Austen, *Tr. A. I. M. E.*, 1893, XXIII, 407.
Thomson, *Iron Age*, 1895, LV, 374.
Struthers, *Sch. Mines Quart.*, 1895–96, XII, 292.

The pyrometer consists essentially of two Nicol prisms; the polarizer *P* and the analyzer *A* are so adjusted that their planes are at 90° to one another when the zero of the graduated scale *C* is opposite the fixed index *I*. Between the two prisms is a quartz plate *Q* cut perpendicularly to its axis. *M* is a removable plate to permit adjusting the prisms or the quartz plate. One end of the telescope is closed by the eye-piece *O*, the other by a parallel diffusion glass *G* slightly ground. In order to exclude outside rays, an extension-tube, 4 1/2 in. long, is attached to *G*. This is especially necessary for observing low temperatures, in which case it is of advantage to add a concentrating lense *L*, focused by rack and pinion, to collect the feeble rays. Supposing the quartz plate to be removed, a ray of monochromatic light on passing through the polarizer *P* will be extinguished if it is watched through the analyzer *A*. By interposing the quartz plate, the light will be intercepted, while still polarized, and will proceed in another plane, *i.e.*, be deflected and become visible. In order to extinguish again,

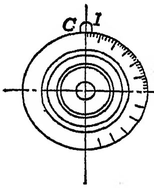


FIG. 62.

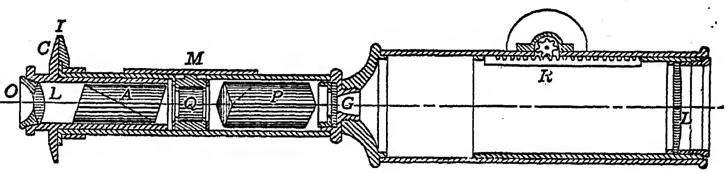


FIG. 63.

FIGS. 62 and 63.—Mesuré-Nouel pyrometric telescope.

one Nicol must be turned a certain angle which is proportional to the thickness of the quartz plate and nearly inversely so to the square of the wave-length. The quartz plate being a constant for the instrument (11 mm.), there remain two variables, the angle of deflection and the wave-length. The former can be measured by rotating the analyzer from the zero-point to the extinction of the light; the latter is proportional to the intensity of the light, and this to the temperature of the incandescent body. The angle of rotation can, therefore, serve as a measure of temperature. The light emanating from an incandescent body, however, is not homogeneous. When it is dull-red, red light is emitted, and as the temperature rises, orange, yellow, green and blue successively appear. In rotating the analyzer, a series of colors is, therefore, produced and not a simple ray extinguished as with a homogeneous light; the rotation to produce a certain color, however, remains proportionate to the intensity of the light and hence to the temperature.

In rotating the analyzer, the color in changing from red to green passes through a lemon-yellow tint which has been selected as the standard for medium-high temperatures. For a white heat the standard tint is violet-gray, for low temperatures the tint disappears, there being an abrupt change from green to red, and at a dark-red there occurs simply the extinction of the red. The practical lower limit of the instrument is 750° C.

TABLE 70.—DATA FOR MESURÉ-NOUEL OPTICAL PYROMETER

Angle of rotation, degrees...	33	40	46	52	57	62	66	69
Temperature, deg. C...... .	800	900	1,000	1,100	1,200	1,300	1,400	1,500

Comparative tests made by Professor C. L. Norton[1] have shown that with a single observation readings are accurate to 25° C. with an average of four readings to 10° C. Thomson[1] speaks well of the instrument for steel works; Roberts-Austen[2] and LeChatelier[3] have found it satisfactory; while Hecht[4] reports the contrary when watching the temperature of a porcelain kiln. Waidner-Burgess[5] place the limits of accuracy at 50 and even 100° C. with temperatures above 1,000° C.

The Cornu-LeChatelier Photometric Pyrometer,[6] Figs. 64 and 65, consists essentially of two telescopes, 13 1/2 and 15 in. long, placed in a horizontal plane

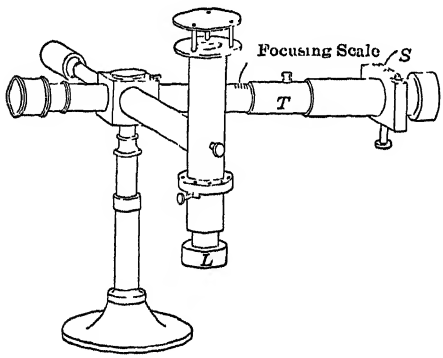


FIG. 64.—Cornu-LeChatelier photometric pyrometer.

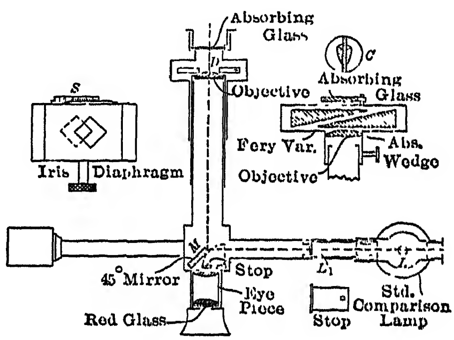


FIG. 65.—Cornu-LeChatelier photometric pyrometer.

and at right angles to one another, supported by a pedestal, 13 in. high. They are so arranged that the light of a standard comparison oil lamp, 10 in. high, reflected by a mirror M, and the light of the incandescent body whose temperature is to be measured after focusing, can be observed through a common eyepiece in front of which is placed a red glass. The intensity of the light of the incandescent body is diminished by varying the opening of the iris diaphragm s in the direct telescope until it matches the intensity of the standard lamp.

[1] Private Communication.
Op. cit.
[2] *Op. cit.*
[3] *Compt. rend.*, 1892, CXIV, 470.
[4] *Thonind. Z.*, 1890, XIV, 575.
[5] *Op. cit.*
[6] LeChatelier, *Compt. rend.*, 1892, CXIV, 214, 391, 470.
Roberts-Austen, *Tr. A. I. M. E.*, 1893, XXIII, 431.
Thwaite, *J. I. and St. I.*, 1892, I, 211.
Waidner-Burgess, *op. cit.*

The adjustment necessary for reducing the two adjacent fields to the same intensity forms the means of measuring the temperature. The arm opposite the one carrying the standard lamp is a counter-weight to give stability to the instrument. This is accurate to 1 per cent., but the difficulties with its use in works lie in the requirement of having an oil lamp which rarely remains uniform, and the lack of portability. In the latest instrument[1] the oil lamp has been replaced by an incandescent electric lamp as with the Wanner pyrometer.

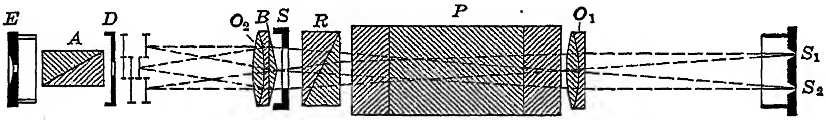


FIG. 66.—Wanner optical pyrometer.

The Wanner Photometric Pyrometer[2] is shown in Figs. 66 and 67. This telescopic instrument, 12 in. long, like that of Cornu-LeChatelier, makes a photometric comparison of the intensity of light emitted by an incandescent body, whose light is to be measured with that of a small incandescent 6-volt lamp fed by a portable storage battery weighing about 15 lb. The light from the incandescent body enters directly through the slit S_2, Fig. 66; that from the incandescent lamp is reflected from the ground surface of a prism into slit S_1. The two beams pass through the objective O and the direct-vision prism P which spreads each out into a spectrum. The two spectra then pass through the Roch prism R which breaks up each beam into two polarized planes at right angles to one another. The slit S cuts out all light excepting a narrow band in the red part of the spectrum which furnishes monochromatic light. The beams of red light pass through the bi-prism and lens O_2 which spreads them out and increases their number from 2 to 8. The bi-prism is so chosen that two of these spectra, polarized in planes at right angles and coming from the hot body and the incandescent lamp respectively, are brought into contact in the field view. All other images are cut out by the diaphragm D. By rotating the Nicol prism A, one field can be strengthened and the other weakened until both have the same intensity.

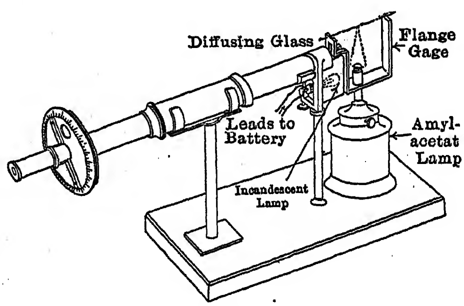


FIG. 67.—Wanner optical pyrometer.

The amount of rotation read on the graduated head, 3 1/2 in. in diam.,

[1] Leeds-Northrup, Philadelphia.

[2] Wanner, *Stahl u. Eisen*, 1902, XXII, 207; *Iron Age*, 1904, Feb. 18, p. 25.
Waidner-Burgess, *op. cit.*
Symposium, *J. I. and St. I.*, 1904, I, 140, 185.
Improved Form "Scimatco," Scientific Materials Co., Pittsburgh, Pa.; also Wanner, *Stahl u. Eisen*, 1911, XXXI, 736.

serves as a measure of the intensity. The intensity of the light from the incandescent lamp will vary with the current of the battery. The lamp must be set to a standard intensity at least once a day. This is done, as shown in Fig. 67, by sighting the pyrometer on a ground-glass diffusing screen illuminated by an amyl-acetate lamp kept at a constant height which is defined by a flame-gauge attached to the lamp. The index on the graduated head is now set to a definite mark (the normal point), and the current through the lamp regulated by rheostat until the two fields are of the same intensity. The loss of light involved by this system makes the lower limit of temperature measurement about 900° C. With the necessary precautions as to strength of current, readings accurate to 1 per cent. are readily made.

6. Radiation Pyrometers.[1]—In this class of apparatus the heat radiated from a furnace or any hot body is made to act upon a measuring instru-

Fig. 68. Fig. 69.

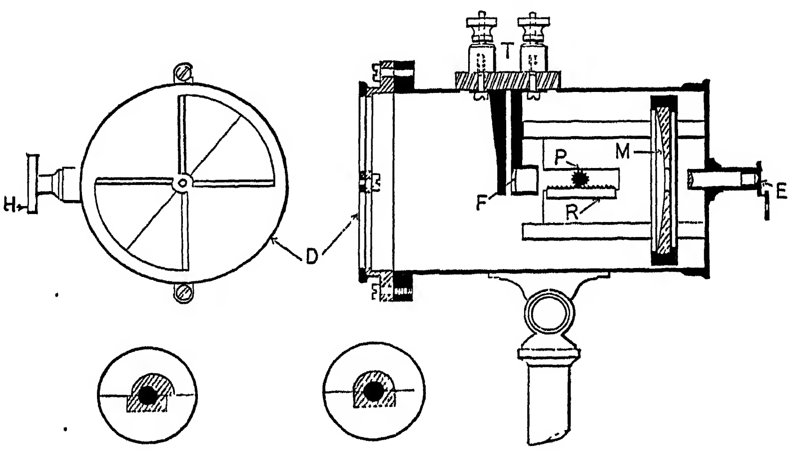


Fig. 70a. Fig. 70b.

Figs. 68 to 70.—Féry radiation mirror pyrometer.

ment, and the change produced noted, which stands in a direct relation to the temperature.

The Féry Radiation Mirror Pyrometer[2] has found favor in some metallurgical plants, as the temperatures are accurate and are read quickly, and the apparatus is not inconvenient to handle. Fig. 68 gives an end view showing a hinged sectoral diaphragm with shutters for reducing the opening of the telescope, Fig. 69 a longitudinal section, and Fig. 70*a* and *b* two focusing diagrams. The essential parts are, *M*, a gold-coated glass mirror with opening in center; *E*, the eye-piece; *H*, the head of a pinion-rod attached to pinion *P* engaging in rack *R* for moving to and fro the mirror to focus the incandescent object; *F*, the

[1] Shook, *Met. Chem. Eng.*, 1912, x, 238, 334, 416, 478, 534.

[2] Wilson-Maeulen, *School Mines Quart.*, 1907, xxviii, 353; Foster, *Met. Chem. Eng.*, 1909 vii, 39; Taylor Instrument Co., Rochester, N. Y.

focus with copper-constantan thermocouple surrounded by two semicircular mirrors set at a slight angle to one another, so that the observer looking through eye-piece E sees the black image of the thermocouple split into an upper and lower half; T, two insulated binding posts connected with brass strips that are attached to the wires of the thermocouple and conduct the current through leads to a portable sensitive millivoltmeter.

Heat rays striking the mirror are reflected toward F, and then focused upon F by moving the mirror with rack and pinion. This is accomplished when the two halves of the black spot seen through the eye-piece in the center of the field match or conform as in Fig. 70*b*; when the head H is turned and the focus destroyed, one-half of the black spot moves to the right, the other to the left as shown in Fig. 70*a*. The focused rays warming the thermocouple cause this to send out a current, the e.m.f. of which is measured. The temperature-reading is practically independent of the distance of the mirror from the hot body, as the greater the distance the smaller becomes the image, and as only a small part of the image comes into consideration; all that is necessary is that the hot image be large enough to more than cover the thermocouple. In practice the telescope may be about 1 yard distant for every inch in diameter of the hot body. For temperatures ranging from 500 to 1,100° C., the diaphragm, Fig. 68, is unhooked and allowed to hang down out of the way so as to admit all the rays into the telescope; for temperatures higher than 1,000 to 1,100° C., it is swung into place, and the shutter opened to a fixed stop so as to admit a certain portion of the rays; when not in use, the shutter is closed. The portable millivoltmeter has three rows of gradations, one in red to check electrically the accuracy of the instrument, and two in black denoting degrees centigrade scale; one of these ranging from 500 to 1,100° C. is to be read when the diaphragm is down and all the rays enter the tube, the other ranging from 1,000 to 2,000° C., when the diaphragm is up and the shutter opened to the stop. The instrument is calibrated by the maker against fixed temperatures.

7. Acoustic Pyrometers.—The Wiborgh Thermophone is the only representative. It is a small cylinder, 1 in. long by 0.75 in. diam., of refractory material containing a small amount of explosive substance enclosed in a metal casing. When a cylinder is placed in a furnace or chamber, the temperature of which is to be measured, it will be brought in a certain time to the firing-point and explode with a sharp detonation. The time which elapses from the moment the cylinder is placed in the furnace until it explodes is noted to 1/5 of a second. The corresponding temperature is given in a table supplied by the makers of the cylinders. Temperatures from 300 to 2,200° C. can be approximately determined with the thermophone; they agree to 1/5 second or to 20° in 1,000° C. It is, of course, absolutely necessary that the cylinders be protected from moisture, and be kept at an initial temperature of say 20° C.

8. Electric Pyrometers.—There are two types of these, the electric resistance and the thermoelectric.

(*a*) *Electric-resistance Pyrometers.*—Measurement is based upon the increase of resistance a wire, usually Pt, offers to the passage of a current with the rise

of temperature. The apparatus contains three parts, a battery to produce the current, a coil the resistance of which to the current is to be measured, and a bridge for measuring the resistance. Two pyrometers may serve as examples, the W. Siemens and the Callender.

The Siemens pyrometer[1] is represented diagrammatically in Fig. 71. A current passing out from battery *B* is divided; one branch goes through the platinum wire *C* coiled around a clay cylinder, the other to the resistance coil *R*. At ordinary temperature the resistances of the two wires are balanced. When the clay cylinder is introduced into the heated space, the temperature of which is to be measured, the resistance of its wire increases and causes a deflection in the galvanometer *G*, which indicates the temperature directly if it has been standardized against known temperatures. The main defect of the original form was that its zero-point changed. In the latest form[2] this has been corrected by the use of Callender's compensating leads.

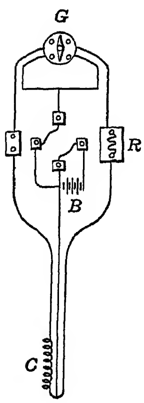


FIG. 71.—Siemens electric resistance pyrometer.

THE CALLENDER[3] PYROMETER.—Callender found that the change of zero in the Siemens instrument was not inherent in the method, but was due to strains in and to contaminations of the wires, and proved that the method would give most accurate results if these were removed. His pyrometer is represented diagrammatically in Fig. 72: The pyrometric coil *P* is connected with the lead wires *AM* and *KN*. Wires *CL* and *DL* are the "compensating wires"; they have the same resistances as *AM* and *KN*, and are so placed that corresponding parts may always be at the same temperature. Wires *AB* and *BC*, which have equal resistances, and *DE*, a set of resistance coils, are brought together in a box; *FK* is a straight-wire bridge with scale, *H* the sliding contact-piece, and *G* the galvanometer; the

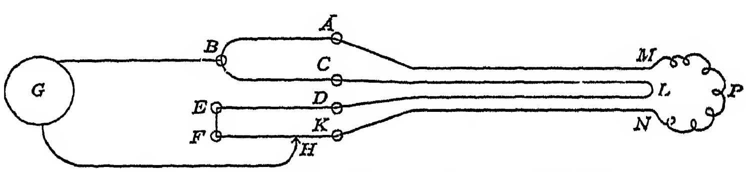


FIG. 72.—Callendar electric resistance pyrometer.

battery is connected at *A* and *C*. Supposing the balance to have been found, with the galvanometer pointing to zero, by inserting suitable resistances in *DE*

[1] Siemens, *Proc. Roy. Soc.*, 1871, XIX, 351; *Trans. Soc. Telegr. Eng.*, 1874, III, 297.
Callender, *Phil. Trans. Roy. Soc.*, 1887, CLXXVIII, 161; *J. I. and St. I.*, 1892, I, 164.
Roberts-Austen, *Tr. A. I. M. E.*, 1893, XXIII, 415.

[2] *J. I. and St. I.*, 1904, I, 118.

[3] Callender, *Phil. Trans. Roy. Soc.*, 1888, CLXXVIII, A, 160; *J. I. and St. I.*, 1892, I, 169.
Callender-Griffith, *Phil. Trans. Roy. Soc.*, 1891, 182, A, CLXXXI, 43.
Roberts-Austen, *Tr. A. I. M. E.*, 1893, XXIII, 417.
Report, *J. I. and St. I.*, 1904, I, 103.

and shifting the contact-piece *H*: the resistance *AM*, *P*, *NK* will be equal to *HF*, *FE*, *ED*, *DL* and *LC*. As any change in the leads *AM* and *KN* is compensated by that in *CL* and *LD*, all changes in *P* will be measured by the coils in *DE* that have been thrown into circuit, plus the reading on *FK*. One of the industrial instruments,[1] made by the Cambridge Scientific Instrument Co., Cambridge, England, is shown in Fig. 73. Two heavy platinum wires, the out and return wires or true leads, and the compensating wires, or blind leads, pass from the head through an externally glazed porcelain tube; they are held apart by mica discs and are wound on a serrated frame of mica. The terminals are connected with a battery and a self-reading resistance-box contained in a portable box not shown.

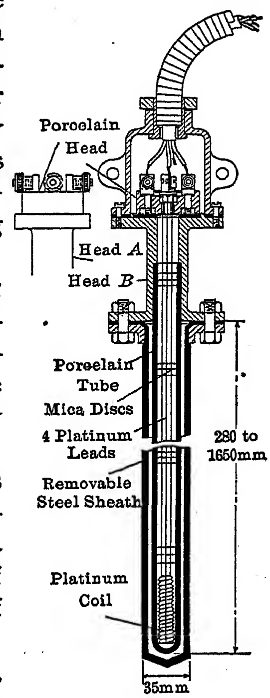


FIG. 73.—Industrial form of Callendar electric resistance pyrometer.

Resistance pyrometers are very accurate, within 0.05 per cent. up to 1,000° C. As instruments for metallurgical purposes they have the disadvantage of being complicated and expensive, and at the same time fragile and easily damaged. They are, therefore, not generally used at smelteries.

(*b*) *Thermo-electric Pyrometers.*—Measurement is based upon the fact that by heating the thermo-junction of two dissimilar metals, a current is produced in a closed circuit which, according to Barus,[2] is nearly, if not exactly, proportional to the absolute temperature of the junction.

The pyrometer of H. LeChatelier,[3] invented in 1886, has become the leading instrument for the accurate measurement of temperatures ranging from 300 to 1,600° C. It consists, Fig. 74, of a thermo-electric couple *J*, two connecting leads *L*, cold junction *C*, and a galvanometer *V*. The couple is composed of two No. 24 wires (0.022 in. thick); one is Pt, the other a homogeneous alloy of Pt with 10 per cent. Rd or Ir. The wires are thin in order that the couple may almost instantaneously assume the temperature of the heated space. The junction (called "hot junction" as it is exposed to the heat to be measured) is made by twisting together the ends, or by soldering them with Au, Pd or Pt. The wires are isolated from one another by passing them through small refractory clay tubes (0.24 in. diam., 11.8 in. long) with a double bore, Fig. 76 (or fused silica tubes), or by covering them with asbestos string which is wound over and under in the form of the figure 8, each wire passing through one of the loops. Asbestos

[1] *J. I. and St. I.*, 1892, I, 176; 1904, I, 104.

[2] U. S. Geol. Survey, *Bull.* No. 54, p. 52.

[3] *Compt. rend.*, 1892, CXIV, 470; *Tr. A. I. M. E.*, 1893, XXIII, 418; *Eng. Min. J.*, 1892, LIII, 399; *J. I. and St. I.*, 1891, I, 90; 1892, II, 33; 1893, I, 112; 1904, I, 106; *School Mines Quart.*, 1892, XII, 143; *Stahl u. Eisen*, 1892, XII, 894; *Berg. Hüttenm. Z.*, 1892, LI, 277, 301, 316.

melting at 1,200 to 1,300° C. can be used only with temperatures lower than 1,200° C. The insulated wires are enclosed in an iron tube, 0.5 in. inner diam., so as to permit introducing the junction into the heated space. The arrangement represented in Figs. 75 and 76 is intended for measuring the temperature in an open-hearth furnace. It shows the iron tube, 10 ft. long, with double-bore clay tubes and wires, clamped to a stick, 3 ft. 3 in. long. Two isolated copper leads from the galvanometer, held in one countering, are wound

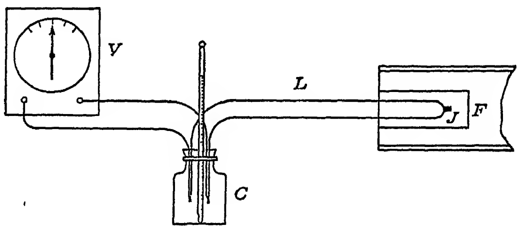
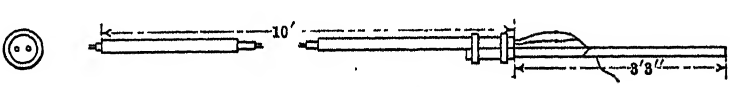


FIG. 74.—LeChatelier thermo-electric pyrometer. (Norton.)

around the stick and their protruding ends are connected with the Pt and Pt–Rd wires. With a shorter iron tube, say 6 ft. long, a wooden handle is secured to the tube, as shown in the Stupakoff[1] arrangement, Fig. 77. The handle carries the binding posts for the leads from the galvanometer; the posts form small spools for extra lengths of Pt and Pt–Rd wire to permit replacing parts of the couple that break or burn off. In taking the temperatures of fused metals there is danger of the junction alloying; or, in making a prolonged observation in a reducing atmosphere, the wires may be affected. It can be protected by covering it with a paste made of quartz with 10 per cent. clay and some silicate of soda, or by enclosing it in silica-tubing or in a fire-clay tip as shown in



FIGS. 75 and 76.—Protection cylinder with double-bore clay tubes

Fig. 77. The angular connection in this figure can be replaced by one forming a straight continuation of the rod. It takes about 1.75 min. exposure for the protected couple to assume the temperature of the surrounding metal; a naked couple requires not over 5 sec. The movable hand-screen in Fig. 77 protects from the heat the hand of the observer.

The connecting leads, attached to the Pt and Pt–Rd wires by twisting or soldering, are isolated (rubber-covered) electric copper wires not smaller than No. 14 gauge (0.083 in.). The resistance of 328 ft. No. 20 (0.035 in.) copper

[1] *Foundry*, 1898, XIII, 100.

wire is only 2.25 ohms, and insignificant figure in comparison with that of the galvanometer, which is at least 200 ohms. The leads can, therefore, be made very long and the galvanometer set up at a considerable distance from the furnace.

In accurate work, it is necessary to know the temperature of the junctions of the thermoelectric couple with the copper leads, the so-called "cold junction." For this purpose they are enclosed in a small stoppered bottle *C* with thermometer as shown in Fig. 74. The temperature of the bottle may be prevented from fluctuating by wrapping it in some insulating material such as felt, magnesia or asbestos. The simplest way of keeping the temperature of the cold junction constant is to keep it at 0° C. in melting ice.

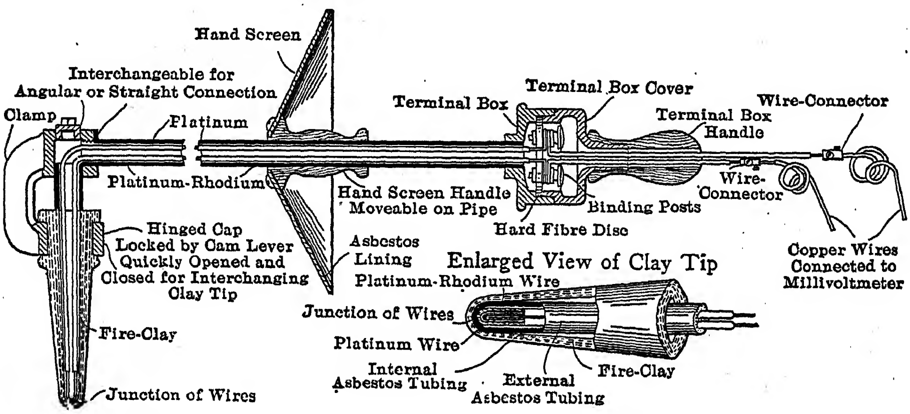


FIG. 77.—LeChatelier-Stupakoff thermo-electric pyrometer.

The galvanometer for measuring the currents formerly in common use is the Deprez-d'Arsonval type which is accurate to 0.01 per cent. In it the position of a coil of wire suspended in a magnetic field is changed with the strength of the current passing through it. The change is indicated by observing the path of a beam of light reflected from a mirror attached to the coil (Carpentier), or the deflection of a pointer attached to it (Keiser and Schmidt). The Carpentier apparatus, Figs. 78 to 83, consists of two wooden frames *A* and *B* fastened back to back to a center board *C* with handle, thus making the whole compact and portable. The box *A* is suspended from a hook in a place free from vibrations, the swinging door opened and the apparatus then adjusted by means of set-screws *v v* and the small plumb-bob. The box-shaped cover *D* is now taken off; this leaves *B*, as shown in Fig. 81, to be suspended and leveled. The distance between *A* and *B* is 1 m. (3.3 ft.). The galvanometer consists of the permanent horse-shoe magnet, *AB*, Fig. 83, between the poles of which a soft-iron core *E* is supported by a standard (not shown). The core is surrounded by a torsion balance *E*, a rectangular frame, Figs. 82–83, of fine German-silver wire of at least 200 ohms resistance (400–500 ohms is better). It is held in position by two vertical wires with balls at the ends, all of German-silver. The

upper ball is suspended from a hard-rolled copper plate *l*, bent to the form of a spring which can be tightened or loosened with the thumb-screw *V*. The frame

FIG. 78. FIG. 79. FIG. 80. FIG. 81. FIG. 82.

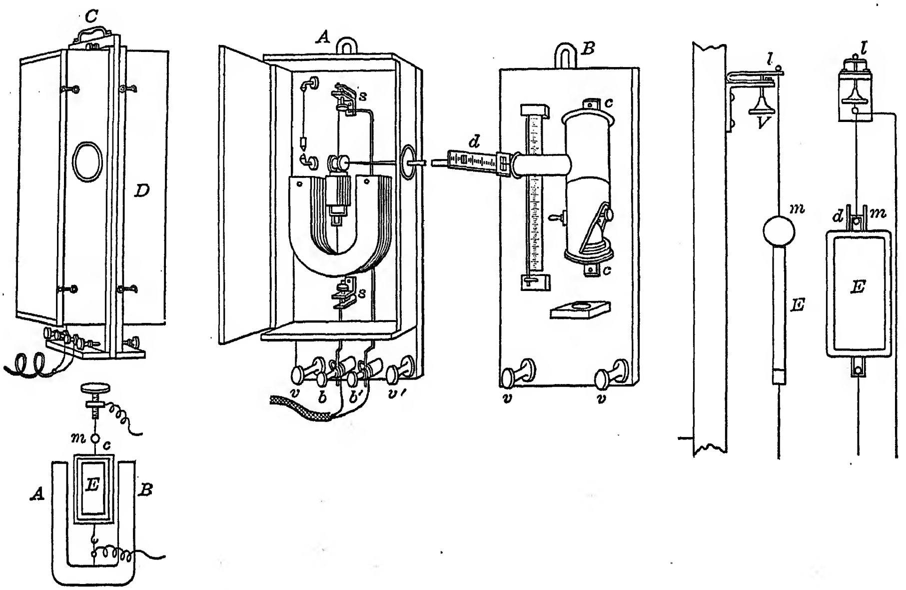


FIG. 83.

FIGS. 78 to 83.—LeChatelier pyrometer with Duprez-d'Arsonval galvanometer and Carpentier arrangement.

carries a small plano-convex mirror *m* balanced by disc *d;* the position of the mirror may be adjusted from the lower suspension ball; *b* and *b'* are the + and − poles. Box *B* holds the lamp inclosed in a sheet-iron cylinder *c* and the graduated transparent sliding scale *d*. Benzine or naphtha are burnt in the lamp; a small gas-flame or, better still, a small incandescent electric lamp may serve as a source of light. On one side of the chimney is a small tube with lens to collect the rays of light and converge them on the mirror. The scale-frame in front of the tube has a small window with a vertical wire. The image of the bright window and its dark wire are reflected by the mirror *m* onto the transparent scale *d* and make an exact reading easy, especially if, as is usual, the place is darkened by a curtain. When the image appears on the scale, this is placed at zero either by moving the adjustable scale in its frame or by turning the galvanometer with the set-screws or by revolving one of the suspension-wires, especially the lower one.

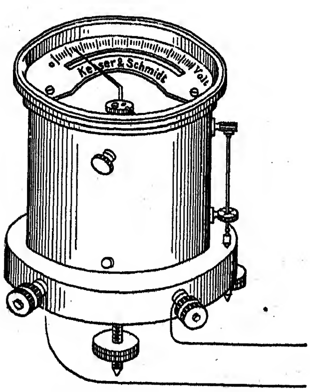


FIG. 84.—Keiser-Schmidt galvanometer.

In the Keiser and Schmidt galvanometer, 6 in. diam. × 3.5 in. high, Fig. 84 the coil has to be much more sensitive, as the pointer has to travel over a much larger angle than in the Carpentier instrument. This is attained by having only the upper suspension, and by using a very fine wire. The apparatus has to be handled with care. When it is to be used, it is leveled and the screw in the side loosened to remove the suspension of the coil. When a reading has been made, the side-screw is tightened up again.[1]

The apparatus having been set up is ready to be calibrated. This is done by exposing the hot junction to melting and boiling substances with known

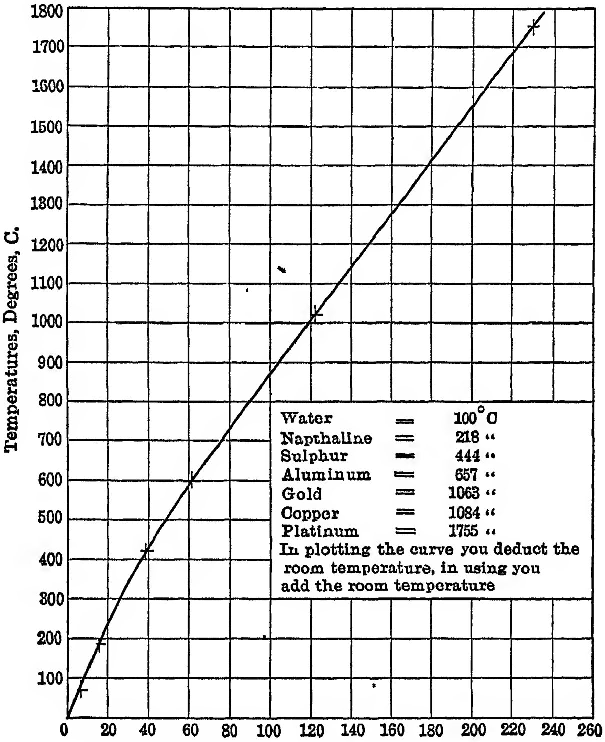


FIG. 85.—Calibration of LeChatelier pyrometer.

melting- and boiling-points. By plotting the temperatures as ordinates and the deflections as abscissæ (Fig. 85), a flat parabolic curve is obtained from which all the intervening temperatures may be read off. Some such fixed points are: boiling H_2O, 100°; boiling naphthalene, $C_{10}H_8$, 218°; boiling S, 444°; melting Al, 657°; Au, 1,063°; Cu, 1,084°; Pd, 1,550°; Pt, 1,755°. Exposing the couple to boiling water in a beaker or to melting naphthalene in a test-tube is simple; with *S* boiling in a test-tube care must be taken not to plunge the couple into

[1] In recent years galvanometers by Siemens-Halske, Berlin-New York; the Cambridge Instrument Co., Cambridge, England; the Taylor Instrument Co., Rochester, N. Y.; the Leeds and Northrup Co., Philadelphia, Pa., and probably some other makes have come into more general use than the two galvanometers described.

the fluid overheated *S*, but to keep it in the vapor. A simple way of exposing the couple to a fused metal is to wind a small strip of it around the couple, surround it with a little asbestos, and heat carefully with a Bunsen burner, or for higher temperatures with an oxygen-illuminating or an oxy-hydrogen gas blowpipe. The picture of the transparent scale will move on slowly (Fig. 85), come to a stop while the metal is melting and then rise again when the molten metal is being superheated. Heating much above the melting-point of a metal is to be avoided, as there is danger of the couples becoming alloyed. When the flame is removed, the metal will cool down slowly and the receding picture again comes to a stop, while the metal solidifies. Thus the melting-point is verified by the freezing-point.

With a larger mass of metal, some kind of crucible furnace will be needed. In either case the heating should be performed slowly. With water, naphthalene and sulphur, the image, of course, comes to a stop when the boiling-point has been reached. In plotting the curve, the room temperature (or temperature of the cold junction) has to be deducted from the observed fixed points, and in using the curve later on, the room- or cold-junction temperature has to be added. Holman[1] devised the logarithmic plot which requires only two points (S and Cu for low, Cu and Pt for high temperatures) which are then connected by a straight line.

The advantages of the thermo-electric pyrometer are, that it is accurate, that the observed temperature depends solely upon the difference in temperature between the hot and cold junctions, that it is adapted for measuring temperatures of small spaces, that it indicates the temperatures quickly and is thus suited for measuring varying temperatures (cooling curves), that it is durable as long as it is protected from Si and metal, that it can be readily checked at any time.

In order to permit noting automatically all thermal changes in metals, alloys or metallic compounds, continuously recording devices have been constructed. The record is made by means of a pen (as with the Uehling-Steinbart record, Fig. 61) or photographically. The leading apparatus are those of Roberts-Austen,[2] Saladin,[3] Callender,[4] Siemens and Halske,[5] Kurnakoff,[6] Wolgodine,[7] Northrup,[8] Burgess[9] and Brown[10].

98. Fuels Proper. Introduction.—Fuels occur in nature in the solid, liquid, and gaseous state. They are used in the arts either in the natural state, as

[1] *Proc. Am. Acad. Art. Sc.*, 1895–96, XXXI, 218, 234.

[2] *Tr. A. I. M. E.*, 1893, XXIII, 420.

[3] *Iron and Steel Metallurgist and Metallographist*, 1904, VII, 237.

[4] *Phil. Mag.*, 1910, XIX, 538.

[5] *Zt. Instrumentenkunde*, 1904, XXIV, 350; 1905, XXV, 273.

[6] *Zt. Anorg. Chem.*, 1904, XLII, 184.

[7] *Rev. Mét.*, 1907, IV, 552.

[8] *Tr. Am. Electrochem. Soc.*, 1909, XV, 331.

[9] *Bureau of Standards*, Washington, 1908, *Bull.* No. 5, "Methods of Obtaining Cooling Curves," p. 199.

[10] *Electrochem. Met. Ind.*, 1909, VII, 329.

they come from the mine or well, or they are first subjected to some preliminary treatment. Fuels are conveniently classed under the following heads:

SOLID FUELS:

Natural Fuels: Wood, peat, brown coal (lignite), bituminous coal, anthracite.

Prepared Fuels: Pulverized fuel, compressed fuel, carbonized fuel (charcoal, peat-coke, lignite-coke, coke).

LIQUID FUELS:

Natural Fuels: Petroleum.

Prepared Fuels: Distilled oil.

GASEOUS FUELS:

Natural Fuels: Natural gas.

Prepared Fuels: Iron blast-furnace gas, producer gas, water gas, coal gas, oil gas.

99. Natural Solid Fuels in General.—All natural solid fuels, with the exception of perhaps graphite, are of vegetable origin. The younger fuels, wood and peat, are being formed at present; the older ones, brown coal, bituminous coal and anthracite were formed in preceding geological periods and have undergone many changes since they were buried in the earth. Thus the color has changed from white to yellow, brown and black; the specific gravity has been nearly doubled; the amount of H_2O has greatly decreased as has the percentage of O with a corresponding increase of the percentage of C. Table 71 of Grüner[1] for solid fuels free from water and ash brings out clearly the changes vegetable matter has undergone by mineralization. Anthracite and the values for disposable H and the calorific power have been added to the original table.

100. Wood.[2]—Wood, which for many ages was the sole fuel of man, has become of minor importance to the metallurgist since the advent of coal. It holds its own, however, as a metallurgical fuel in districts in which coal or other mineralized fuel is expensive. Trees are generally classed as deciduous (broad-leaved) and evergreen (coniferous). Their heat value depends upon the proportion of wood-substance, and this is indicated by the apparent specific gravity, *i.e.*, the relationship between a volume of wood (including the cell-spaces) and an equal volume of water. Wood with an apparent specific gravity of 0.55 and over when dry is often called hard wood, and soft wood if the apparent specific gravity is below 0.55. The true or actual specific gravity of wood-substance is about 1.5. A hard-and-fast line cannot be thus drawn between hard and soft wood, as the apparent specific gravity of a species is not a fixed number, but covers a certain range.

In Table 72 by Roth,[3] giving the weights of some kiln-dried woods, are inscribed the three divisions, hard, medium and soft woods. Best southern pine is enumerated under hard wood, ordinary pine under medium; heavier spruce and fir are included under medium light, ordinary spruce and fir under soft.

[1] "Traité de Métallurgie," Dunod, Paris, 1875, I, 67.

[2] F. Roth, "Timber," *Bull.* No. 10, U. S. Dept. of Agriculture, Division of Forestry, Washington, 1905.

[3] *Op. cit.*, p. 28; see also *Report*, Tenth Census (1880) U. S., Vol. IX, pp. 248, 266.

TABLE 71.—GRÜNER'S CLASSIFICATION OF SOLID FUELS

Name of fuel	Chemical composition, per cent.			Ratio, O[1] : H	Disposable H	Calorific power, cal.	Fixed carbon	
	C	H	O[1]				Per cent.	Nature
Cellulose, $C_6H_{10}O_5$	44.44	6.17	49.39	8	none	3,600	28–30	Pulverulent, dull.
Wood	50–51.75	6–6.30	44–41.95	7	0.5–1.0	4,000	30–35	Pulverulent, dull.
Peat	58–63	6–5.5	36–31.5	6–5	2.0	4,500	35–40	Pulverulent, dull.
Brown coal, lignite	65–75	6–4	29–21	5	1.5–2.5	5,000–8,000	40–50	Pulverulent, sometimes lustrous.
Bituminous coal: Dry, long-flame, non-coking (splint coal).	75–80	5.5–4.5	19.5–15	4–3	2.5–3.1	8,000–8,500	50–60	Pulverulent or slightly coherent.
Bituminous coal: Fat, long-flame (gas-coal)	80–85	5.8–5	14.2–10	3–2	3.7–4.0	8,500–8,800	60–68	Caked, but very friable.
Bituminous coal: Fat, properly so-called (steam, smith's coal).	84–89	5–5.5	11–5.5	2–1	3.6–4.8	8,800–9,300	68–74	Caked, moderately compact.
Bituminous coal: Fat, long-flame (coking coal)	88–91	5.5–4.5	6.5–4.5	1	3.9– .7	9,300–9,600	74–82	Caked, very compact, little friable.
Bituminous coal: Lean, anthracitic coal	90–93	4.5–4	5.5–3	1	3.7	9,200–9,500	82–90	Fritted or pulverulent.
Anthracite	93–95	4–2	3–3	1–0.75	2.5	9,000–9,200	90–92	Pulverulent, lustrous.

[1] Includes N, which rarely exceeds 1 per cent.

TABLE 72.—ROTH: WEIGHTS OF DIFFERENT SPECIES OF KILN-DRIED WOODS

Character of wood		Common name of species	Apparent sp. gr.	Weight of 1 cu. ft. lb.
Hard	Very heavy	Hickory, oak, persimmon, osage orange, black locust, hackberry, blue beech, best of elm, ash.	0.70–0.80	42–48
Hard	Heavy	Ash, elm, cherry, birch, maple, beech, walnut, sour gum, coffee-tree, honey-locust, best of southern pine, tamarack.	0.60–0.70	36–42
Hard	Medium	Southern pine, pitch-pine, tamarack, Douglas spruce, western hemlock, sweet gum, soft maple, sycamore, sassafras, mulberry, light grades of birch and cherry.	0.50–0.60	30–36
Medium	Light	Norway and bull pine, red cedar, cypress, hemlock, heavier spruce and fir, redwood, basswood, chestnut, butternut, tulip, catalpa, buckeye, heavier grades of poplar.	0.40–50	24–30
Soft	Very light	White pine, spruce, fir, white cedar, poplar......	0.30–40	18–24

Wood is sold by the cord: $4 \times 4 \times 8$ ft. = 128 cu. ft. The cord has[1] about 56 per cent. solid wood and 44 per cent. interstitial spaces. According to Fernow[2] 1 cord fire wood (>6 in. diam.) contains 69.44 per cent. solid wood or 75 cu. ft.; 1 cord billet wood (3–6 in. diam.) 55.55 per cent. or 60 cu. ft.; for brushwood (<3 in. diam.) the figure is 18.52 per cent. and for roots, 37 per cent. One cord of air-dried hickory or hard maple weighs about 4,500 lb.; white oak, 3,850; beech, red or black oak, 3,250; poplar, chestnut, elm, 2,350; average pine, 2,000 lb.

The ultimate composition of wood is fairly uniform for different species (see Table 71); the rational analysis shows that it consists of cellulose with its incrusting layer of lignin, of sap, water, and a small amount of inorganic matter forming the ash. Cellulose (analysis, see Table 71), the wood-substance or wood-fiber is the main heat-giving constituent of wood; it forms 95 to 96 per cent. of wood freed from H_2O. Lignin and sap are composed mainly of carbohydrates and water which hold in solution inorganic salts. Their amounts vary with the season of the year and form about 3 per cent. of wood free from H_2O. The percentage of water varies with the species, the age and parts of a tree, the soil,

[1] Birkinbine, *J. Charcoal Iron. Work.*, 1881, II, 293.

[2] *Op. cit.*, 1882, III, 20.

the climate and the season of year. Table 73 by Roth[1] gives approximately the percentage of water lost by different species when kiln-dried.

TABLE 73.—LOSS OF WATER IN KILN-DRYING WOOD TO 70–80° C.

Name of species	Per cent. water lost in kiln-drying	
	Sapwood or outer part	Heartwood or inner part
Pine, cedar, spruce and fir	45–65	16–25
Cypress, extremely variable	50–65	18–60
Poplar, cottonwood, basswood	60–65	40–60
Oak, beech, ash, elm, maple, birch, hickory, chestnut, walnut, and sycamore.	40–50	30–40

Wood newly felled contains 30–50 per cent. (average, 40 per cent.) H_2O. Trees when young contain more H_2O (up to 60 per cent.) than when old; shoots and twigs are richer in H_2O than is the trunk; a rich soil and a mild climate increase the percentage of H_2O; in spring and summer wood contains more water than in winter (the best time for cutting). Green wood after it has been exposed for a couple of years to the drying influences of air becomes "air-dried," but still retains 20 to 25 per cent. H_2O. It is usually cut in 4-ft. lengths, split and stacked. If too small to be split, or if the bark is to be utilized, the wood is sometimes cut when the sap begins to rise, and the bark removed. Wood should be protected from rain and wet; if floated, it should not remain in the water any longer than necessary, as it becomes water-logged and loses heating value. If the H_2O is to be reduced to below 20 to 25 per cent. this can be done only by artificial heating (kiln-drying). At 125° C. wood loses all its hygroscopic H_2O, at 150° C. decomposition begins. Kiln-drying has ceased to be of any importance in metallurgical works.

Wood ash[2] is usually white; with air-dried wood it averages 1 per cent. The smallest amount is found in the stem (0.15 per cent.), the largest in the branches, leaves and roots (2 per cent.). The analyses given in Table 73*a* are by Åkerman and Särnström.[3] Wood-ashes consist mainly of alkali and alkali-earth carbonates. The same investigators found that broad-leaved trees contained from 4 to 5 times as much P as conifers; that conifers cut in winter contained more P than when cut in summer, and that the bark and branches of a tree were richer in P than the trunk.

[1] *Op. cit.*, p. 31.

[2] *Report*, Tenth Census (1880) U. S., Vol. IX, pp. 248, 266.

[3] Through Stohmann-Kerl, "Handbuch der Technischen Chemie," Brunswick, 1893, IV, p. 361; see also Lundström, *J. I. and St. I.*, 1888, I, 260.

TABLE 73a.—ANALYSIS OF WOOD-ASHES

Species	Ash, per cent. in wood with 20 per cent. H_2O	100 parts ash contain									
		Alkali	CaO	MgO	MnO	F_2O_3	Al_2O_3	SiO_2	SO_3	P_2O_5	CO_2
Scotch fir, heartwood......	0.138	13.04	39.25	10 80	5.36	3.04	....	1.80	....	0 36	21 87
Scotch fir, sapwood........	0.182	16.88	34 62	6.78	4 77	2.76	2.65	2.40	4.96	3.87	19 50
Scotch fir, bark...........	1.157	9.47	49 80	6 75	2 84	1 03	10 70	1 85	2.83	5.95	7.50
Norway spruce, heartwood.	0.242	13.88	39 60	7 52	3.51	1.64	1 35	2.20	3.20	0.39	25 30
Norway spruce, sapwood...	0.204	18.03	37.25	5.45	3 22	1 18	....	2 75	......	3 42	21.76
Norway spruce, bark......	4.471	5.40	47.70	3.38	3 51	0 59	3 25	1.35	0 83	2.21	30 70
Birch....................	0.219	18 59	30 75	10.33	3.51	0.85	3.40	1.96	3.90	5.12	21 36

Air-dried wood begins to burn at 280° C.;[1] soft wood is more readily kindled than hard, and burns more quickly and with a longer flame. The calorific power[2] of wood free from H_2O was given in Table 71 as 4,000 cal., with wood containing 20 per cent. H_2O this figure is reduced to 3,000 to 3,200 cal. In burning, a yield of 80 per cent. or 2,400 to 2,500 cal. can be counted upon. According to Brix,[3] 1 lb. of wood with 16.2 per cent. H_2O will evaporate 3.67 lb. H_2O. For perfect combustion 1 kg. dry wood requires 4.57 cbm., or 1 lb., 73.36 cu. ft. air. The calorific intensity of dry wood with a calorific power of 3,600 cal. is 2,500° C., that of air-dry wood with a calorific power of 2,750 cal. is 1,900° C.

101. Peat.[4]—Peat is an accumulation of more or less decomposed vegetable matter in swampy places. Peat bogs are forming at present; none are older than the Quaternary. The plants from which peat was formed are mainly mosses, grasses, ferns, rushes, reeds, heath, leaves, etc. In order that this vegetable matter may be decomposed by fermentation it must be submerged in water. Peat bogs are formed mainly in the temperate zone under special conditions of moisture and heat on a soil impervious to water. The process of decay may be expressed by $6C_6H_{10}O_5 = 7CO_2 + 3CH_4 + 14H_2O + C_{26}H_{20}O_2$.

[1] Börnstein, *J. Gasbel. und Wasserversorgung*, 1906, XLIX, 648; see also Table 59.

[2] See also *Report*, 10th Census (1880) U. S., Vol. IX, p. 251; Poole, "Calorific Power of Fuels," New York, 1900, p. 246.

[3] *Bull. Soc. Ind. Min.*, 1855–56, I, 48.

[4] Johnson, S. W., "Peat and Its Uses," Orange, Judd Co., New York, 1866.

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Journal of the American Peat Society, Toledo, O., Vol. I, 1908–09, and following.

The older and thicker the deposit, the further will decaying have progressed; the denser and harder will be the peat, and the higher its calorific value. On the top of a bog will be found the green sod, then follows the middle layer composed of tangled, light-brown roots and remains of plants partly decomposed; in the lower layer the peat appears as a fine-grained, compact, deep brownish-black mass free from fiber which often shows a pitchy luster. As the waters leading to a bog carry with them sands and slimes which settle, peat will always be comparatively rich in ash. Peat bogs are 4 to 12 ft. thick; occasionally they reach a depth of 50 ft. In North America[1] they are found mainly in the north-eastern portion. Their southern limit is along a line drawn through New Jersey, Pennsylvania, Ohio, Indiana and Delaware; their western limit is in central Wisconsin and eastern Minnesota. In Europe[2] bogs are worked extensively in Great Britain, Scandinavia,[3] Germany,[4] Holland, Russia and Austria. Peat when dug is wet, containing as much as 80 per cent. H_2O; when air-dry it retains 15 to 20 per cent. This figure may be reduced by artificial drying at 100 to 120° C., but being very hygroscopic, peat quickly again absorbs H_2O. At 120° C. decomposition begins. Hausding[5] gives 0.213–1.039 as the range of specific gravity. Kane[6] estimates the weight of 1 cu. yd. air-dried light peat as used for domestic purposes at 500 lb.; of good peat at 900 lb., and of the densest peat at 1,100 lb. Grüner[7] gives 550 lb. as the weight of 1 cbm. (=1.3 cu. yd.) of light fibrous, and 840–880 lb. of dark earthy peat. Schorr[8] states that a good quality of raw peat with 85 per cent. H_2O weighs 63 to 66 lb. per cu. ft. The rational analysis shows peat to consist of peat-substance (C_{26}, H_{20}, O_2), water and ash. The range of composition of peat-substance was given in Table 71. The ultimate analyses, selected from Websky,[9] in Table 74, may serve as examples.

TABLE 74.—ULTIMATE ANALYSES OF PEAT

Locality	C	H	O	N	Ash
Grunewald	49.88	6.54	42.42	1.16	3.72
Havel	53.51	5.90	40.59[10]	....	....
Reichswald	58.69	7.04	35.32	1.79	2.04
Linum	59.47	6.52	31.51	2.50	18 53
Princetown	60.00	6.00	33.80[10]	....	10.00
Harz	62.54	6.81	29.24	1.41	1.09
Reichswald	63.86	6.48	27.96	1.70	2.70

[1] Davis, *Eng. Mag.*, 1909, XXXVIII, 810; *Bull.* 394, U. S. Geol. Surv., 1909.
[2] Dal, *Eng. Mag.*, 1902–03, XXIV, 204.
[3] Bache, *Engl. Instit. Civ. Eng.*, 1901, CXLVI, 229.
[4] *Verh. Verein. Beförd. Gewerb.*, 1904, LXXXIII, 171.
[5] *Op. cit.*, p. 21.
[6] Sexton, *op. cit.*, p. 58.
[7] *Op. cit.*, Vol. 1, p. 80.
[8] *Tr. A. I. M. E.*, 1905, XXXV, 82.
[9] *J. prakt. Chemie*, 1864, XCII, 65.
[10] Includes N.

The proximate analyses[1] of Ontario peats are of interest.

TABLE 75.—PROXIMATE ANALYSES OF AIR-DRIED PEAT OF THE UNITED STATES[2]

Kind of peat	Locality	Water	Ash	Sulphur	Heating value		
					Calories	B t u.	
						Air dried	Water free
Brown, fibrous. . .	Fremont, N. H	6 34	7 93	0.69	5,161	9,290	9,920
Brown, fibrous. .	Hamburg, Mich...	7.50	6 55	.28	5,050	9,090	10,026
Light-brown, fibrous	Rochester, N. H .	11 64	4.06	.22	5,042	9,083	10,280
Dark-brown	Westport, Conn. ..	12.70	4 12	.24	4,772	8,590	9,839
Brown, structureless.	New Durham, N. H.	6.06	17 92	88	4,415	7,947	8,460
Brown.	New Fairfield, Conn.	9 63	7 93	46	4,367	7,861	8,698
Brown, fibrous. ...	Westport, Conn.....	19.69	3.23	19	4,273	7,691	9,578
Brown	Kent, Conn... . .	12 10	7.22	.83	4,269	7,684	8,743
Brown, fibrous. .	Cicero, N. Y....	14 57	7.42	.25	4,209	7,576	8,869
Brown..	Black Lake, N. Y	8.68	16 61	.99	4,179	7,522	8,237
Brown, fibrous... ..	La Martine, Wis ..	9 95	16.77	.79	4,149	7,468	8,293
Salt marsh..	Kittery, Me	13 50	12 04	1 94	4,066	7,319	8,462
Black	Greenland, N. H....	6.62	24.11	1 01	3,992	7,186	7,695
Light-brown, structureless.	Waupaca, Wis......	6.62	24.44	.65	3,872	6,970	7,465
Brown, fibrous.	Madison, Wis. . . .	8 99	18 77	.38	3,857	6,943	7,628
Brown, sandy .. .	Kent, Conn.........	9.06	36 06	1 46	3,291	5,924	5,924
Black	———, N. Y...... ..	6 52	28 50	.57	2,867	5,161	5,521

The amount of water contained in peat has already been referred to. The percentage of ash varies greatly: 5 to 10 per cent. is a low figure, 10–20 per cent. is not uncommon; peat with 25+ per cent. ash ceases to be an industrial fuel. The ash consists mainly of silicates of Fe, Al and Ca; alkali and P are low, S in salt-water peat is generally high. The analyses[3] of Table 76 may serve as examples.

Air-dried peat begins to kindle[4] at about 250° C.; it burns with a shorter flame than wood. The calorific power[5] of peat is low; that of pure peat-substance was given in Table 71 as averaging 4,500 cal. Grüner[6] estimates 3,000 cal. to be a fair figure for peat with 25 to 30 per cent. H_2O and 6 to 8 per cent. ash.

The cutting and preparation of peat is carried on in spring and summer. The first step is to remove the surface sod or earth. The manner of further procedure depends upon the character of the bog. If the peat is sufficiently solid to permit cutting, the part of the bog to be worked is partially drained, the peat

[1] Bureau of Mines of Ontario, *Report*, 1903, p. 202.

[2] *Bull.* 16, Bureau of Mines, Washington, 1912, p. 53.

[3] Balling, C. A. M., "Metallurgische Chemie," Bonn, 1882, p. 165.

[4] Börnstein, *J. Gasbel. Wasserversorgung*, 1906, XLIX, 414.

[5] Andersson-Dillner, *Oest. Zt. Berg. Hüttenw.*, 1902, L, 102; see also Poole, *op. cit.*, p. 245 and Table 59.

[6] *Op. cit.*, I, 84.

cut in blocks, either by hand, with a long narrow spade ("slane") having a tongue set at right angles to its edge, and air-dried, or by a traveling excavator when the peat goes to a briquetting machine. If the peat is too soft and wet to allow cutting, it is dug by hand, spread, trodden to remove the water and molded ("molded peat"). The extraction of such peat on a large scale is carried on with dredges[1] or floating excavators. The material may be either molded directly into briquettes, or first purified by shredding, pulping, and washing, followed by a partial or complete drying (see Briquetting Peat, §109).

102. Brown Coal (Lignite).[2]—Brown coal, the youngest of the fossil fuels, comprises all fuels older than peat and of more recent origin than the Jurassic. Most brown coals occur in the Tertiary. They are widely distributed over the earth. In the United States,[3] cretaceous brown coals occur in

TABLE 76.—ANALYSES OF PEAT-ASH

Locality	K_2O	Na_2O	CaO	MgO	Al_2O_3	Fe_2O_3	P_2O_5	SO_3	SiO_2	HCl	CO_2	NaCl	SiO_2 CO_2	Cl	Chemist
Ireland...........	0.46	1.39	40.92	1.61	3.79	15.97	1.41	14.51	3.22	0.98	15.04	......	......	......	Kane.
Scotland..........	0.74	0.99	1.18	0.40	30.72		.....	5.52	......	......	......	......	60.62	......	Andersen.
Austria...........	0.56	0.65	15.32	1.37	14.73	8.76	1.07	2.59	......	......	10.08	45.56	......	......	Ferstl.
Bavaria...........	1.92	0.95	31.47	2.66	13.25		0.96	2.06	7.91	......	......	......	38.74	0.57	Zöller.
Holland...........	1.49	1.17	11.75	4.57	2.98	5.33	trace	9.77	0.86	......	......	1.50	51.57	......	?
United States......	20.69	0.58	40.52	6.06	5.17		0.50	5.52	8.23	......	......	......	31.71	0.15	Johnson.
United States......	0.80	......	35.59	4.92	......	9.08	0.77	10.41	1.40	......	......	......	37.32	0.43	Johnson.

[1] *Cass. Mag.*, 1896, x, 111.

[2] Zincken, C. F., "Die Physiographie der Braunkohle," Rümpler, Hanover; 1807; Suppl. I (Waisenhaus, Halle, 1872), II (Mentzel, Leipsic, 1878).

Stohmann-Kerl, "Handbuch der Technischen Chemie," 1893, IV, 443.

Richter, C., and Horn, P., "Die Deutsche Braunkohlenindustrie," Vol. II; "Die Mechanische Aufbereitung (Washing and Briquetting)," Knapp, Halle, 1910.

Donath, E., Zur chemischen charakteristik der Braunkohlen, *Oest. Zt. Berg. Hüttenw.*, 1912, LX, 281.

[3] *Report*, Tenth Census U. S., 1880, xv, 775, 791.

Dumble, E. F., "Brown Coal and Lignite of Texas," Austin, Tex., 1892.

Phillips, W. B., "Coal, Lignite and Asphalt Rocks of Texas," *Bull.* 3, University Texas, 1902; *Bull.* 15, University Series.

Phillips, W. B., and Worrell, S. H., "Texas Coals and Lignites," *Bull.*, University Texas, 189, Sc. Series, No. 19, 1911.

Randall, D. T., and Kreisinger, H., "North Dakota Lignite as a Fuel for Power-plants and Boilers," *Bull.* No. 2, Bureau of Mines, 1910.

Francforter, "Water in the Lignites of the Northwest," *J. Am. Chem. Soc.*, 1907, XXIX, 1488.

western North and South Dakota, and eastern Montana and Wyoming. Tertiary brown coals are confined to the South: Texas, Oklahoma, Arkansas, Louisiana, Mexico and Alabama. The physical and chemical properties of brown coals differ greatly with the degree of mineralization. Five varieties may be distinguished: lignite, earthy brown coal, brown coal proper, pitch coal, and glance coal. A sharp distinction cannot be made, as the varieties grade into one another. Another classification, that of Pontonic,[1] is based upon the character of the material from which the brown coal was formed.

LIGNITE includes trunks of trees more or less flattened out and altered chemically. Some varieties—the fossil wood—have the structure and appearance of wood that has been browned by decomposition; they can be cut and split as is ordinary wood; others—lignitic brown coal—have lost the wood-like structure and show a conchoidal fracture. Lignites occur as single trunks in sands and bogs, or massed together forming beds. When mined, lignite contains as much as 50 per cent. H_2O; it loses part of it by air-drying, but crumbles in the process. When dried artificially, it takes up again from 10 to 15 per cent. H_2O. The specific gravity of fossil wood is 0.5, of lignitic brown coal 1.3; 1 cu. ft. of the latter weighs 35 to 45 lb. The ultimate composition of lignite free from H_2O and ash shows the following range: C, 57 to 67 per cent.; H, 5 to 6 per cent.; O+N, 28 to 37 per cent. Fossil wood contains 35 to 40, lignitic brown coal 45 to 50 per cent. fixed C.; decomposition begins at about 210° C.; the amount of ash is not less than 4 and often reaches 15 per cent.; its composition is similar to that of peat except that it is practically free from P. Analyses are given in Tables 77 and 78. The calorific power of lignite free from water and ash is 5,000 to 5,500 cal., that of lignite with 30 per cent. H_2O and 10 per cent. ash is about 3,300 cal., of which 70 per cent. or 2,480 cal. may be figured upon in practical use.

EARTHY BROWN COAL is amorphous, earthy, friable, varies in color from yellow to brownish-black. The coal near Halle, Germany (containing some lignite and the hydrocarbons retinite, amber, and pyropissite), forms the basis of an important paraffine and mineral-oil industry.[2] It contains as much as 50 per cent. H_2O and 30 per cent. ash, and is rich in pyrite. The chemical composition and calorific power are similar to those of lignite. The coal is burned either in the form of fuel-dust (see §107) or of briquette (see §151).

BROWN COAL PROPER is compact, light blackish-brown, dull, shows traces of woody structure, has an even conchoidal fracture, a brown streak, and a specific gravity 1.20 to 1.25. It contains 5 to 10 per cent. H_2O; 1 cu. ft. loose

Hayes, U. S. Geol. Survey, *Twenty-second Ann. Rep.*, 1890–91, Part III; *Bull.* No. 213, 1903.

Burchard, *Bull.* No. 225, U. S. Geol. Survey, 1904.

Campbell-Parker, *Bull.* No. 394, U. S. Geol. Survey, 1909.

[1] Erdmann, *Braunkohle*, 1907, VI, 393.

[2] Klein, G., "Handbuch für den deutschen Braunkohlenberghau," Knapp, Halle, 1911.
Berg. Hüttenm. Z., 1890, XLIX, 306.
Stohmann-Kerl, Handbuch der Technischen Chemie, 1898, VI, p. 1899.
Graefe, E., "Die Braunkohlenteer Industrie," Knapp, Halle, 1906.

coal weighs 45 lb. It does not crumble much when exposed to the air.[1] The ultimate composition, when free from H_2O and ash, is C, 65 to 75 per cent., H, 4 to 6 per cent., O+N, 21 to 29 per cent. Distillation, which begins at about 250° C., yields 40 to 50 per cent. fixed C, which retains the form of the coal but is much fissured and friable; of the volatile matter, 15 to 20 per cent. is H_2O; 14 to 16 per cent. tar; 21 to 24 per cent. gas. The coal burns with a dull, smoky flame, having a characteristic disagreeable odor. The percentage of ash is high, but lower than that of earthy brown coal; it consists of silicates of Al_2O_3 and CaO, and is rich in Fe_2O_3 and SO_3. The calorific power when free from H_2O and ash, is 6,500 to 7,000 cal., that of the ordinary coal with 15 to 20 per cent. H_2O and ash 5,200 to 6,000 cal. It is burned direct as fuel-dust or in the form of briquette, and indirectly in a gas producer.

PITCH AND GLANCE COAL.—Pitch coal is compact, blackish-brown to pitch-black, has a waxy to greasy luster and an uneven to slightly conchoidal fracture. Glance coal is the firmest and hardest variety, it has a glassy luster and a perfect conchoidal fracture. Its specific gravity is 1.15 to 1.20, and contains 1 to 2 per cent. H_2O. The ultimate composition when free from H_2O and ash is: C, 70 to 80 per cent.; H, 6 to 8; O+N, 12 to 24 per cent. Distillation yields 30 to 45 per cent. spongy fixed C; the volatile matter is made up of H_2O 10 to 20 per cent.; tar, 20 to 35 per cent., gas 15 to 25 per cent. The coal burns readily with a long, smoky, sooty flame; it softens and even fuses and swells while burning.

The calorific power when free from H_2O and ash is 7,000 to 8,oco cal. The coal is used direct and in gas-firing.

BROWN COALS as a rule are readily distinguished from older varieties of peat. It is often impossible to draw a sharp line between an old brown coal and a young bituminous coal. Table 79 gives the behavior of the four natural solid fuels when subjected to some physical and chemical tests.[2]

103. Bituminous Coal.[3]—Bituminous coal is the leading metallurgical fuel. It was formed previous to the Jurassic and occurs mainly in the Carboniferous.

[1] Erdmann-Stoltzenberg, *Braunkohle*, 1908, VII, 69.

[2] Frémy, *Compt. rend.*, 1861, LII, 114.
Marcilly, *Ann. chim. phys.*, 1862, LXVI, 167.
Berthelot, *Compt. Rend.*, 1869, LXVIII, 183, 259, 334, 392, 445.
Percy, "Fuel," London, 1875, p. 300.
Donath, *Oest. Zt. Berg. Hüttenw.*, 1903, LI, 310.
Boudouard, *Rev. Mét.*, 1911, VIII, 38; *Eng. Min. J.*, 1909, LXXXVII, 995.

[3] Freise, History up to the Fifteenth Century, *Oest. Zt. Berg. Hüttenw.*, 1905, XLIX, 645, 661.

Muck, F., "Die Steinkohlenchemie," Engelmann, Leipsic, 1891.

Somermeier, E. E., "Coal, Its Composition, Analysis, Utilization and Valuation," McGraw-Hill Book Co., New York, 1912.

Microscopical Examination, *Bull. Soc. Ind. Min.*, 1899, XIII, 1100.

Distribution, Hayes, 22d *Ann. Rep.* U. S. Geol. Surv., 1902, Part III; *Bull*, 213, U. S. Geol. Survey, 1903.

Campbell-Parker, *Bull.* 394, U. S. Geol. Surv., 1909.

Parker, *Min. Res.*, U. S. Geol. Surv., 1911, Part II.

TABLE 77.—ULTIMATE ANALYSES OF BROWN COALS

	Locality	Sp. gr.	Water-free					H_2O	Character	Authority
			C	H	O+N	S	Ash			
Germany	Riestedt	1.218	61.13	5.09	31.95	....	1.83	31.66	Lignite	Zincken.
Germany	Loederberg	1.219	55.30	4.90	31.95	....	7.85	49.50	Earthy brown coal	Zincken.
Germany	Halle	....	59.00	5.50	24.00	....	11.50	45.00	Common brown coal	Preissig.
Austria	Wildshut	1.306	53.79	4.26	25.39	0.98	15.58	26.15	Lignitic brown coal	Zincken.
Austria	Hrastawetz	....	79.90	4.85	13.39	0.22	1.66	....	Pitch coal	*Oester. Zt., Berg. Huttenw.*, Wesen, 1905, LIII, 537.
Austria	Leoben	....	57.92	5.22	23.86	....	11.00	....	Glance coal	Zincken.
United States	Anderson County	....	53.02	4.06	24.12	1.02	17.74	....	Tertiary, Texas	Dumble.
United States	Smith County	....	57.40	3.60	23.31	0.95	14.74	9.83	Tertiary, Texas	Dumble.
United States	Gregg County	....	60.79	4.96	23.68	0.88	9.27	12.00	Tertiary, Texas	Dumble.
United States	Harrison County	....	66.32	3.95	21.56	2.20	8.97	13.35	Tertiary, Texas	Dumble.
United States	Weber Cañon	....	64.84	4.34	16.81	1.60	3.00	9.41	Cretaceous, Utah	*Tr. A. I. M. E.*, 1873–74, II, 61.
United States	Echo Cañon	....	69.84	3.90	12.92	0.77	3.40	9.17	Cretaceous, Utah	
United States	Carbon Station	....	64.99	3.76	16.94	1.07	1.68	11.56	Cretaceous, Wyo.	
United States	Carbon Station	....	69.14	4.36	10.79	1.03	6.62	8.06	Cretaceous, Wyo.	
United States	Cañon City	....	67.58	7.42	13.42	0.63	5.77	5.18	Cretaceous, Colo.	
United States	Baker County	....	60.72	4.30	14.42[1]	2.08	3.80	14.68	Cretaceous, Ore.	

[1] Air-dry.

TABLE 78.—PROXIMATE ANALYSES OF BROWN COALS

Locality	Vol. M.	Fix. C.	S	Ash	H_2O		Character	Authority
Texas	34 45	28.40	3 20	15 80	18 15	Tertiary	Lignite	Dumble.
Texas	43 95	25.40	0.08	14 15	16 55		Lignitic.	Dumble.
Texas	43 55	42.50	3.37	5 80	8 15		Brown coal.	Dumble.
Texas	35.16	39 56	3.68	6.30	15 30		Earthy brown coal.	Dumble.
Texas	29.06	39.73	1.33	23.08	8 15		Brown coal	Dumble.
Texas	53 70	32.55	0.89	6.75	7 00		proper.	Dumble.
Texas	41.52	33.40	0.96	7 70	16 40		Brown to	Dumble.
Texas	39.16	40.16	1.38	10 45	8 70		pitch coal.	Dumble.
Texas	37.14	41.19	3.02	6 50	12 15		Pitch coal	Dumble.
Texas	43.18	36.59	5.70	7.17	13 06		Glance coal.	Dumble.
Texas	48.10	41.05	0 80	10.85	....		....	*Bull.* 189, Univ. Tex., 1911.
South Platte, Colo.	28.64	43.31	..	5 10	22 95	Cretaceous	....	U. S. Geol. Surv., 22*d*, *Annl. Rep.*, 1900–1901, Pt. III, pp. 425, 501.
South Park, Colo.	33 30	48.00	.....	3 50	15 20		....	
Hams Fork, Wyo.	38 80	42.72	......	4.60	13 88		....	
Powder River, Wyo.	37.92	38.54	...	13 54	10 52		....	
Rock Springs, Wyo.	34 42	52.60	...	2.00	6.98		....	
Sheridan, Wyo.	37.55	44.70	....	4 70	13.05		....	
Clark's Fork, Mont.	38.22	48.33	......	6.92	6.53		....	
Trail Creek, Mont.	31.87	49.22	.....	8 57	10.51		....	
Widiston, N. D.	30	28	......	10?	32		....	*Bull.* 2, Bur. Min., 1910.

TABLE 79.—SOME CHARACTERISTICS OF SOLID FUELS

Fuel	Sp. gr.	Streak	Water absorbed when dry, per cent.	Blow-pipe	Distillation	HNO_3	H_2SO_4	$NaClO_3$	KOH
Peat	0.113–1.033	....	>20	Burns quickly with flame.	Acid distillate, small residue.	Slightly attacked.	Little attacked.	Not attacked.	Partly dissolved.
Brown coal	0.5–1.3	Brown-black.	>10	Burns quickly with flame and disagreeable odor.	Acid distillate, small residue.	Readily attacked, yellowish resinous residue or complete solution.	Completely decomposed, pale orange-yellow solution.	Complete solution.	Solution browned by younger varieties.
Bituminous coal	1.16–1.63	Black-brown.	<7	Burns slowly with flame.	Ammoniacal distillate, large residue.	Slightly attacked or not at all.	Attacked, solution light port-wine color.	Not attacked.	Not colored.
Anthracite	1.32–1.70	Black.	<5	Burns very slowly without flame.	Small distillate, no empyreumatic oils, large residue.	Not attacked....	Little attacked.	Not attacked.	Not colored.

On account of the complete mineralization all lignitic structure has disappeared. Supplementing Table 79, bituminous coal is darker, softer, more friable, heavier and richer in C than brown coal. Bituminous coal is massive, breaks, as a rule, readily parallel to the bedding planes, but shows no cleavage. The fracture is conchoidal to uneven. However, it does often break approximately at right angles to the bedding planes, which is due to a jointed structure (the cleat). Bituminous coal is brittle,[1] soft, black, and has a luster ranging from earthy dull through resinous to submetallic. Its specific gravity when containing little ash, varies from 1.25 to 1.35; one cu. yd. weighs about 1,800 lb.

Bituminous coal is composed of coal-substance,[2] water, ash and gases. The ultimate and proximate analyses are given in Table 71. It is held by Muck[3] that the coal substance does not contain any free C, and that a rational analysis will prove it to be composed of C–H–O–minerals. Tests along this line have been only partially successful.[4]

The amount of H_2O present averages 5 per cent.[5] The percentage of ash in high-grade coal ranges from 4 to 7 per cent.; in ordinary coal from 7 to 14 per cent.; it runs higher with inferior grades. Its distribution through the coal has been investigated by means of Roentgen rays.[6] Muck's collection[7] of 221 analyses of coal ashes shows the following range of composition: SiO_2, 1.70 to 60.13; Al_2O_3, 2.21 to 41.11; Fe_2O_3, 5.59 to 73.80; CaO, 1.08 to 21.37; MgO, 0.00 to 9.82. Ordinarily the ash is composed mainly of a clayey substance which contains SiO_2 45, Al_2O_3 30, Fe_2O_3 10, CaO and MgO 10, S, P, and Alkali 5 per cent.[8] Its fusibility is governed by the ratio of acid and bases as is that of all silicates (see § 191). Prost[9] found that the ashes fused at temperatures ranging from 1,160 to 1,500° C.; he calculated from the chemical analysis the refractory quotient $Q = \frac{\text{O in } Al_2O_3}{\text{O in} Fe_2O_3.CaO.Mgo} : \frac{\text{O in } SiO_2}{\text{O in } Al_2O_3}$ for 23 coke ashes

[1] Compressive Strength, *Mines and Minerals*, 1900, XX, 451.

[2] Wheeler, *Tr. A. I. M. E.*, 1907, XXXVIII, 621.
Parr-Wheeler, *Bull.* 37, Univ. Ill., 1909.

[3] *Op. cit.*, p. 4, *et seq.*

[4] Anderson, *Tr. Inst. Min. Eng.*, 1898–99, XVI, 335, review and research.
Bedson, *J. Soc. Chem. Ind.*, 1908, XXVII, 147.
Frazer-Hoffman, *Techn. Paper* 5, Bureau of Mines, 1912.
Lewes, *Coal Age*, 1912, II, 259.

[5] Carpenter, Hygrometric Properties of Coal, *Tr. Am. Soc. Mech. Eng.*, 1897, XVIII, 938.
Crisfield, *J. Frankl. Inst.*, 1911, CLXXII, 495.

[6] Courier, *Bull. Soc. Ind. Min.*, 1898, XII, 713; *Eng. Min. J.*, 1898, LXVI, 11; 1899, LXVIII, 547.
Knotte, *Stahl u. Eisen*, 1900, XX, 692.
Lomax, *Tr. Inst. Min. Eng.*, 1911, XLII, 2.
Garrett-Burton, *op. cit.*, 1912, XLIII, 295.

[7] *Op. cit.*, p. 98.

[8] Parr, S. W., and Wheeler, W. F., *Bull.* 37, Univ. Ill., Urbana, Ill., 1909.
Cobb, *J. Soc. Chem. Ind.*, 1904, XXIII, 11.

[9] *Rev. Un. Min.*, 1895, XXXI, 86 (*Coll. Guard.*, 1895, LXX, 796); *Bull. Assoc. Belge d. Chim.*, 1897, XI, No. 4 (*Coll. Guard.*, 1897, LXXIV, 602; 1898, LXXV, 473).

TABLE 80—ANALYSES OF COAL ASH

	Connellsville, Pa.	Crested Butte, Colo.	El Moro, Colo.	Cardiff, Colo.	New Straitsville, O.	Pigeon Creek, O.	Horr, Mont.	Belt, Mont.	Sangamon Co., Ill.	Peoria Co., Ill.	Dominion Iron & Steel Co's. coke, Nova Scotia.	Morris Run, Pa.	France	Dowlais, England	Belgium
SiO_2	44.64	45.25	84.50	48.9	58.75	37.40	57.0	51.6	33.1	25.7	1.84	60.95	47.30	35.73	49.46
Al_2O_3	25.12	23.07	8.50	23.9	35.30	40.77	21.2	33.6	17.9	6.8	1.88	35.34	27.30	41.11	33.28
Fe_2O_3	22.73	26.43	7.10	18.9	2.09	9.73	19.0	10.0	42.5	12.9	3.60	1.78	10.00	11.15	5.50
CaO	6.95	3.50		5.8	1.20	6.27	3.3		5.6	54.5	0.44	1.55	6.30	2.75	2.76
MgO	1.91	1.40		1.4	0.68	1.60			0.9	1.5	0.27	0.04	0.90	2.65	0.78
K_2O					1.08										
Na_2O						1.29							...		3.83
SO_3					0.24	1.99						0.28	1.10	4.45	1.50
P_2O_5	(a)				0.13	0.51							1.00	0.99	1.42
S				0.8	0.41	0.08					1.40				
Cl					trace	trace				none				...	
References	1	2	3	4	5	6	7	7	9	10	11	12	13	14	15

(a) 0.034420 P, in coke.

[1] *Report Tenth Census U. S.*, 1880, XV, 72. [2] Hofman, "Lead," N. Y., 1902, p 299. [3] Emmons. *Monograph*, XII, U. S. Geol. Survey, p. 642. [4] *Tr A. I. M. E.*, 1904, XXXIV, 289. [5] and [6] *Geol. Survey of Ohio*, 1870, pp. 130, 428; [7] *Tr. A. I. M. E.*, 1904, XXXIV, 289. [9] and [10] *Bull.* 37, Univ. Ill, 1909. [11] *Iron Age*, 1908, LXXXI, 760. [12] *Tr. A. I. M. E.*, 1892–93, XXI, 801. [13] *Bull. Soc. d'Enc.*, 1873, XX, 125. [14] Percy, "Fuel," London, 1875, p. 352. [15] *Rev. Un. Min.*, 1895, XXI, 86.

to express their relative fusibility (see Fire-clays, §158). LeChatelier[1] found a range of temperature of 1,150 and 1,450° C.

The S in coal[2] reaches sometimes 3 per cent.; usually it is under 2 per cent.[3] It occurs in combination with Fe as pyrite, with CaO as gypsum, and with organic matter. Organic sulphur[4] is the term for the S which remains after deducting from the total S that which is necessary to combine with Fe as FeS_2 and with Ca and Mg as $CaSO_4$ and $MgSO_4$. It is assumed to be organically combined with coal. In coking, part of the S remains with the coke (fixed S),[5] part passes off with the volatile matter (volatile S). It is customary to determine the total S in the coal and the coke; the difference between the two gives the volatile S. The volatile S comes from the pyrite[6] ($FeS_2=FeS+S$, or $2FeS_2=Fe_2S_3+S$[7], or $7FeS_2=Fe_7S_8+6S$) and the organic S. Part of the organic S is fixed.[8] Thus McCreath[9] found that from 22 samples of coal containing from

Example

No. of samples	Organic sulphur, p. c.	
	Total	Lost in coking
7	63.51	34.57
11	11.36	37 88
2	74.58	20.97
2	2.20	44.81

2.20 to 63.51 per cent. organic S, from 20.97 to 44.81 per cent. was volatilized. The P_2O_5[10] in the ash may reach 1.5 per cent., but usually is much lower, *e.g.*, 24 samples of coal from the Pittsburg bed,[11] Pa., averaged 0.0217 per cent. (= 0.0344 per cent. in Connellsville coke). According to Carnot[12] the P has its source principally in the spores and pollen of plants; cannel coal having more spores than other varieties of coal is richest in P. Catlett[13] found P in the "Big Seam," Columbiana, Ala., to occur in combination with Al_2O_3 as evansite ($Al_6P_2O_{14}+18aq$). All the P of a coal is found in the coke made from it.

[1] *Rev. Mét.*, 1905, II, 723.
[2] Schäfer, *Zt. Berg. Hütten. Sal. Wesen i. Pr.*, 1910, LVIII, 281.
[3] Weeks, "Manufacture of Coke," *Report*, Tenth Census U. S., 1880, X, 22.
[4] Drown, *Tr. A. I. M. E.*, 1881, IX, 656.
[5] Kimball, *Tr. A. I. M. E.*, 1880, VIII, 181.
[6] Simmersbach, *Stahl u. Eisen*, 1898, XVIII, 18.
[7] *Second Geol. Surv. Pa.*, Vol. MM, pp. 123–126.
[8] McCallum, *Can. Min. J.*, 1909, XXX, 531.
[9] *Second Geol. Surv. Pa.*, Vol. MM, pp. 126–127.
[10] LeChatelier-Durand Claye, *Bull. Soc. d'Enc.*, 1873, XX, 123.
[11] Campbell, Phosphorus in Pittsburg seam, *Mines and Minerals*, 1907–08, XXVIII, 408.
[12] *Compt. rend.*, 1884, XCIX, 154.
[13] *Tr. A. I. M. E.*, 1911, XLII, 902.

Occluded Gases.—Bituminous coal usually contains occluded gases. Mills-Rowan[1] show that 100 g. coal hold as much as 59.2 c.c. gas; they quote one seam in England which yielded at 20° C., 238 c.c. gas. Brookman[2] found in 100 g. Westphalian coal from 7 to 150 c.c. gas; he found it to be composed of CH_4, 0 to 96 per cent.; CO_2, 2 to 91 per cent.; N, 2 to 61 per cent.; C_xH_y, 0 to 3 per cent.; CO, 0 to 9 per cent.

Parr and Barker[3] and Barker[4] in the examination of Illinois coals found that as soon as the occluded gas, mainly CH_4, was released, absorption of O from the air which oxidized coal substance to CO_2 began. Other recent investigations are those of R. T. Chamberlin[5] and Porter-Ovitz.[6]

Collections of analyses are given by Mills-Rowan, Muck, Parr, Barker, Chamberlin and others. The gas (fire-damp) diffused through the pores of the coal and liberated as soon as the pressure is released is the cause of explosions in gassy mines. The gas which escapes in part rapidly at first, continues to do so after the coal has been mined and may then still prove to be a source of danger when the coal is stored. About 1/4 of the volume escapes in crushing the coal, the rest in from 3 to 18 months.

Spontaneous Combustion.—Coal when exposed to air disintegrates[7] (weathers). This is due to its taking up O, part of which oxidizes available H and, to a small extent, some C; the rest remains occluded. Weathering is likely to reduce the calorific value and the coking power of coal. Weathered coal may be heavier than when freshly mined; this will be the case when the absorbed O is heavier than the occluded gas and the oxidized C which pass off into the air. Spontaneous ignition[8] of coal rich in volatile matter when stored in unventilated pockets is caused principally by occluded O having an oxidizing effect upon H, C, and FeS_2. This takes place more rapidly with pure and fine than with impure lump coal, at a high than a low temperature,[9] and with moist coal than with dry.

[1] *Op. cit.*, pp. 84–85.
[2] *Berg. Hüttenm. Z.*, 1899, LVIII, 438.
[3] *Bull.* 32, University Ill., 1909.
[4] *Tr. A. I. M. E.*, 1909, XL, 24.
[5] *Bull.* 383, U. S. Geol. Surv., 1909.
[6] *Bull.* 1, Bureau Mines, 1910; *Technical Paper* 2, 1911.
[7] Wheeler, *Tr. A. I. M. E.*, 1908, XXXIX, 1153.
Brown, *Cass. Mag.*, 1909, XXXVII, 86.
White, *Bull.* 382, U. S. Geol. Surv., 1909.
Parr-Hamilton, *Bull.* 17, Univ. Ill., 1907.
Parr-Wheeler, *Bull.* 38, Univ. Ill., 1909.
Storage under Water, by Seidl, *Glück auf*, 1909, XLV, 37, 75, 119, 152.
[8] Porter-Ovitz, *op. cit.*
Porter, *Cass. Mag.*, 1912, XLII, 54.
[9] Lewis, *Coll. Guard.*, 1890, LIX, 521, 606; *Coll. Eng.*, 1890, X, 193; *Iron*, 1890, XXXV, 338; *Eng. and Min. J.*, 1906, LXXXII, 65.
Cremer, *Min. Ind.*, 1895, IV, 201.
Janda, *Oest. Zt. Berg. Hüttenw.*, 1903, LI, 376, 388.
Parr-Kressmann, *Bull.* 46, Univ. Ill., 1910.

Coking Coal—If coal is heated in a closed vessel, *i.e.*, with exclusion of air, it gives off its volatile matter, which burns with a yellow smoky flame, and leaves behind the fixed C. According to the behavior in this operation coal is said to be coking or non-coking. A coking coal softens, becomes pasty, fuses and swells into a spongy mass, bubbles of gas escaping, and solidifies, when all the volatile matter has been expelled, to a coherent, grayish-black, cellular, fritted mass—the coke. A non-coking coal changes its appearance only slightly, and leaves behind a sandy residue. Between these two extremes there are numerous gradations. It is not known what causes coking.[1] The ultimate analysis (Table 71) teaches that coal either high in C or high in O will not coke, but Stein[2] has shown that of two coals having the same ultimate composition and a suitable percentage of C and O, one may furnish a good coke, while the other will not. As the statements of Stein have been questioned, a more recent instance occurring in 1907 with the Maryland Steel Co., Sparrow's Point, Md., with carload lots may be given: Moshannon coal with V.H–C. 24.58, F.C. 68.02, ash 7.40, S 0.76, P 0.006 per cent. gave a sandy coke, while Greenwich coal with V.H–C. 24.36, F.C. 67.84, ash 7.80, S 0.89, P 0.004 per cent. gave a firm uniform coke. As coking is connected with decomposition, the process is not simply one of fusion. The older theory that the tar given off is charred and cements the particles into a solid mass, is unsatisfactory. It was believed that the available hydrogen was the cause of coking, but the proof is lacking.[3] The H:O ratio as a criterion looks more favorable. Thus White[4] finds that U. S. coals with an H:O ratio of 59 per cent. or over show good coking qualities; a ratio as low as 55 per cent. gives fair coke, but one of 50 poor coke if any. The cause of coking must be sought in the rational analyses of coal. Muck,[5] *e.g.*, heated cellulose, starch and gum, which have the same ultimate composition, in a closed vessel and found the following:

Material	Formula	Behavior resembles	Fixed carbon	
			Per cent.	Character
Cellulose	$(C_6H_{10}O_5)x$	Wood	6.71	Not fused, retains original form.
Starch	$(C_6H_{10}O_5)y$	Coking coal	11.30	Fused, much swollen, black.
Gum	$(C_6H_{10}O_5)z$	Steam coal	20.42	Fused, more dense and less lustrous than starch-coke, brownish-black.

Tests only can decide the matter. Orientation tests have been devised by

[1] Simmersbach, *Stahl u. Eisen*, 1904, XXIV, 446.
Lewes, *Coll. Guard.*, 1911, CII, 1126, 1178, 1231, 1284.

[2] "Chemisch und Chemisch-technische Untersuchungen der Steinkohlen Sachsens," Leipsic, 1857.

[3] White, *Bull.* 382, U. S. Geol. Surv., 1909, p. 55.

[4] *Op. cit.*, p. 52.

[5] *Op. cit.*, p. 5.

Boudouard and Pishel. Boudouard[1] found that humic acid was absent in a coking coal and present in non-coking coals. Pishel[2] observed that in pulverizing coal in a mortar, coking coal adhered strongly to the mortar, and non-coking only slightly or not at all. Usually the first test will be an examination of the fixed C obtained in heating the coal in a platinum crucible.[3] The next may be to determine the binding power as advocated by Campredon[4] who mixes 1 g. coal crushed through a 50-mesh sieve with increasing multiples of sand, passing through a 26- and remaining on a 50-mesh sieve, and cokes in a porcelain crucible at a red heat. After a few trials the largest amount of sand the coal can take and still form a coherent mass (*i.e.*, the binding power) is readily ascertained. This shows a range from 0 to 17 for coals, and is 20 for pitch.

Table 81.—Binding Power of Coals

Locality	Vol. H.C.	F. C.	Ash	Binding power
South Wales, Great Britain	10.90	82.90	6.20	....
New Castle, Great Britain	34.25	54.93	10.80	2
Scotland, Great Britain	34.72	56.93	8.35	4
Cardiff, Great Britain	19.80	72.50	7.70	6
Lens, France	27.20	64.10	8.70	13
New Castle, Great Britain	27.83	63.42	8.75	14
New Castle, Great Britain	29.50	62.00	8.50	17
Coal tar (pitch)	44.82	54.58	0.60	20

A similar method was advocated already in 1870 by Richters.[5] The third test will be to charge coal crushed through a 4-mesh sieve in suitable boxes into a coking oven to be coked with a regular charge. Such a box (a 6-in. cube) may be made of No. 15 iron wire woven to 10-mesh. Developed it will have the shape of a cross (Fig. 86), enough wire cloth to be left on the four sides to allow bending over. The inside of the box is covered with thin paper to prevent the coal from sifting through; the outside is held together by several loops of stout wire. When these preliminary trials have proved satisfactory, actual working tests[6] are in place.

[1] *Rev. Mét.*, 1909, VI, 446; *Eng. Min. J.*, 1909, LXXXVII, 995.

[2] *Econ. Geol.*, 1908, III, 265; *Eng. Min. J.*, 1908, LXXXVI, 479.

[3] Illustrations of the behavior of Westphalian coals are given by Muck, *op. cit.*, p. 31. See also Lessing, *Eighth Internat. Congress Appl. Chem.*, 1912, X, 195.

Stillman, Th. B., "Engineering Chemistry," Chem. Publ. Co., Easton, Pa., 1897, p. 19.

Heinrichsen-Taczak, *Stahl u. Eisen*, 1908, XXVIII, 1277; *Glückauf*, 1908, XIV, 1325.

Campbell, *Econ. Geol.*, 1911, VI, 502.

Stanton-Fieldner, *Technical Paper* 8, Bureau of Mines, 1912.

[4] *Compt. Rend.*, 1895, CXXI, 820; *J. I. and St. I.*, 1896, I, 348.

[5] Dingler, *Pol. J.*, 1870, CXCV, 71.

[6] Belden-Delamater-Groves-Way, *Bull.* 5, Bureau of Mines, 1910.

It is known that weathering, prolonged heating to 300° C.[1] and heating with imperfect exclusion of air diminishes or destroys the coking power, and this the more quickly the finer the coal. The calorific power of some coals is given in Table 71; 1 lb. coal will evaporate 6.70 to 9.50 lb. water.

There exist to-day several classifications of coal.[2] They are based upon the ultimate composition,[3] on the ratio of C to total, available and non-available H,[4] on the ratio of V.H–C to F.C.,[5] on the ratio of F.C. to total C in pure coal;[6] on the calorific value[7] or the ratio of C: (O+ash), and on the behavior upon heating in a closed vessel.[8] Owing to the great variety of the coals in different parts of the world showing peculiar properties of their own, no general classification has as yet been found which suits all cases. Different countries and districts have their own ways of solving the problem. Five classifications may serve as examples, those of Grüner, Frazer, Campbell, Parr and Grout.

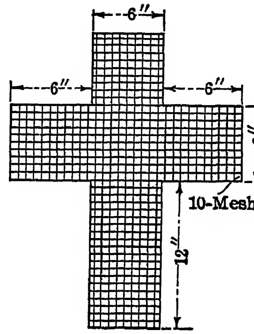


FIG. 86.—Developed wire coking-box.

Grüner's table (see Table 71), includes the ultimate composition, the ratio of O and H, the V.H–C and F.C., and the character of the F.C. Prepared in 1873, it was based upon the data then available. The numerous investigations made since, having extended the field, show that the lines drawn do not always agree with the accumulated facts of to-day. Thus some Westphalian coals of the Ruhr[9] district with 68 per cent. F.C. cannot be classed as steam or coking coals,

[1] Marsilly, *Ann. Min.*, 1857, XII, 347 and following.
Francis, *Tr. A. I. M. E.*, 1909, XL, 62.
Parr on "Coalite," *Eng. Min. J.*, 1907, LXXXIV, 734; *Min. Rep.*, 1907, LVI, 358; see also Boudouard, White and Porter-Ovitz, *loc. cit.*; Parr-Olin, *Bull.* 60, University of Ill., 1912; *Met. Chem. Eng.*, 1912, X, 755.

[2] Newton, *Mines and Minerals*, 1912, XXXIII, 139.

[3] Regnault, *Ann. Min.*, 1837, XII, 205.
Grüner, Traité de Métallurgie Générale, 1875, I, 67.
Seyler, *Proc. S. Wales Inst. Eng.*, 1898–1900, XXI, 483; 1900–1902, XXII, 112.
Boulton, W. S., "Practical Coal Mining," Gresham Publ. Co., London, 1907, Vol. I, pp. 67–95.

[4] Fleck, Dingler, *Pol. J.*, 1866, CLXXX, 460; CLXXXI, 48, 267; 1870, CXCV, 132.
Richters, *Zt. Berg. Hütten. Sal. Wesen i. Pr.*, 1871, XIX, 87.
Campbell, *Tr. A. I. M. E.*, 1906, 36, XXXVI, 1033.

[5] Hilt, Dingler, *Pol. J.*, 1873, CCLXVIII, 264.
Frazer, *Tr. A. I. M. E.*, 1877–78, VI, 449.
Kent, *Min. Ind.*, 1899, VIII, 125; *Mines and Minerals*, 1898-99, XIX, 111.
Parr, *J. Am. Chem. Soc.*, 1906, XXVIII, 1425; *Min. Rep.*, 1906, LIV, 394; *Eng. Min. J.*, 1907, LXXXIII, 1292; Bull. 3, Univ. Ill., 1908.

[6] Grout, *Econ. Geol.*, 1907, II, 225, 322, 1909; IV, 653.

[7] Campbell, *op. cit.*
White, *Bull.* 382, U. S. Geol. Surv., 1909.

[8] Schondorff, *Zt. Berg. Hütten. Sal. Wesen i. Pr.*, 1875, XXIII, 135.

[9] Muck, *op. cit.*, p. 16.

but are decidedly gas coals; again, some coals with 85–87 per cent. F.C. are not anthracitic, but decidedly coking coals. In the Saar district[1] the F.C. of coking coals varies from 61 to 72 per cent. which would bring them under Grüner's gas and steam coals. Proximate analyses of typical coking coals, Table 82, from the Appalachian field[2] show a range of F.C. from 53.27 to 73.02 per cent. which brings them all below the range 74 to 82 per cent. of Grüner's coking coal. Analyses[3] of Pennsylvania gas, steam and smithing coals tell a similar story.

TABLE 82.—PROXIMATE ANALYSES OF COKING COALS FROM THE APPALACHIAN FIELD

	Pennsylvania		West Virginia		Kentucky	Tennessee	Alabama
	Bennington	Connellsville	Monongahela	Pocahontas			
H_2O	1.73	1.26	1.52	0.69	1.80	1.50	1.65
V.H.–C.	23.89	31.79	37 96	19.96	32 34	32.51	32.48
F.C.	67.09	59.79	53.27	73.02	60 10	59.33	60.15
Ash	6.69	7.16	6.03	5.67	5.10	5.82	4.82
S	0.66	0.60	1.22	0.66	0 66	0.84	0.90

Nevertheless, Grüner's classification will remain a good starting-point until more is known about the constitution of coal.

Frazer's classification[4] of Pennsylvania coals is based upon the fuel ratio or the quotient of the F.C. divided by V.H–C. There are four classes as shown in Table 83. This scheme works well commercially with Pennsylvania coals, but fails from a general point of view in that it groups together all coals with a fuel ratio of 5 and less, and that it makes no provision for bituminous coals bordering upon brown coals.

TABLE 83.—FRAZER'S CLASSIFICATION OF COALS

Kind	F.C.	V.H–C.	$\frac{\text{F.C.}}{\text{V.H.-C.}}$
Bituminous	83.33	16.67–100	5 · 0
Semi-bituminous	88.89–83.33	11.11–16.67	8– 5
Semi-anthracite	92.31–88.89	7.69–11.11	12– 8
Anthracite (hard, dry)	100–92.31	0– 7.69	100–12

[1] Schondorff, *op. cit.*, p. 150.
[2] Fulton, "Coke," Scranton, Pa., 1905, p. 25.
[3] Halberstadt, *Coll. Eng.*, 1895–96, XVI, 257.
[4] *Tr. A. I. M. E.*, 1879, VI, 430; 1906, XXXVI, 825.

Campbell[1] tried the different schemes advocated for classification upon the 56 coals examined by the Coal Testing Plant of the U. S. Geological Survey at the Louisiana Purchase Exposition, St. Louis[2] and settled provisionally upon the C–H ratio as giving the most satisfactory basis.

TABLE 84.—CAMPBELL'S CLASSIFICATION OF COALS

Group	Name[3]	C:H
A	Graphite	∞
B	Anthracite	?–30?
C	Anthracite	30?–26?
D	Semi-anthracite	26?–23?
E	Semi-bituminous	23?–20
F	Bituminous	20–17
G	Bituminous	17–14.4
H	Bituminous	14.4–12.5
I	Bituminous	12.5–11.2
J	Lignite (brown coal)	11.2–9.3
K	Peat	9 3–?
L	Wood Cellulose	7.2

Parr[4] criticises Campbell's classification of total C : total H, as no distinction is made between inert H in combined water and available H in combustible volatile hydrocarbon, and the ratio F.C. : V.H.–C is not considered. He bases his classification, (1) upon the ratio of volatile and fixed carbon, $VC \times \frac{100}{C}$, in which VC is the difference between total carbon and fixed carbon, and C the total carbon; (2) upon the inert volatile matter (*i.e.*, H_2O) in coal.

[1] *Tr. A. I. M. E.*, 1906, XXXVI, 324.

[2] *Bull.* 261, U. S. Geol. Surv., 1905; *Eng. Mag.*, 1905, XXIX, 19; *Professional Paper* No. 48, U. S. Geol. Surv., 1906.

[3] Cannel coals are not included on account of the absence of accurate ultimate analyses.

[4] *J. Am. Chem. Soc.*, 1906, XXVIII, 1425; *Min. Rep.*, 1906, LIV, 394; *Bull.* 3, Ill. State Geol. Surv., 1906, p. 27; *Eng. Min. J.*, 1907, LXXXIII, 1242; *Circular* No. 3, Univ. Ill. *Bull.*, 1908, p. 11.

TABLE 85.—PARR'S CLASSIFICATION OF COALS

COAL	Anthracite	Anthracite proper	Ratio $\frac{VC}{C}$ below 4 per cent.
		Semi-anthracite.	Ratio $\frac{VC}{C}$ between 4 and 8 per cent.
		Semi-bituminous.	Ratio $\frac{VC}{C}$ between 10 and 15 per cent.
	Bituminous proper	A	Ratio $\frac{VC}{C}$ from 20 to 32 per cent. Inert volatile from 5 to 10 per cent.
		B	Ratio $\frac{VC}{C}$ from 20 to 27 per cent. Inert volatile from 10 to 16 per cent.
		C	Ratio $\frac{VC}{C}$ from 32 to 44 per cent. Inert volatile from 5 to 10 per cent.
		D	Ratio $\frac{VC}{C}$ from 27 to 44 per cent. Inert volatile from 10 to 16 per cent.
	Black lignites		Ratio $\frac{VC}{C}$ from 27 per cent. up. Inert volatile from 16 to 20 per cent.
	Brown lignites		Ratio $\frac{VC}{C}$ from 27 per cent. up. Inert volatile from 20 to 30 per cent.

Grout[1] uses as bases for classification both the fixed carbon and the total carbon in the pure coal.

TABLE 86.—GROUT'S CLASSIFICATION OF COALS

Name	Fixed, total carbon	Per cent. of pure coal
Graphite	F.C.	over 99
Anthracite	F.C.	over 93
Semi-anthracite	F.C.	83–93
Semi-bituminous	F.C.	73–83
Bituminous	F.C. T.C.	47–73 80–88
Sub-bituminous	F.C. T.C.	47–73 73.6–80
Cannel	F.C. T.C.	35–47 73.6–88
Lignite	F.C. T.C.	30–55 65–73.6
Peat and turf	F.C. T.C.	below 50 below 65
Wood	......	......

[1] *Econ. Geol.*, 1907, II, 225; 1909, IV, 653.

TABLE 87.—TYPICAL ANALYSES OF AMERICAN COALS

State	County	Kind of sample	Coal bed	As received	Proximate dry coal		
				Moisture	Volatile matter	Fixed carbon	Ash
Alabama	Bibb	Mine	Youngblood	3.34	34.72	56.98	8.30
	Bibb	Mine	Thompson	3.10	36.00	57.7	6.3
	Jefferson	Mine	Pratt	2.38	26.53	68.47	5.00
	St. Clair	Mine	Harkness	3.39	31.77	59.08	9.15
Arkansas	Pope	Mine	Shinn Basin	2.07	10.02	80.48	9.50
	Logan	Mine	Paris	2.77	15.11	75.56	9.33
	Sebastian	Mine	....	3.21	15.33	75.07	9.60
Colorado	Boulder	Mine	....	20.45	38.38	55.30	6.32
	Garfield	Mine	"A"	5.32	38.33	52.39	9.28
	Huerfano	Mine	Robinson	6.90	37.53	53.38	9.09
	Las Animas	Mine	....	2.28	30.50	60.13	9.37
	Routt	Mine	....	12.20	40.78	53.96	5.26
	Routt	Mine	Crawford 11-ft.	6.94	3.68	81.24	15.08
Illinois	Franklin	Mine	No. 7	11.50	30.17	59.51	10.32
	Saline	Mine	No. 5	5.56	36.44	54.33	9.23
	Williamson	Mine	No. 7	9.18	30.06	61.00	8.94
Indiana	Greene	Run-of-mine	No. 4	13.58	37.11	53.46	9.43
	Sullivan	Run-of-mine	....	12.03	40.53	47.10	12.37
	Vigo	Run-of-mine	....	13.53	40.25	47.31	12.44
Kentucky	Hopkins	Run-of-mine	No. 9	7.92	39.19	49.88	10.93
	Johnson	Mine	No. 1	6.43	38.69	57.85	3.46
	Webster	Mine	Owen (No. 11)	6.29	34.11	57.77	8.12
Maryland	Allegany	Mine	Upper Sewickley	3.4	15.5	77.8	6.7
	Allegany	Mine	Big Vein	2.7	14.5	78.0	7.5
Montana	Carbon	Mine	No. 5	9.31	37.64	50.54	11.78
	Dawson	Mine	Whole	34.55	54.00	35.00	11.00
	Fergus	Mine	McDonald Creek	14.13	31.87	56.40	11.73
	Yellowstone	Mine	Buckey	16.66	33.42	57.68	8.90
New Mexico	Colfax	Mine	....	2.12	36.84	51.31	11.85
	Rio Arriba	Mine	No. 1	1.71	36.89	56.07	7.04
	San Juan	Mine	....	15.79	41.55	47.32	11.13
Ohio	Belmont	Run-of-mine	No. 8	5.31	38.78	52.22	9.00
	Perry	Run-of-mine	No. 6	7.55	41.10	49.85	9.05
North Dakota	Stark	Mine	....	32.64	43.34	39.71	16.95
	Williams	Run-of-mine	....	38.92	41.81	49.37	8.82
Oklahoma	Coal	Mine	McAlester	7.07	39.18	49.16	11.66
	La Flore	Mine	McAlester	3.13	32.74	60.80	6.46
	Pittsburg	Mine	McAlester	4.54	39.29	51.74	8.97
	Pittsburg	Mine	Hartshorne	3.53	37.67	57.24	5.09
Pennsylvania	Allegheny	Mine	Pittsburg	2.60	33.54	61.00	5.46
	Cambria	Mine	"B"	3.52	17.94	75.96	6.1
	Fayette	Mine	Pittsburg	3.24	28.04	64.61	7.35
	Indiana	Mine	Upper Freeport or E.	5.02	27.40	64.39	8.21
	Washington	Mine	Pittsburg	1.44	35.12	58.61	6.27
	Westmoreland	Run-of-mine	Pittsburg	3.98	29.30	60.12	10.58
	Schuylkill	Mine	Diamond	2.76	2.55	84.40	13.05
	Schuylkill	Mine	Mammoth Middle Split.	2.80	1.19	90.75	8.06
	Sullivan	Mine	B-6 ft.	3.38	8.77	79.33	11.90
	Sullivan	Mine	B-Lower	3.4	9.7	78.2	12.09
Tennessee	Campbell	Mine	Rex	2.92	33.00	59.99	7.01
	Morgan	Run-of-mine	Brushy Mountain	5.59	35.61	54.05	10.34
Texas	Wood	Run-of-mine	....	33.85	41.57	47.39	11.04
Utah	Carbon	Mine	Upper	5.1	40.5	51.3	8.23
	Emery	Mine	Pugsley	4.0	42.6	51.2	6.17
	Summit	Mine	Wasatch	12.2	48.0	48.1	3.86
Virginia	Tazewell	Mine	Pocahontas No. 3	4.0	18.0	77.5	4.49
West Virginia	Fayette	Mine	Sewell	4.95	19.11	77.59	3.3
	Fayette	Mine	Fire Creek	3.6	17.46	76.16	6.38
	McDowell	Mine	Pocahontas No. 3	2.74	14.33	80.59	5.08
	Marion	Mine	Pittsburg	3.13	34.02	58.24	7.74
	Raleigh	Mine	Beckley	4.7	13.5	81.9	4.6
Wyoming	Bighorn	Mine	....	17.67	33.13	57.65	9.22
	Carbon	Mine	No. 1	9.57	47.21	44.72	8.07
	Sheridan	Mine	Carney	22.76	44.30	51.25	4.45
	Sweetwater	Mine	Upper Van Dyke	15.91	39.52	56.04	4.44

Ultimate dry coal					Air-drying loss	Calorific value dry coal		Kind of coal
Sulphur	Hydrogen	Carbon	Nitrogen	Oxygen		Calories	B.t.u.	
1.08	4.99	77.35	1.34	6.94	1.8	7,758	13,964	Bituminous.
.44	5.13	78.70	1.28	8.18	1.4	7,775	14,000	Bituminous.
1.50	5.02	83.67	1.57	3.24	1.5	8,244	14,839	Bituminous.
2.42	4.97	76.40	1.58	5.48	1.8	7,685	13,832	Bituminous.
1.78	3.46	81.97	1.50	1.79	1.4	7,773	13,991	Semi-anthracite.
2.87	3.82	80.95	1.50	1.53	2.2	7,870	14,167	Semi-bituminous.
3.22	3.50	80.97	1.57	1.14	2.4	7,799	14,038	Semi-bituminous.
.43	4.81	70.61	1.41	16.42	14.1	6,651	11,972	Sub-bituminous.
.80	4.93	71.57	1.67	11.75	3.0	7,290	13,122	Bituminous.
.57	5.03	74.27	1.10	9.94	4.4	7,207	12,973	Bituminous.
.51	5.09	75.97	1.11	7.95	1.5	7,834	14,101	Bituminous.
.50	4.73	74.02	1.61	13.88	4.8	7,198	12,956	Sub-bituminous.
.61	2.26	78.87	1.39	1.79	4.2	7,009	12,616	Anthracite.
.68	4.52	73.89	1.22	9.37	6.4	7,163	12,893	Bituminous.
3.04	4.81	72.80	1.38	8.74	3.6	7,437	13,386	Bituminous.
.99	4.49	75.37	1.26	8.95	5.4	7,350	13,230	Bituminous.
1.05	4.79	73.51	1.64	9.58	9.1	7,311	13,214	Bituminous.
4.85	4.73	69.04	1.23	7.78	7.2	7,069	12,724	Bituminous.
3.64	4.95	68.97	1.23	8.77	10.7	7,034	12,661	Bituminous.
3.82	4.90	70.90	1.52	7.93	2.2	7,253	13,055	Bituminous.
1.25	5.32	80.06	1.57	8.34	3.7	7,989	14,380	Bituminous.
1.44	5.11	74.46	1.46	9.41	4.5	7,632	13,738	Bituminous.
1.08	4.40	83.51	1.60	2.69	2.5	8,140	14,650	Semi-bituminous.
1.02	4.40	82.92	1.87	2.33	1.7	8,070	14,520	Semi-bituminous.
2.19	4.64	65.65	1.48	14.26	3.4	6,415	11,547	Sub-bituminous.
1.68	4.22	64.78	.86	17.46	15.5	6,018	10,833	Lignite.
4.30	4.06	69.72	.86	9.33	7.0	6,747	12,145	Bituminous.
1.20	4.51	71.06	1.16	13.17	7.2	6,817	12,271	Sub-bituminous.
.65	4.80	71.48	1.36	9.86	.5	7,359	13,246	Bituminous.
.66	5.11	76.69	1.59	8.91	0.4	7,758	13,964	Bituminous.
2.11	4.96	66.19	1.65	13.96	7.6	6,578	11,840	Sub-bituminous.
3.52	5.07	74.68	1.18	6.55	3.9	7,535	13,563	Bituminous.
3.07	5.02	72.50	1.40	8.96	4.5	7,288	13,118	Bituminous.
5.26	3.74	58.69	.73	14.63	10.4	5,748	10,346	Lignite.
.79	4.21	64.41	1.11	20.66	31.7	6,130	11,034	Lignite.
3.92	4.68	69.28	1.55	8.91	3.0	6,856	12,341	Bituminous.
.89	5.32	80.59	1.87	4.87	2.4	8,042	14,476	Bituminous.
3.38	4.62	73.43	1.68	7.92	1.9	7,397	13,315	Bituminous.
1.58	5.46	79.29	1.99	6.59	1.8	7,996	14,393	Bituminous.
.79	5.24	80.25	1.49	6.77	1.4	8,034	14,461	Bituminous.
1.10	4.55	85.06	1.27	1.92	2.8	8,222	14,800	Semi-bituminous.
.98	4.99	80.61	1.27	4.80	1.8	7,992	14,386	Bituminous.
2.04	4.92	80.26	1.48	3.09	4.2	7,931	14,276	Bituminous.
.79	5.14	79.91	1.46	6.43	... .	8,028	14,450	Bituminous.
1.04	4.71	76.38	1.59	5.70	3.1	7,702	13,864	Bituminous.
.56	1.97	81.47	.70	2.25	1.6	7,185	12,933	Anthracite.
.92	1.63	86.78	.65	1.96	1.5	7,601	13,682	Anthracite.
.65	3.31	81.18	1.04	1.92	2.6	7,565	13,617	Semi-anthracite.
.84	3.37	80.59	.98	2.13	2.9	7,545	13,580	Semi-anthracite.
1.17	5.02	77.21	1.67	7.92	1.5	7,734	13,921	Bituminous.
3.42	4.89	74.23	1.72	5.40	4.3	7,558	13,604	Bituminous.
.77	4.41	65.18	1.07	17.53	26.7	6,296	11,333	Lignite.
.54	5.25	74.49	1.66	9.83	1.5	7,385	13,290	Bituminous.
.41	5.29	76.01	1.30	10.82	.3	7,500	13,500	Bituminous.
2.16	5.07	72.61	1.45	14.85	4.5	7,120	12,820	Sub-bituminous.
.65	4.49	86.04	1.07	3.26	3.5	8,405	15,130	Semi-bituminous.
.86	4.78	86.43	1.56	3.07	4.1	8,437	15,187	Semi-bituminous.
.69	4.69	84.05	1.61	2.58	2.7	8,193	14,747	Semi-bituminous.
.61	4.50	86.61	1.32	1.88	1.8	8,365	15,057	Semi-bituminous.
.93	5.16	77.41	1.54	7.22	1.6	7,745	13,941	Bituminous.
.87	4.40	86.36	1.56	2.21	4.1	8,360	15,050	Semi-bituminous.
.20	4.11	69.33	1.02	16.12	6.3	6,537	11,767	Sub-bituminous.
.43	4.91	69.47	1.45	15.67	1.4	6,910	12,438	Sub-bituminous.
.45	4.53	71.06	1.32	18.19	6.6	6,783	12,209	Sub-bituminous.
1.31	5.02	73.44	1.53	14.26	11.5	7,187	12,937	Bituminous.

[1] Selected from *Bulletin* No. 22, Bureau of Mines, 1913.

104. Anthracite.[1]—Excluding graphite, anthracite represents the ultimate stage of mineralization of vegetable matter. Pennsylvania anthracite is very homogeneous, hard, compact, brittle, jet-black, often shows iridescent colors, has a conchoidal fracture, the luster is vitreous to sub-metallic; specific gravity, 1.6; weights of broken coal are given below, 1.6–1.8 cu. ft. broken coal equals 1 cu. ft. coal in place; is difficult to ignite; burns almost without flame (at least the harder varieties); contains frequently occluded gas (CH_4). Chance[2] gives the figures of Table 88 as the range of composition:

TABLE 88.—RANGE OF ULTIMATE COMPOSITION OF ANTHRACITE

Pennsylvania anthracite	Location	C, per cent.	H, per cent.	H_2O, per cent.	Ash, per cent.	O, N and other gases, per cent.	Specific gravity
Hard (dry)..	Eastern Middle Field; eastern part of Southern, Western Middle and Northern Fields.	91–98	0–3	0.5–2.5	1–?	0–2 or 3	1.45–1.75
Semi (soft)...	Western part of Southern, Western Middle and Northern Fields.	85–90	Principally combined.	1 5–3	3–?	5–10	1.57–1.65

The proximate analyses by Ashburner,[3] Table 89, give the general character of the four Pennsylvania anthracite fields.

TABLE 89.—PROXIMATE ANALYSES OF PENNSYLVANIA ANTHRACITE

Field	Eastern Middle		Western Middle				Southern		Northern or Wyoming
District	Hazelton		Mahanoy, Shamokin				Lehigh, Pottsville		Wilkes-barre, Scranton
Bed	Wharton	Mammoth	Primrose	Mammoth	Buck Mountain	Seven-foot	Primrose(?) F. bed	Mammoth	Mammoth
H_2O	3.713	4.119	3.541	3.163	3.042	3.410	3.008	3.087	3.421
V. M.	3.080	3.084	3.716	3.717	3.949	3.978	4.125	4.275	4.381
F. C.	80.404	86.379	81.590	81.143	82.662	80.868	87.982	83.813	83.268
S	0.585	0.496	0.499	0.899	0.462	0.512	0.506	0.041	0.727
Ash	6.218	5.922	10.654	11.078	9.885	11.232	4.379	8.184	8.203
	100.00								
Sp. gr.	1.620	1.617	1.654	1.657	1.607	1.651	1.584	1.631	1.575

[1] Ashburner, *Tr. A. I. M. E.*, 1885–86, XIV, 706.
Chance, *Report*, A. C., 2d Geol. Survey of Pa.

[2] *Op. cit.*, p. 18; 22d *Ann. Report*, U. S. Geol. Survey, 1900–01, Part III, p. 61.

[3] *Mineral Resources U. S.*, 1883–84, p. 69.

Some anthracite is mined in Colorado, New Mexico, and a small amount in Rhode Island and Massachusetts.

TABLE 90.—PROXIMATE ANALYSES OF ANTHRACITE FROM COLORADO, NEW MEXICO, RHODE ISLAND AND MASSACHUSETTS

Locality	H_2O	V.H.-C.	F. C.	Ash	Reference
Colo., N. M......	1 56	5 93	88 76	3 75	*Eleventh Census, U. S.*, 1890, Mineral Industries, p. 361.
R. I...	....	2.60	78.00	15 06	*Tr. A. I. M. E.*, 1887–88, XVI, 709.
Mass............	1 02	3 76	74.24	20 97	*Tr. A. I. M. E.*, 1884–85, XIII, 515.

Anthracite is used principally (65 per cent. total) as a domestic fuel; it forms a blast-furnace fuel; in small sizes (pea, buckwheat, rice, barley) it serves for raising steam, and is a good reducing agent for zinc ores.

PREPARATION OF ANTHRACITE.—Pennsylvania anthracite goes through a breaker to be sized and cleaned for industrial use.[1] As the consumption varies with the time of year, producers require large storage-plants.[2]

The data in Table 91 of the Susquehanna Coal Co.[3] are representative of the commercial grades.

TABLE 91.—COMMERCIAL SIZES OF ANTHRACITE

Grade	Size of screen, inches		Weight per cu. ft. lb.	From 1 cu. ft. solid, cu. ft.
	on	through		
Lump..................................	$4\frac{1}{2}$–9	....	57	1.614
Broken..................................	$2\frac{3}{8}$–$2\frac{7}{8}$	$3\frac{1}{4}$–$4\frac{1}{2}$	53	1.755
Egg..................................	$1\frac{3}{4}$–$2\frac{1}{4}$	$2\frac{3}{8}$–$2\frac{7}{8}$	52	1.769
Large Stove..............................	$1\frac{1}{4}$–$1\frac{7}{8}$	$1\frac{3}{4}$–$2\frac{1}{4}$	51.5	1.787
Small stove..............................	1–$1\frac{1}{4}$	$1\frac{1}{4}$–$1\frac{1}{2}$	51.25	1.795
Chestnut..............................	$\frac{5}{8}$–$\frac{3}{4}$	1–$1\frac{1}{4}$	51	1.804
Pea..................................	$\frac{3}{8}$–$\frac{5}{8}$	$\frac{5}{8}$–$\frac{7}{8}$	50.75	1.813
No. 1 buckwheat..............................	$\frac{3}{16}$–$\frac{3}{8}$	$\frac{3}{8}$–$\frac{5}{8}$	50.75	1.813
No. 2 buckwheat..............................	....	$\frac{3}{16}$–$\frac{3}{8}$	50.75	1.813

With the decrease in size there is usually an increase in the percentage of ash as seen in Table 92.[4]

[1] Coxe, *Tr. A. I. M. E.*, 1890–91, XIX, 398.
Sterling, *op. cit.*, 1911, XLII, 264.

[2] Norris, *Tr. A. I. M. E.*, 1911, XLII, 314.

[3] *Eng. and Min. J.*, 1889, XLVII, 497; see also Sterling, *loc. cit.*

[4] Ashburner, *op. cit.*, p. 720.

TABLE 92.—RELATION OF SIZE AND ASH-CONTENT IN ANTHRACITE (PER CENT.)

Grade	H_2O	V.H.-C.	F. C.	S.	Ash
Egg....	1.722	3.518	88 489	0 609	5.662
Stove....	1.426	4 156	83.672	0.572	10.174
Chestnut....	1.732	4 046	80.715	0 841	12.666
Pea....	1.700	3 894	79 045	0.697	14.664
Buckwheat....	1.690	4.058	76.918	0.714	16.620

105. Cannel Coal.[1]—This occurs interstratified in bituminous coal, but differs much from it. It is hard, compact, takes a good polish, is dark grayish-black, dull, shows no banded structure, resembles black shale, has a conchoidal fracture with smooth surface; specific gravity 1.23–1.28. It is easily kindled, burns readily with a long luminous flame and decrepitates upon heating.

Grüner[2] gives as ultimate analysis of Lancashire (England) coal free from ash and moisture: C, 85.81; O+N, 8.34; H, 5.85 per cent. Proximate analyses of some Kentucky coals have been published by Macfarlane.[3] More recent analyses[4] are given in Table 93. Cannel coal is used almost exclusively for gas-making, occasionally in open-grate fires.

BITUMINOUS SHALES OR BOGHEAD COAL[5] may be considered as a shale impregnated with bitumen[6] similar in composition to cannel coal. Table 94 gives the results of distilling California bituminous shales.[7]

TABLE 93.—ANALYSES OF KENTUCKY CANNEL COAL

Sample	No. 1	No. 2	No. 3	No. 4	No. 5	No. 6	No. 7	No. 8	No. 9
Moisture....	1.45	1.65	1.38	1.03	1 30	1.33	1 20	1.52	1.20
Volatile....	45 25	51 80	50 19	42.12	53.25	52.72	38.75	46.08	41.80
Fixed carbon..	28.50	33.25	32.21	27.85	33.50	32.66	34.43	42.47	46.00
Ash....	24.80	13 30	16.22	29.00	11.95	13.29	25.62	9.03	11.00
Total....	100.00	100.00	100.00	100.00	100.00	100.00	100.00	100.00	100.00
Sulphur....	1.57	1.78	1.46	2.74	1.68	1.57	0.54	0.79	0.901
Coke....	53.50	46.55	48.43	....	....	29.37	....	....	....
Specific gravity...	1.406	1.283	1.308	1.417	1.248	1.227	1.362	1.268	....
Color of ash....	brown	brown	brown	lilac	lilac	lilac	gray	brown	....
B.t.u., per pound	12,045	13,160	12,605	11,289	13,060	12,530	10,605	12,900	....

1 Microscopical Examination, *Bull. Soc. Ind. Min.*, 1899, XIII, 1057.

2 *Op. cit.*, Vol. 1, p. 125.

3 *Tr. A. I. M. E.*, 1889–90, XVIII, 436.

4 Ky. Geol. Surv., Progress Rep., 1908–09, p. 117.
Pfening, *Coal Age*, 1912, II, 546.

5 Microscopical Examination, *Bull. Soc. Ind. Min.*, 1899, XIII, 988.

6 Eldridge, *Twenty-second Ann. Rep.*, U. S. Geol. Surv., 1900–1901. *Bull.* 213, U. S. Geol. Surv., 1903, p. 296.
Day, *Mineral Resources U. S.*, 1910, II, 833; includes Bibliography of U. S., Geol. Survey, p. 839.
Hubbard, P., "Bitumens and their Essential Constituents," U. S. Dept. Agriculture Office Public Roads, Circular No. 93, 1911.

7 Cooper, *Min. Sc. Press*, 1899, LXXVII, 149.

TABLE 94.—DISTILLATION OF BITUMINOUS SHALE

Products of distillation		Good shale High	Good shale Low	Boghead coal High	Boghead coal Low
V.M.	Gas	13.65	2 54	37.32	4 83
	Ammonia water	3 65	6 47	2.43	3.23
	Tar and oil	11.04	17.65	20.65	50.29
	Sulphur	0 99	......	0.18	....
	Water 100° C.	2.82	....	0.80	....
		32.15	26.66	61.38	58.35
F.C.	Carbon	4.16	26.66	9 01	12 40
	Sulphur	1.05	10.81	0.06	....
	Ash	62.64	65.53	29.55	29.25

106. Coal Statistics.—The world's production of coal in 1910[1] was 1,300,000,000 short tons, of which the U. S. contributed about 39 per cent.

TABLE 95.—WORLD'S PRODUCTION OF COAL
(million short tons)

Country	United States	Great Britain	Germany	Austria-Hungary	France	Belgium	Russia and Finland
Million short tons.	501.6	296 0	245.0	54.6	42 5	26 4	25.0

Country	Japan	India	China	Canada	New South Wales	Spain	Transvaal
Million short tons.	16.5	13.3	13.2	12.8	7.9	4.5	4.4

Country	Natal	New Zealand	Mexico	Holland	Queensland and Victoria	Others
Million short tons	2.6	2.1	1.4	1.2	1.11	27 9

The coal fields of the U. S. include an area of something over 496,000 sq. miles. Table 96 gives the classification of the U. S. Geological Survey and some statistical data.

[1] Mineral Resources, U. S. Geol. Surv., 1910, Part II, p. 58.

TABLE 96.—CLASSIFICATION AND STATISTICS OF U. S. COAL FIELDS

Coal Field	Area, square miles	Production in 1910, million short tons	Price per short ton, $
ANTHRACITE			
Pennsylvania	480	2.2	1.90
Colorado, New Mexico	29	Including Rocky Mountain.	?
BITUMINOUS			
Including brown coal or lignite, etc.			
Eastern Province:			
Atlantic Coast Region: Virginia, North Carolina.	210	Including Appalachian.	Va. 0.90, N. C. ?
Appalachian Region: Pennsylvania, Ohio, Maryland, Virginia, West Virginia, Eastern Kentucky, Tennessee, Georgia, Alabama.	69,332	287.8	Pa. 1.02, O. 1.05, Md. 1.12, Va. 0.90, W. Va. 0.90, Ky. 0.99, Tenn. 1.11, Ga. 1.46, Ala. 1.26.
Interior Province:			
Northern Region: Michigan.	11,000	1.5	Mich. 1.91.
Eastern Region: Indiana, West Kentucky, Illinois.	48,500	72.6	Ind. 1.13, Ky. 0.99, Ill. 1.14.
Western and Southwestern Region: Iowa, Missouri, Kansas, Arkansas, Oklahoma, Texas.	54,244	22.3	Ia. 1.75, Mo. 1.79, Kans. 1.61, Ark. 1.56, Okla. 2.22, Tex. 1.07.
Rocky Mountain and Great Northern Plains Provinces:			
Arizona, North Dakota, Montana, South Dakota, Wyoming, Utah, Colorado, New Mexico, Idaho.	124,671	28.8	Ar. ?, N. D. 1.49, Wyo. 1.55, Vt. 1.68, Colo. 1.42, N. M. 1.39, Id. 3.92.
Pacific Coast Province and Alaska:			
Washington, Oregon, California, Alaska.	1,830	54.0	Wash. 2.50, Ore. 3.48, Cal. and Al. 2.74.
Grand total	310,296[1]	501.6	Av. bituminous 1.12. Av. anthracite 1.90. General av. 1.25.

Comparing the area and production, it will be seen that the Appalachian region, second largest in area, is first in production, while the Northern Plains Provinces, largest in area, stand third in rank. This is due to the fact that the best steam and coking coals come from the Appalachian region. It may be said[2] that passing from East to West, the different coal fields diminish in value.

[1] Not including 100,705 square miles of which little is known, but which may contain workable beds, and 31,805 square miles where coal lies under heavy cover and is not at present available.

[2] U. S. Census, 1880, Vol. xv, p. 617.

Their decrease in F.C. and corresponding increase in V.H.–C. is shown by the following table of McCreath:[1]

	Iowa	Illinois	Indiana	Ohio	Pennsylvania				Maryland, Cumberland	Pennsylvania		Rhode Island, graphitic anthracite
					Pittsburg	Connells-ville	Johns-town	Benning-ton		Semi-anthra-cite	Anthra-cite	
H_2O.. ..	1.40	1.25	1 10	2.70	1.28	1.26	1 03	1.20	0.89	1 25	1.35	1.18
V.H.-C..	41 40	41.85	37 06	33.49	38.10	31.79	16.49	23.33	15.52	9.60	3.45	3.80
F.C...	48.50	48 90	57 59	56 90	54.39	59.79	73.84	69.02	74.29	81.30	89 06	85.70
Ash	7.50	7.00	3.50	5.99	5.44	7.16	7.97	5.69	8.59	6.90	5.81	8 52
S..	1 20	1.00	0 75	0 92	0 79	0 60	1.97	0 76	0.71	0 85	0.30	0.80

107. **Pulverized Fuel** (FUEL DUST).[2]—In mining, much fine coal is produced. To the different methods of utilizing non-coking mine-fines, such as burning on step-grates, agglomerating, using in gas-producers, etc., a new one has been added which converts the coal into fine dust and then burns it, thoroughly mixed with air to secure perfect combustion, in special fire-places heated to sufficiently high temperatures to ignite the coal. The first attempts at burning fuel dust probably were those of Niepce[3] in 1818. Crampton in 1868 returned to the subject; it lapsed again until 1892 when with Wegener it took a new start. Since then different methods and various mechanical apparatus have been devised, but on the whole the method has made only slow progress.

In order to work satisfactorily, it is essential that the V.H.–C. of the coal be over 30 per cent., as a short-flame coal with under 20 per cent. V.H.–C. does not ignite; that the moisture be uniform; that the coal slack be finely and uniformly pulverized and intimately mixed with the air requisite for perfect combustion; and that the mixture be conveyed automatically and continuously into an hot ignition chamber. If these conditions are fulfilled, and the coal-dust par-

[1] From Fulton, J., "Coke," Scranton, Pa., 1905, p. 24.

[2] Colomer, *Coll. Guard.*, 1895, LXIX, 798.
Donkin, *Tr. Fed. Inst. Min. Eng.*, 1895–96, XI, 321.
Gary, *Thonind. Z.*, 1896, XX, 545, 564, 581.
Editor, *op. cit.*, 1899, XXIII, 1665, 1677.
Travis, *Power*, 1904, XXIV, 168.
Carey, *J. Soc. Chem. Ind.*, 1905, XXIV, 369; *Eng. Min. J.*, 1905, LXXX, 1113; 1906, LXXXI, 901.
Bartlett, *Iron Age*, Nov. 6, 1902; *J. Assoc. Engin. Soc.*, 1903, XXI, 44.
Ennis, *Eng. Mag.*, 1907–08, XXXIV, 463, 577.
Meade, *Tr. Am. Inst. Chem. Eng.*, 1909, I, 99; *Electrochem, Met. Ind.*, 1909, VII, 61.
Hughes, *op. cit.*, 1911, IV, 349.
Editor, *Eng. Min. J.*, 1908, LXXXV, 642.
Hart-Davis, *Tr. Inst. Min. Eng.*, 1912, XLIII, 112.
Sörensen, *Eng. Min. J.*, 1906, LXXXI, 274.
Shelby, *op. cit.*, 1908, LXXXV, 542.
Thomas, *op. cit.*, p. 660.
Browne, *J. Can. Min. Inst.*, 1912, XV, 113.

[3] *Ann. Mines*, 1875, VII, 176.

ticles are surrounded and held in suspension by the requisite air, approximate perfect combustion can be obtained. An excess of air cools the flame and causes smoking, a lack causes the combustion of the V.H.–C., while the F.C. cokes.

Coal with 2 per cent. H_2O makes a flame about 10 ft. long; with 0.5 and less per cent. the flame is shorter; hence in using pulverized fuel in a puddling furnace stress is laid upon having the coal thoroughly dried before it is crushed.

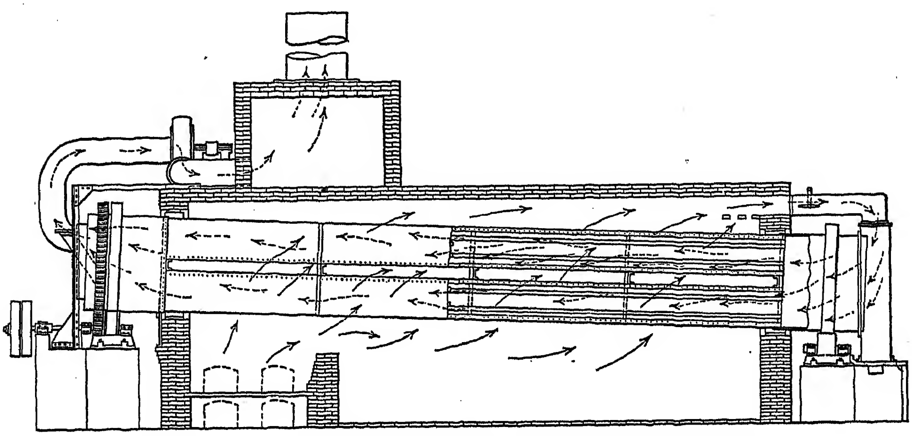


FIG. 87.—Four-compartment dryer, C. O. Bartlett & Snow Co.

The many advantages claimed for the method, such as small consumption of coal, approximate perfect combustion, raising of the boiler efficiency from 50 to 75 per cent., perfect control of fire, cheapness of attendance, small formation of clinkers, ease of stopping, urging and slackening of fire, use of low-grade fuel, have hardly been all realized. They are said to more than balance the cost of grinding the coal. Slack is ground at the rate of 35 cents a ton.[1] The slack to be ground should not contain over 2 1/2 per cent. H_2O, a larger amount causes the particles to ball in crushing, to pack in feeding the dust into the burner, and prevents the necessary intimate contact of dust and air. The dryers[1] in common use are externally-fired revolving inclined cylinders. The apparatus is placed in a separate building, windows and doors are kept open to carry off dust, rafters and roof are swept weekly to prevent accumulation of explosive dust.. The dryer of the C. O. Bartlett & Snow Co. of Cleveland, O., shown in Figs. 87

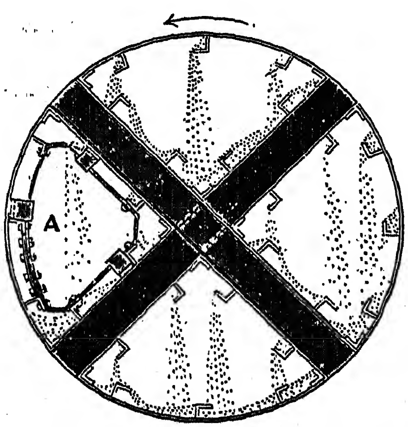


FIG. 88.—Four-compartment dryer, C. O. Bartlett & Snow Co.

[1] Bryan, *Am. Machinist*, 1906, p. 52.

[2] Ennis, *Eng. Mag.*, 1907–08, XXXIV, 463.

and 88, may serve as an example. It is an inclined hollow iron revolving cylinder with 4 compartments, which receives nut coal at the upper end, and is heated from a fireplace situated beneath the feed. The hot gases playing around the cylinder are drawn by an exhaust fan toward the discharge end, leave the heating chamber through the roof, pass downward through a flue and rise through the cylinder in contact with coal which in its downward travel is showered through the ascending hot-air current; this passes off near the feed end into the fan. A No. 6 dryer, 27 ft. long and 4 ft. diam., treats per hr. 6 tons coal reducing the water from 10–12 per cent. down to 1 per cent. and requires 5 1/2 h.p.; a No. 10 dryer, 28 ft. long and 5 ft. diam., treats per hr. 10 tons coal with an expenditure of 9 h.p. With coal at $3.00 per ton the cost of heating-fuel is about 6 cents per ton nut coal to be dried.

The coal to be ground should not be coarser than 1.5 in. The dust ought to be fine enough for 75 per cent. to pass through a 200-, and 95 per cent. through a 100-mesh screen. The grinding and pulverizing machinery used shows the greatest variety. Common machines (§270 and foll.) are the Ball Mill, the Huntington, Griffin, Kent, Fuller-Lehigh, and Raymond Roller Mills, and the Tube Mill, the Hardinge Mill. As a rule the comminuting is separated from the feeding apparatus; the coal is ground, delivered to a receiver and thence transferred to the feed-hopper of the burner. In a few instances, notably in the Day or Ideal apparatus,[1] and the Aëro Pulverizer Co.'s device,[2] the breaking and pulverizing of coal, mixing with air, and delivering the mixture to the furnace is one continuous process. This combination tends to complicate the machine and to increase the power necessary for operating.

The Aëro Pulverizer No. 10, at Lebanon, Pa., making 975 r.p.m. and requiring 20 h.p., grinds per hr. 1,400 lb. crushed coal containing from 2 to 3 per cent. H_2O to a product of which 20 per cent. will be retained on a 100-mesh sieve. The fuel-dust is burnt in a nodulizing furnace used for agglomerating magnetite concentrates.

A central-discharge tube-mill, 22 ft. long and 5 ft. diam., making 20 r.p.m. grinds per hr. 6 tons coal (containing 0.5 per cent. H_2O) coarsely pulverized in a Huntington mill. The dust, of which 10 per cent. is coarser than 100-mesh, is used in firing a rotary cement kiln.

At the works of the Canadian Copper Co., Copper Cliff, Ont.,[3] two Raymond Impact Pulverizers convert in 10 hr., to nearly all finer than 200-mesh, 50 tons coal (10 per cent. ash) which has been crushed to nut size and dried in a Ruggles-Cole dryer. The dust serves to fire reverberatory matting furnaces.

Special precautions have to be taken in grinding to guard against dust explosions. On account of the danger of spontaneous ignition, the coal-dust may not be stored over 2 or 3 days; ordinarily it is used up nearly as fast as it is

[1] *Power*, 1904, XXIV, 170; *Am. Mfr. and Iron World*, 1900, LXV, 316.
Ennis, *Eng. Mag.*, 1907–08, XXXIV, 472.

[2] *Eng. and Min. J.*, 1906, LXXXI, 274.

[3] *J. Canad. Min. Instit.*, 1912, XV, 115.

ground. It has been suggested to transport coal-dust in pipes in the form of a liquid mud, to settle it at the place of consumption, drain off the water, and dry the coal sufficiently to permit its being fed mechanically. The plan has not progressed beyond an experimental stage.

In the different apparatus for burning, the fuel-dust is either drawn (Wegener, Westlake,[1] Ruhl, Pinther, Schwartz-Zkopf,[2] etc.) or blown (Friedeberg, De Camp, Engineering and Powdered Fuel Co., Cyclone,[3] Day, Freitag,[4] etc.) into the fire chamber. With the former there is no automatic regulation as is the

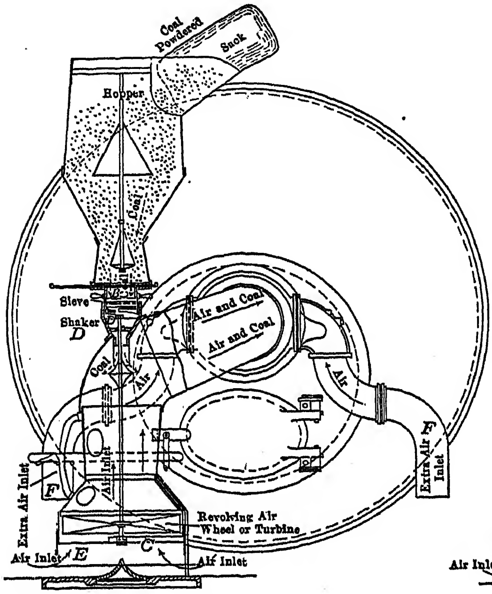


FIG. 89.

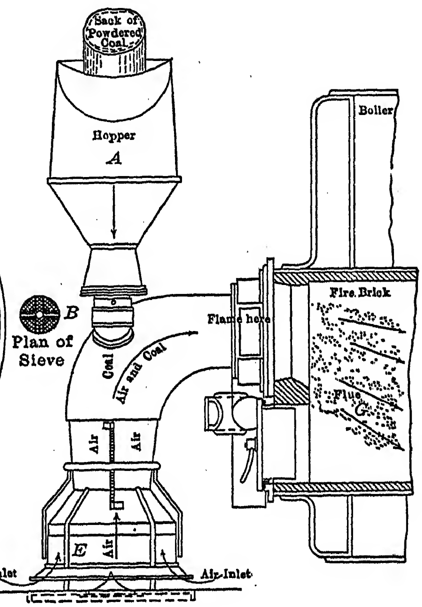


FIG. 90.

FIGS. 89 and 90.—Wegener apparatus for burning fuel dust.

case with the latter. Of the devices described in the references given, three may serve as types.

The Wegener Apparatus,[5] Figs. 89 and 90, consists of a hopper *A* receiving the powdered coal; the outlet is governed by a gate with pear-shaped valve. The rate of discharge is regulated by tapping the sieve *B* (5.5 in. diam.) from 150 to 250 times per min. by means of knockers or shakers *D* attached to the vertical shaft of the disk-fan *C* which is revolved from 50 to 80 times per min. by the inrushing air current *E*. The descending coal-dust meets in the vertical pipe

[1] *Eng. Record*, 1900, XLII, 615; *Eng. News*, 1901, XLV, 178.
[2] *Railway and Eng. Rev.*, Oct. 6, 1900; *J. West. Soc. Eng.*, March 2, 1904.
[3] *Eng. News*, 1901, XLVI, 415.
[4] *Zt. Verein deutsch. Ing.*, 1899, XLIII, 988; *Am. Mfr.*, 1899, LXV, 316.
[5] *Power*, 1904, XXIV, 168.

(20 in. diam.) the ascending air current and is carried into the brick-lined flue *G* where it kindles about 1 ft. from the front. On either side of the main pipe is an extra air inlet *F*. While tests under boilers have shown an efficiency of

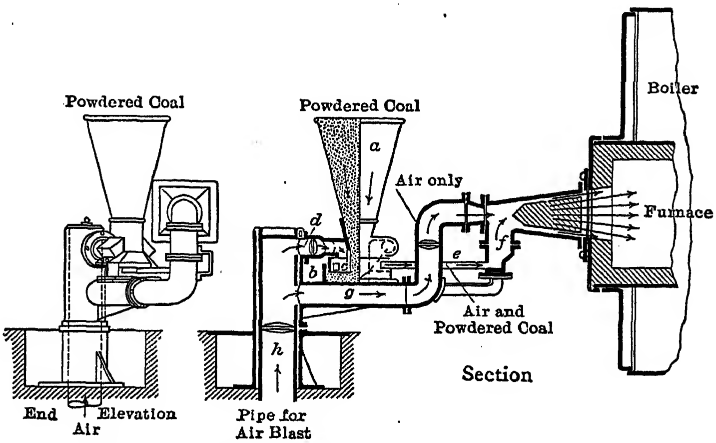


FIG. 91.—Friedeberg apparatus for burning fuel dust.

75 to 80 per cent., the main drawback found with the apparatus is that it will burn only 211 lb. coal per hr.

The Friedeberg Apparatus[1] is shown in part section and elevation in Fig. 91. The hopper *a* filled with coal-dust ends in box *b* the lid of which has 2 pockets *c*,

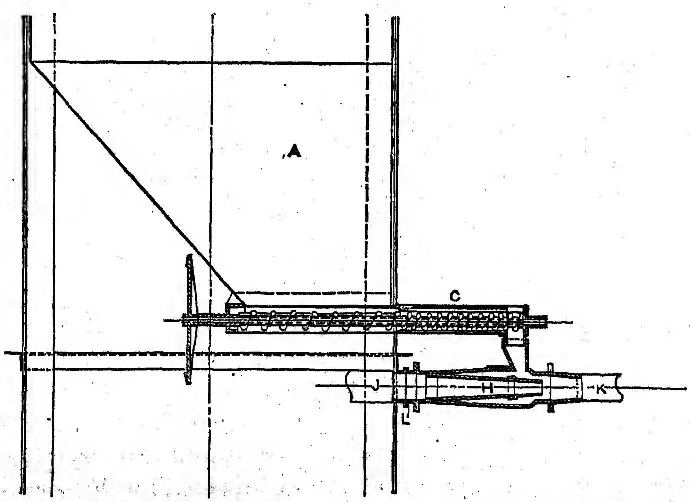


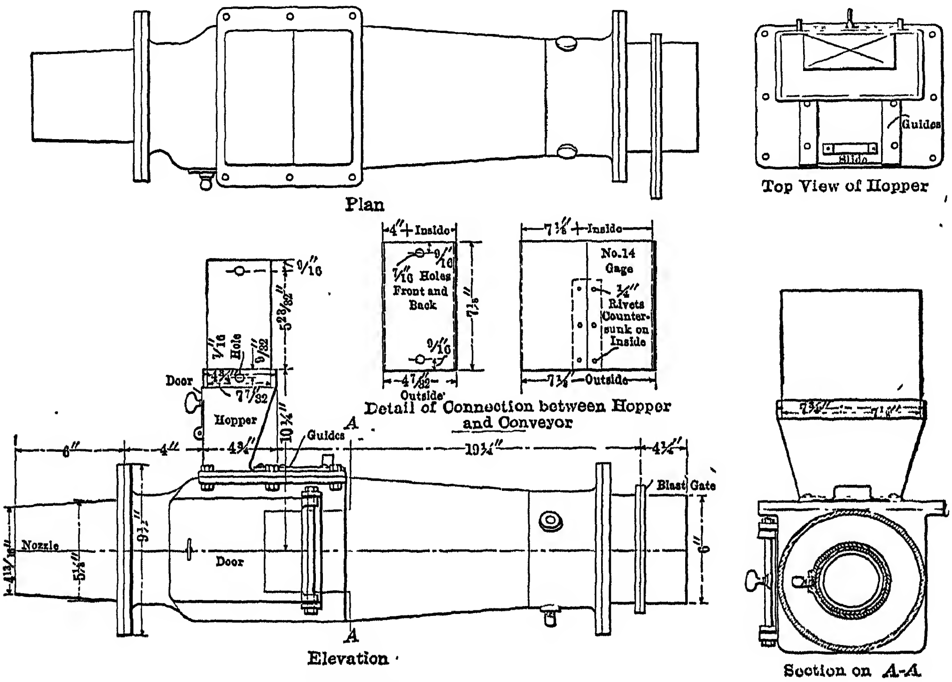
FIG. 92.—B. F. Sturtevant Co., apparatus for burning fuel dust.

each connected with a branch of blast-pipe *d*. The blast (pressure 6 in. water) stirs up the dust, carries it through pipe *e* into uptake *f* where it meets the air

[1] *Tr. Fed. Inst. Min. Eng.*, 1895–96, XI, 328.

from the second blast-pipe *g*, is mixed with it and delivered through a conical nozzle of fire-brick to the combustion chambers. As seen in the drawing, the air from the fan entering through pipe *h* is divided into the two branches *d* and *g*, the velocities of which are regulated by dampers. Boiler efficiencies of from 66 to 77 per cent. have been obtained in trial tests. The apparatus has been used for heating crucible furnaces when a temperature of 1,450° C. has been attained. It is absolutely necessary that the percentage of moisture in the dust be low and its distribution uniform.

Injector Apparatus: Most, if not all, of the fuel-dust burners used in the U. S. are of this type as, *e.g.*, the Culliney,[1] used by the American Iron and Steel



FIGS. 93 to 97.—B. F. Sturtevant nozzle for burning fuel dust.

Mfg. Co., Lebanon, Pa., in puddling and heating furnaces, the Ellis[2] and others. The apparatus of the B. F. Sturtevant Co., Hyde Park, Mass., may serve as example. Fig. 92 gives a vertical longitudinal section which shows the general arrangement. The coal-dust in hopper *A* is discharged by screw-feed *C* driven by a variable-speed gear, and drops into the cast-iron delivery-pipe *K* where it meets the contracted blast-pipe *H* which is connected by the sheet-iron pipe *J* with the blast main; *L* is the blast-gate. Details of the Sturtevant cast-iron nozzle used at the reverberatory matting furnaces of the Canadian Copper Co., Copper Cliff, Ont., are shown in Figs. 93 to 97. The main differences between

[1] U. S. Patent No. 784307, March 7, 1905.
[2] U. S. Patent No. 844857, Feb. 19, 1907.

this arrangement and the one shown in Fig. 92 are that the blast-pipe does not protrude beyond the hopper outlet, and that there is a door in the side of the nozzle near the outlet to give access in case of clogging or some other disturbance. The distance between hopper outlet and furnace is 8 ft. A cast-iron pipe of suitable length is slipped over the nozzle.

108. Compressed Fuels in General.[1]—Compressed fuel, patent fuel or fuel briquette, is a general term for any finely divided combustible that has been compacted to the form of a solid block. The manufacture of coal briquettes began in France in 1842; it then spread into England (1846), Belgium (1852), Austria (1858) and Germany (1860), and has been since then an established industry in the leading European coal centers. Briquetting brown coal began at a later date, about 1870. Germany leads Europe in this industry. It is estimated that about 3 per cent. of all the coal mined is briquetted. In the U. S. very little briquetting is carried on. This is accounted for by the low price of run-of-mine coal and the high cost of pitch. Semi-anthracite culm[2] was briquetted near Richmond, Va., anthracite screening at Milwaukee, Wis., semi-bituminous slack at Huntington, Ark.; briquetting plants are in operation in California, Michigan and perhaps other states. With the advent of by-product coking ovens the lack of pitch, the main cause for the early failures, is being removed.

In order to make a satisfactory coal briquette, it is desirable that the mixture contain 13–17 per cent. V.M.; if it runs higher, the briquette is likely to smoke when burning; if lower, to crumble. A mixture of lean and fat coals, of

[1] Gurlt, A., "Die Bearbeitung der Steinkohlenbriquettes," Vieweg, Brunswick, 1880.
Preissig, E., "Die Presskohlenindustrie," Craz and Gerlach, Freiberg, 1887.
Jünemann, F., "Die Brickett-Industrie," Hartleben, Vienna, 1903.
Dumble, E. T., "Brown Coal and Lignite," Geol. Survey of Texas, Austin, Tex., 1892.
Colquhoun, W., "Manufacture of Coal Briquettes, *Engl. Instit. Civ. Eng.*, 1894, CXVIII, 210; *Eng. and Min. J.*, 1895, LX, 347.
Björling, P. R., *Coll. Guard.*, 1897, LVII, 556, 601, 647, 695, 738, 784, 826, 872, 919, 962; *Min. Ind.*, 1897, VI, 177.
Bjorling, P. R., "Briquettes and Patent Fuel," Rebman, London, 1903.
Stohmann-Kerl, "Handbuch d. Technischen Chemie," Vieweg, Brunswick, 1893, IV, pp. 411 (Peat), 462 (Brown coal), 569 (Bituminous coal).
Fulton, J., "Coke," Scranton, Pa., 1905, pp. 406–466.
Prelim. Report, Coal Testing Plant, U. S. Geol. Surv., Louisiana Purchase Exposition, St. Louis, 1904, *Bulletin* No. 261. Final Report, *Professional Paper* No. 48.
Schorr, *Tr. A. I. M. E.*, 1905, XXXV, 82, 968.
Franke, G., "Handbuch der Brickettbereitung," Enke, Stuttgart, 1909, Vol. I, "Fuel."
Wright, *Bull.* 30, Bureau Mines, 1911; *Eng. Mag.*, 1910, XXXIX, 46; *Tr. Am. Ceram. Soc.*, 1910, XII, 69.
Blauvelt, *Tr. A. I. M. E.*, 1910, XLI, 255.
Holmes, *Bull.* 29, U. S. Geol. Surv., 1906, p. 48, bibliography.

[2] Wagner, *Cass. Mag.*, 1896–97, XI, 23.
Loiseau, *Tr. A. I. M. E.*, 1877–78, VI, 214; 1879–80, VIII, 314.
Atwater, *Eng. Min. J.*, 1899, LXVII, 264.
Irwin, *Iron Age*, June 19, 1902.
Dorrance, *Tr. A. I. M. E.*, 1911, XLII, 365.

anthracite or coke with brown coal or peat, will not make a briquette that is as good as the natural fuel. A coal briquette[1] should have the following four properties: (1) It should be hard, sonorous, homogeneous, odorless and non-hygroscopic, the content of H_2O should be < 5 per cent., and of ash < 10 per cent., the specific gravity < 1.15. (2) The weight should not exceed 22 lb., and the form be such as to make handling easy and permit breaking without making fines. (3) The breakage in handling and transportation should be < 5 per cent. (4) The briquette should kindle readily and burn with a bright and practically smokeless flame; its calorific power should be at least equal to that of the lump-coal from the screenings of which it was made, the usual pitch binder adding about 3 per cent. to the fuel value.

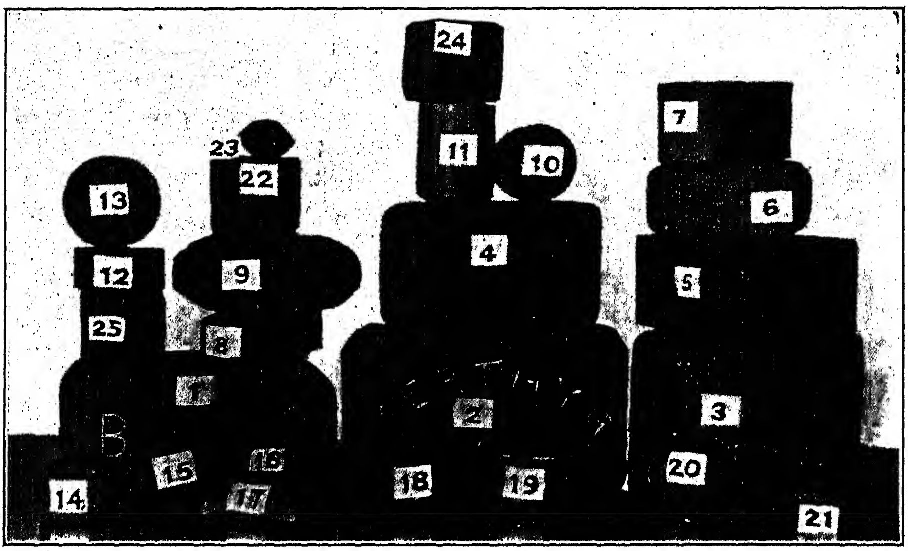


FIG. 98.—Forms of fuel briquettes.

The shape[2] of briquettes varies somewhat (Fig. 98). The prismatic form with rounded corners is common for sizes ranging from $5 \times 5 \times 2$ to $12 \times 8 \times 4$ in., which weigh up to 22 lb. The thickness never exceeds 5 in. Prismatic blocks can be packed closely; the long smooth surfaces of the large sizes, used under boilers of factories, steamers and locomotives do not kindle readily. The briquettes are therefore generally broken with a hammer before feeding into the fireplace. The breaking is facilitated by channels running crosswise. Ignition is also aided by longitudinal perforations. For small sizes the egg- (ovoid, eggette fuel) ball- (boloid fuel) and cylinder-forms are common; their weight ranges from 1/2 to 2 lb. Egg- and ball-shaped briquettes give little dust and waste in transportation, but occupy much space. Briquettes occupy, on the

[1] Schwackhöfer, "Technologie der Wärme und des Wassers," Faesy, Vienna, 1883, p. 69.
[2] *Eng. Mag.*, 1910, XXXIX, 46.

average, from 10 to 14 per cent. less space than the coal from which they are made.

Brown-coal briquettes conform to a large extent to the requirements made of coal briquettes; in addition they contain much less water than the raw air-dried brown coal, as they have to be dried artificially before briquetting. The same holds good for peat briquettes, except that here the difference in water, and often in ash, between raw material and briquette is much more marked.

With the exception of peat and certain earthy brown coals in the neighborhood of Cologne and Halle, Germany, all combustibles require a binder to make a satisfactory briquette; hence briquetting with bond is the prevailing custom.

The binder[1] in common use is pitch from tar and petroleum residues, and recently naphthalene.[2] The other organic binders: rosin, asphalt, molasses, starch (dextrin), albumens, etc., are of minor importance. Organic binders running low in ash and increasing the burning and calorific value of a briquette are to be preferred to inorganic (clay, alum, salt and sodium carbonate, magnesium chloride and oxide, lime, gypsum, water glass, metallic sulphates and sulphite liquor[3]) which not only have no fuel value, but increase the percentage of ash.

Pitch[4] is obtained by the distillation of tar. According to the temperature at which the distillation is carried on, three kinds are distinguished commercially.

Kind of pitch	Distilled at ° C.	Yield, pitch, per cent.	Sp. gr.	Volatile matter	Softens at ° C.	Melts at ° C.
Soft..........	<140	60–75	1.09	Light oils...........	40	60
Medium......	140–220	....	1.09–1.12	Light and heavy oils.	60	100
Hard.........	300–400	50–67	1.12–1.28	Anthracene.........	100	150–200

The pitch commonly used has a specific gravity of 1.19, softens at 70° C., and melts at 100–120° C. It remains hard at ordinary temperature, can be carried in bulk, and broken in a pitch cracker (gyratory crusher, toothed rolls, revolving beaters) to the requisite size of 0.25 in.; it is dull black, not greasy, has a conchoidal fracture; the edges do not splinter when bitten with the teeth; it hardens

[1] Steger, *Zt. Berg. Hütten. Sal. Wesen i. Pr.*, 1902, L, 311.
Muck, *op. cit.*, 1889, XXXVII, 370.
Schenck, *Oest. Zt. Berg. Hüttenw.*, 1890, XXXVIII, 465.
Binder, *op. cit.*, 1899, XLVII, 279 (testing apparatus for pitch).
Mills, *Bull.* 343, U. S. Geol. Survey, 1908.
Dünkelberg, *Stahl u. Eisen*, 1909, XXIX, 551.
[2] Grahn, *Glück Auf*, 1912, XLVIII, 1536, 1764.
[3] Schorr, *Eng. Min. J.*, 1909, LXXXVIII, 451.
Wright, *Bull.* 14, Bureau Mines, 1911, p. 52.
[4] Swoboda, "Der Asphalt," Voss, Hamburg, 1904.
Eldridge, "Asphalt," *Twenty-second Ann. Rep.*, U. S. Geol. Survey, 1901, Part I.

suddenly when dropped in the molten state into water of 15° C. Soft-pitch or tar-briquettes stick unless kept cool, smell and burn with a smoky flame. The following samples of pitch[1] gave the most satisfactory results in the tests made at the St. Louis exposition in 1904. The tests proved that in order to have the requisite binding power, a pitch must contain from 7.5 to 14 per cent. creosotes or adhesive oils. With > 14 per cent. volatile oils, the pitch was too soft; pitch with less than 7.5 per cent. volatile oils can have its binding power increased by adding creosote oils, *i.e.*, the V.M. driven off at and below 315° C. Hard pitch used in Europe as binder contains 75 to 80 per cent. C and 0.25 to 0.50 per cent. ash. The amount of pitch required for briquetting varies from 4 to 10 per cent.; 7 to 9 per cent. is a common figure. The size of particles ought not to be greater than 0.25 in. nor smaller than $\frac{8}{32}$ in. Coal rich in ash is, therefore, often washed to reduce the ash-content which should be < 6 per cent. Very fine material makes a smooth and hard briquette, but it requires a large amount of binder; furthermore the briquette is difficult to ignite and requires a strong draft to burn successfully. Beside the size of the material, the percentage of water it contains has an influence upon the quality of the briquette. Some water must be present; with coal it should not exceed 5 per cent., with brown coal it is more, and with peat still more.

TABLE 97.—PROXIMATE AND ULTIMATE ANALYSES OF PITCH

Sample	Proximate Analysis					Ultimate Analysis					Cal. power, cal.	Distillation			
	H_2O	V. M.	F. C.	Ash	S	H	C	N	O	S		First oil observed at deg. C.	Last oil observed at deg. C.	Oil at 400° C., per cent.	Residue
B......	1.14	49.66	47.88	1.32	0.70	3.97	90.89	1.05	2.07	0.70	8,752	351	......	7.38	Hard coke.
C......	0.88	62.75	35.84	0.53	......	4.72	91.16	1.16	1.85	0.58	8,937	......	......	......	......
D......	1.45	54.05	43.91	0.59	......	4.28	91.57	1.10	......	......	......	332	375	13.01	Hard coke.

The pressure employed in making a briquette has an influence upon the permissible percentage of water. Briquetting-material that is too wet has to be dried. Sometimes air-drying is sufficient; if not, it is supplemented by steam-dryers or special drying furnaces.

[1] *Bull.* No. 261, U. S. Geol. Survey, 1905, pp. 138 and 139.

Crushed and dried coal is mixed with the bond (pitch), heated in a vertical or horizontal pug-mill, and then passes into the compressing machine.

BRIQUETTING MACHINES.—Presses[1] are of two types: Open-mold presses used for peat and some kinds of brown coal, and closed-mold (resistance) presses used generally for the other fuels, occasionally for peat. Closed-mold presses are either tangential or plunger presses; with the latter the pressure can be applied either on one or on both sides of the briquette. The compressing force is usually steam, but in a few instances hydraulic pressure has been employed.

109. Briquetting Peat.—Raw peat contains about 85 per cent. H_2O, 13 per cent. peat-substance and 2 per cent. ash. In briquetting, the 15 per cent. combustible matter and ash to be compacted has to be freed as much from water

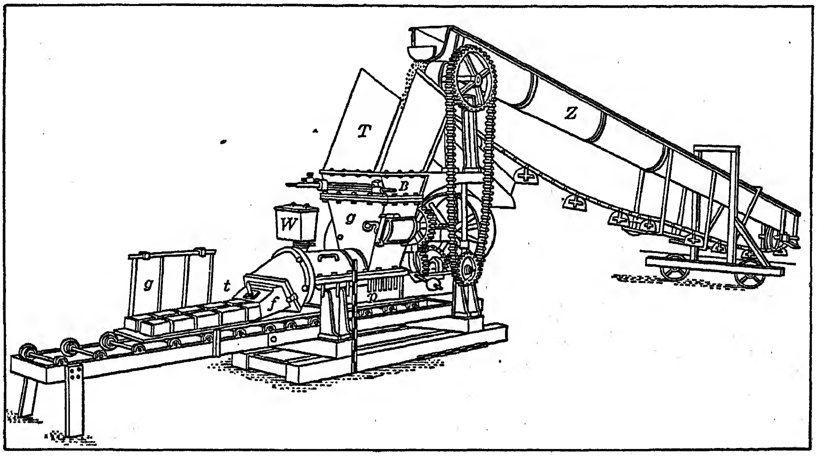


FIG. 99.—Schlickeysen peat briquetting machine.

and air as can be done economically. Thus raw peat, of which 1 cu. ft. with 85 per cent. H_2O weighs 63 to 66 lb., is converted without the use of a binder into air-dried machine-peat with about 20 per cent. H_2O, and into artificially-dried compressed peat with 12 to 16 per cent. H_2O, one cu. ft. weighing 40 to 60 lb.

Open-mold presses are in common use in Europe; the operation is called wet-pressing as the peat has to be wet. Closed-mold presses are in operation in Canada; the operation is called dry-pressing, as the permissible water is limited to about 16 per cent.

The leading open-mold peat-briquetting machine is that of Schlickeysen, Figs. 99–100.[2] It is a portable machine, Fig. 99, which is set up at the edge of a bog and moved from one part to another as the work of excavating progresses. It consists of three parts: The endless-belt conveyer Z which receives the raw peat, discharges it through chute T into the feed-box B, which delivers into the

[1] Steger, *Stahl u. Eisen*, 1903, XXIII, 1313, 1393.
De la Rocha, *Braunkohle*, 1904–05, III, 565, 577.

[2] Hausding, A., "Handbuch der Torfgewinnung." Parey, Berlin, 1904, p. 152.
Koller, Th., "Die Torfindustrie," Hartleben, Vienna, 1898, p. 48.

horizontal chamber *B*, Fig. 100; the propeller-blades of the latter force the pulp through the die *f*, Fig. 99, in the form of a plastic bar, about 3×4 in., on to boards 3 ft. long, the bar being cut into 3-ft. lengths. The charged boards are placed on trucks and run to the drying yard; the prisms are cut there into 10-in. lengths, placed on the drying floor where they air-dry in about two weeks. In the feed-box *B*, Fig. 100, is a hollow shaft *D* with beaters *E*, which makes 200 to 400 r.p.m. These beaters work between six sets of sharp propeller blades *F* keyed to shaft *C*; they cut, tear and feed the raw peat to the blades *F*. Some of the peat, however, would be carried around shaft *A* if it were not removed by the balanced scrapers *H* attached to the shaft *G*. In case the feeder becomes choked, the scrapers are automatically raised, as indicated by the dotted lines of lever and sliding counterweight *J*, and do not feed any peat until the chamber *B′* has been cleared. In case of necessity, the doors *K*, *K′* and *K″* can be opened and obstructing matter removed. Between the six sets of blades *F* there are placed sharp-edge oblong steel bars *P* acting as cutters; they can be exchanged without opening the housing. The cutting and tearing of the beaters, and the mixing, kneading and compressing of the blades sets free the natural binder "pentosane," contained in the peat-substance, which is to cement the particles. Its exact nature is not known, as so far it has not been isolated.

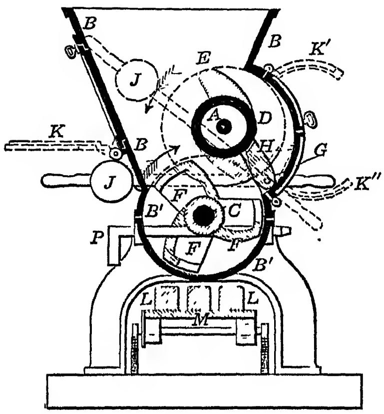


FIG. 100.—Schlickeysen peat briquetting machine. Section through feed-hopper.

The machine will compact 9 cu. ft. of raw peat with 80 to 85 per cent. H_2O into 6 cu. ft.; these when air-dry will have shrunk to 2.5 cu. ft., and retain 20 per cent. H_2O. Any further drying to bone-dry briquettes has to be done by artificial heating. This is rarely done unless the briquette is to be gasified or coked. A medium-sized machine with a screw-chamber 15.75 in. diam. requires 8 to 10 h.p., the feed-shaft makes 200 and the screw-shaft 80 r.p.m. The machine will treat 4,300 cu. ft. of raw peat and produce raw briquettes corresponding to 30 tons when air-dry. The machine weighs 3,850 lb. The cost of a plant complete with engine, tracks, cars, etc., ready to operate, is in Germany $4,500. The labor required, including digging and shoveling peat by hand, is 16 men and one boy, beside the engine- and fire-man. The total cost for labor per day in North Germany is $28.56.

The Dobson press,[1] in successful operation at Beaverton, Ont., may serve as an example of a closed-mold plunger-press. The 85 per cent. H_2O of the raw peat is reduced to 45 per cent. in about 2.5 hr. by spreading the material out in the open to a thickness of 1 to 1.5 in. Before the water-content can be reduced to from 16 to 18 per cent., the air-dried peat has to be disintegrated. This is

[1] *Report*, Bureau of Mines of Ontario, 1903, Toronto, pp. 212 and 222.

accomplished in a breaker which is a horizontal cylindrical sheet-iron box closed at the bottom by a 4-mesh screen. A horizontal shaft revolves in it at the rate of 400 r.p.m.; this shaft carries radial arms, 1 ft. long, connected near the ends by longitudinal rods on which are strung 4-in. steel fingers. The centrifugal force causes the fingers to fly out radially; they disintegrate and shred the peat and scrape it through the grating. The pulped material then goes to the Dobson dryer (Fig. 101).

This dryer is an inclined (14 in. in 30 ft.) cylinder (30 ft. long, 3 ft. diam., 3/8-in. iron) supported by radial cast-iron arms attached to two revolving (1.5 r.p. m.) shafts extending 12 ft. into either end. Between the supports are longitu-

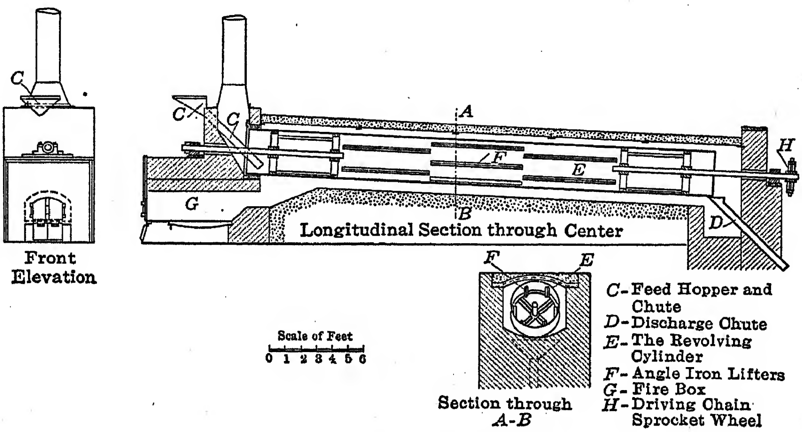
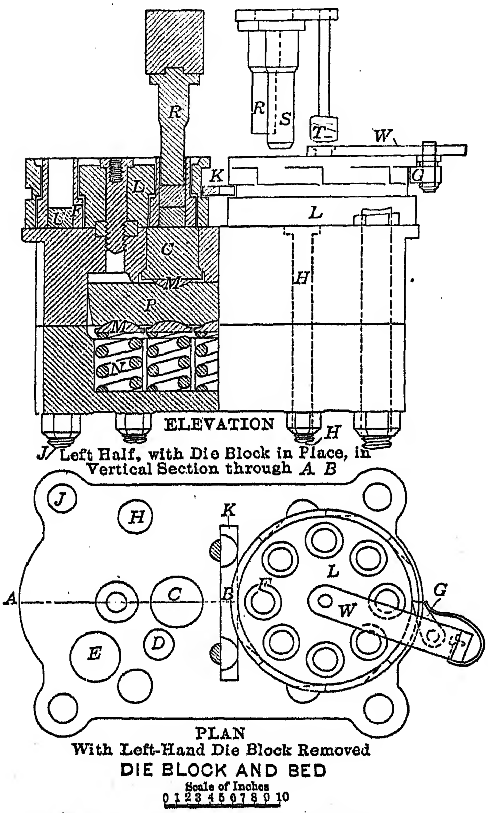


FIG. 101.—Dobson peat dryer.

dinal ribs (3×3-in. angles kept 3 in. from the shell by pins) which raise the peat and shower it through the ascending hot gases. The cylinder is enclosed by a rectangular brick casing. The fireplace is at the upper end of the cylinder; the flame and gases pass downward around the cylinder, enter this at the lower end, ascend in it, and pass off through a sheet-iron stack at the head. The air-dried peat travels through the cylinder in 20 min., and has its water content reduced to 16 per cent. The capacity of the dryer is 12.5 tons in 10 hr., the fuel consumption about 1.5 tons of air-dried peat. The disintegrator and dryer with accompanying conveyor, elevator and exhaust fan require about 15 h.p. The furnace-dried peat is now compressed into cylindrical briquettes (2.25 in. diam. and 2 in. long) by the two-plunger Dobson press shown in Figs. 102 and 103. Each plunger is moved by an eccentric; it has 3 parts the compressing punch *R*, the expelling punch *S* and the oil-swab *T*. Its die-block *L* has 8 dies; a die holds 2 briquettes; the lower one formed in the first round is subjected to another compression by a second briquette being formed on top of it. With every upward movement of the plunger, the die block is turned 1/8 of a revolution by the reciprocating lever and the ratchet *G*, and an empty die filled with peat. In the downward movement, the expelling punch *S* will drive out the bottom or completed briquette and push downward the top one that has been

compressed only once; the compressing punch R will form a new briquette on top of the one previously made in the same die, and the oil-swab T will oil an empty die to diminish the friction. The pressure exerted is 12.5 tons per sq. in.; the machine makes 50 r.p.m. and produces 100 briquettes; the output in 10 hr. is 12.5 tons (25 briquettes = 10 lb.). To operate the press with the accessory shafting, conveyors, etc., requires 13 h.p.



C-Resistance Block
D-Briquette Discharge Hole
E-Hole for Removal of Dies; ordinarily Plug
F-Dies
G-Ratchet
H-Four Clamping Bolts for the Bed
J-Four Tie Rods supporting the whole
K-Hole-down Bar for Die Blocks
L-Die Blocks
M-Spherical Buttons
N-Springs
P-One Bar between Resistance Block and
R-Punch
S-Expelling Punch
T-Oil Swab
U-Peat Briquettes
W-Reciprocating Lever

FIGS. 102 and 103.—Two-plunger Dobson press.

Shorr[1] constructed an experimental hydraulic plunger press with hand pump having a plunger 5 in. diam. Two men can turn out 60 to 80 briquettes per hr. subjecting them to a pressure of 11,400 lb. per sq. in. With a double mold the capacity is 60 to 100 briquettes and the pressure 5,700 lb., which is amply sufficient for the work. Estimates of cost of plant and of briquetting have been given by Shorr.[2]

110. Briquetting Brown Coal.—The earthy brown coal mined in the neighborhood of Cologne and Halle, Germany,[3] is at present the only variety which is briquetted without bond. Common, brown, pitch and glance coals require a binder; as their treatment differs little from that of bituminous coal, they need not be discussed separately.[4]

In briquetting the above earthy brown coal,[5] two methods are employed: wet pressing and dry pressing.

WET PRESSING.—In wet pressing, the coal is reduced to a fine powder by crushers and rolls, mixed

[1] *Eng. and Min. J.*, 1905, LXXX, 969.
[2] *Tr. A. I. M. E.*, 1905, XXXV, 106.
[3] Schorr, *Eng. Min. J.*, 1908, LXXXV, 460; 1910, LXXXIX, 524.
[4] Wright, C. L., "Briquetting Tests of Lignite," Bureau Mines, *Bull.* 14, 1911.
[5] Details see Fischer, Stohmann-Kerl, Dumble, *op. cit.*
Richter, C., Horn, P., "Die Mechanische Aufbereitung der Braunkohle," Knapp, Halle, 1910, pp. 40–159.
Franke, *op. cit.*, pp. 289 to 631.

with water in a horizontal pug mill, 10 to 15 ft. long, and passed through an open-mold press of the auger type. The revolving screw kneads and carries forward the mixture received at the rear end and forces it out at the front through a steam-heated copper or bronze die. The issuing bar is cut into bricks 2.5×4.75×8 in. which shrink upon drying to 2.25×4.25×7.5 in. The air-dried briquettes retain up to 40 per cent. H_2O; they can bear little handling, and are used as a domestic fuel.

Dry Pressing.—In dry pressing, the coal containing 40 to 60 per cent. H_2O is crushed, rolled, screened through an 8-mesh sieve, dried in steam-heated dryers and briquetted while still warm (30 to 40° C.) in an Exeter press. The finished briquette should not contain < 16 or > 20 per cent. H_2O, not < 4 to 6 or > 13 to 14 per cent. V. M. (the higher the V. M., the lower the necessary H_2O). The shelf-dryer of Vogel and revolving dryer of Schulz[1] are in common

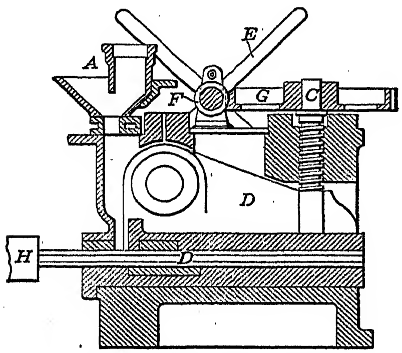


Fig. 104.—Exeter briquette press.

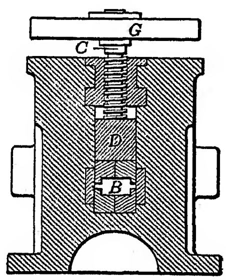


Fig. 105.—Exeter briquette press.

use, as they furnish a uniform product, create no dust, accompanied by danger of explosions;[2] are easy of access and have a large capacity. They have superseded the direct-fired and hot-air dryers. The Exeter press,[3] Figs. 104 and 105, is a horizontal, open-mold, plunger press. The plunger *H*, about 4 in. sq., traveling in a slightly converging mold *B*, has a 6-in. stroke and makes 65 to 80 strokes per min. The pressure developed is 2 to 2.5 tons per sq. in. and requires a 50 h.p. engine. The mold *B*, about 3 ft. 3 in. long, is composed of 4 sections of hardened steel which are held in place by screw *C* through retaining plate *D*. The screw is tightened to proper tension by hand-wheel *E*, worm *F* and worm-wheel *G*. The coal is fed through hopper *A*. With every back stroke of the plunger a certain amount of coal drops from the hopper into the mold and is compressed on the forward stroke against the 15 to 20 briquettes in the mold, one

[1] Freiberg, *Jahrb.*, 1898, Plate VI, p. 26.
Heckmann, Drying lignite briquettes, *Braunkohle*, 1912, XI, 574.

[2] Polster, *Braunkohle*, 1911, X, 597, 613.

[3] Modern multiple presses: Steger, *Stahl und Eisen*, 1903, XXIII, 1313, 1393.
Jordan, *Glück Auf*, 1910, XLVI, 46, 607, 1965, 2015.
Ehrhardt-Sehmer, *Braunkohle*, 1912, X., 713.

briquette being expelled at the open end. Of the total length of stroke of 6 in. 3.75 is passed through without compressing.

The high pressure and the resulting rise in temperature (to 80 to 100 C°.) liquefy bituminous matter (melting-point 70 to 80° C.) of the coal which acts as a binder. If the temperature should rise beyond 100° C., say to 130 or 135° C., vapors will be liberated which interfere with regular work; in order to prevent this, the mold may have to be water-cooled. The finished briquette upon leaving the mold at 50 to 60° C. expands 1/30 of its length and gives up some water. It is essential that the briquettes be piled up loosely, as otherwise there is danger of self-ignition.[1]

111. Briquetting Bituminous Coal and Anthracite.[2]—Bituminous coal and anthracite are briquetted almost wholly with a binder. There are some bituminous coals of decidedly coking character which can be and perhaps are briquetted without a binder. With anthracite culm the addition of a few per cent. of bituminous coal will assist the result considerably. The operations to be considered are: Crushing of coarse coal, drying of fine (washed) coal, crushing or melting of pitch, mixing of coal and pitch in correct proportions, compressing the mixture.

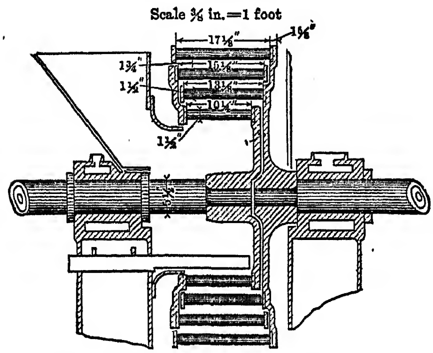


FIG. 106.—Stedman disintegrator.

The leading machine for crushing coal is the Carr, known in the United States as the Stedman Disintegrator.[3] The vertical cross-section given in Fig. 106, shows it to be composed of two pairs of oppositely revolving cages of round bars, each pair carrying a disk mounted on the end of a revolving shaft. The coal with <6 per cent. H_2O is charged into the hopper and fed at the center. Upon entering, it is struck and partly broken by the central cage; the pieces pass outward and are struck and broken by the second cage revolving in the opposite direction, and so on by the third and fourth. Passing out from the last, the comminuted coal is collected in a housing, not shown, and discharged. A machine 40 in. in diam. will crush 175 to 200 tons of coal in 10 hr., a 60-in.

[1] Heinrichsen-Taczak, *Glück Auf*, 1911, XLVII, 1640, 1681.

[2] Franke, *op. cit.*, pp. 1–287.
Schorr, *Tr. A. I. M. E.*, 1905, XXXV, 82, 968.
Parker, *op. cit.*, 1907, XXXVIII, 581.
Galloway, *Proc. South Wales Inst. Min. Eng.*, 1909, XXVI, 389.
Wright, *Bull.* 385, U. S. Geol. Surv., 1909; *Eng. Mag.*, 1910, XXXIX, 195.
Dorrance, *Tr. A. I. M. E.*, 1911, XLII, 365 (anthracite).
Malcolmson, Hartshorne plant, Okla., *Mines and Minerals*, 1909, XXIX, 339.
Hughes, Devillers plant, *Iron Age*, 1910, LXXXV, 432.
Johnson & Son, Leeds, *Coll. Guard.*, 1910, XCIX, 570.
Blauvelt, plant at Detroit, *Tr. A. I. M. E.*, 1910, XLI, 255.

[3] Richards, R. H., "Ore-dressing," New York, 1903, Vol. 1, pp. 284, 285, 298; see also Figs. 430 and 431.

machine 200 tons; 1 h.p. is required for every 4 to 5 tons treated in 10 hr.; the fineness of the product is regulated by the speed of the machine.

Washed coal, which has been freed from a large part of its water by drainage, is further dried by heating in reverberatory furnaces.[1] Centrifugal machines, steam- and hot-air dryers have fallen into disuse, as they are too expensive. With coking coal, the reverberatory furnace has the further advantage that the temperature can be raised to the softening point and the necessary amount of pitch correspondingly reduced.

The Biétrix drying furnace,[2] Figs. 107–109, has found much favor in Europe. It is a circular mechanical reverberatory furnace with hearth *A* revolving in a horizontal plane. From the lateral fireplace *B* the flame travels over the hearth, drops through two ports, returns underneath it, and passes off through port *C* into the flue leading to the stack. The hearth *A*, made of cast-iron, is carried

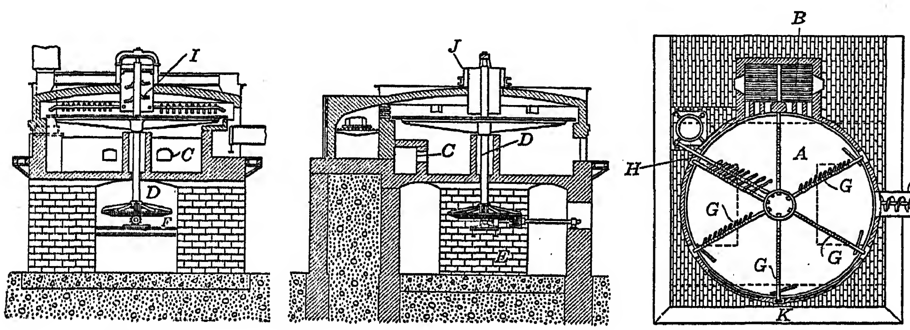


FIG. 107. FIG. 108. FIG. 109.

FIGS. 107 to 109.—Biétrix drying furnace.

by the vertical shaft *D*. This is supported by an adjustable stop-box *E*, held at the top by guides, and rotated from below by means of a bevel gear *F*. In the side of the furnace are six doors. Through four of them pass stationary arms *G* with teeth pointing downward; through the 5-in. door *H* is placed the rabbling device consisting of two arms, of which one is fixed and has a number of adjustable horizontal blades, while the other is movable and serves to change the angle the blades make with the arm. The coal fed by blades *I* in the cylindrical hopper *J* is spread over the hearth by the scrapers and turned over by the teeth. It is discharged through the 6-in. door *K* in the side-wall into a conveyor trough which delivers it to the pug mill and the press. The temperature in the furnace rises to 200° C., and the dried and perhaps softened coal leaves it at 80 to 95° C. A furnace with a hearth 9.18 ft. diam., making 6 r.p.m., treats 35 tons of coal in 10 hr.; a hearth 13.12 ft. with 4 to 5 r.p.m. 60 tons; and one 18.37 ft. with 3.5 to 4 r.p.m. 100 tons. The fuel consumption is 40 lb. of coal per ton of dried coal; 1 man tends a furnace; the cost of a furnace in France is

[1] Jordan, *Glück Auf.*, 1910, XLVI, 40, 607, 1965, 2015.

[2] *Mines and Minerals*, 1904–05, XXV, 163.

$3,600.00 and over. A modification of the furnace has been made by Heim.[1] Pitch is sometimes added to the coal in the liquid state (*wet-pitch process*); ordinarily it is incorporated in the solid state (*dry-pitch process*).

In the *wet-pitch process*, the hard pitch is dissolved in 15 to 30 per cent. tar (freed from light oils) in a horizontal coal-fired boiler-iron trough in which revolves a shaft with pugging arms. A trough is 20 to 23 ft. long, 3.3 to 6.6 ft. wide, 6.6 to 7.2 ft. deep, holds 9 to 18 tons of pitch which is liquefied in 12 to 20 hr. with a consumption of 3 per cent. coal on the weight of the charge. A Couffinhal press (see below) producing 70 tons briquettes in 10 hr. requires two

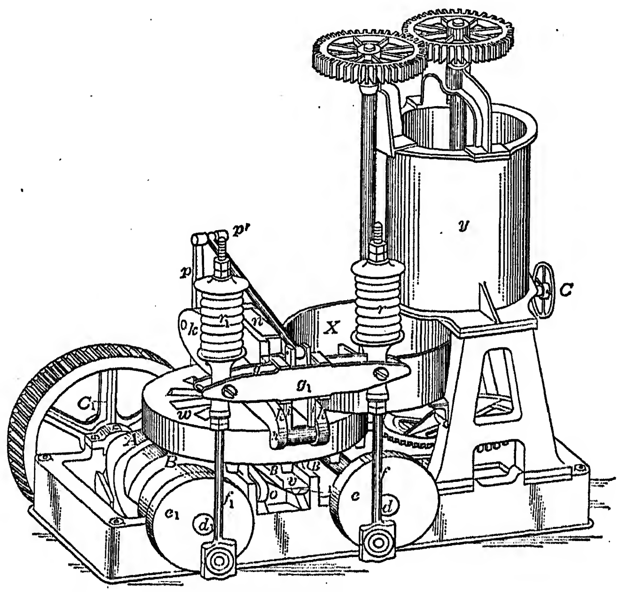


FIG. 110.—Couffinhal press.

troughs each of 9 tons capacity, one to be discharged while the other is being heated.

In the *dry-pitch process* the pitch, broken to nut size by corrugated or diamond-pointed rolls, is usually added to the coal when this is being fed into the disintegrator, thus assuring uniformity in size and completeness of mixture. Screw conveyors or bucket elevators, run at regulated speeds, furnish the desired proportions of pitch and coal. In drying fine coal from a washery, the pitch is often charged with the coal into the furnace or into the trough through which the dried coal leaves the furnace. The mixture of coal and pitch goes to a steam-heated pug mill. Vertical pug mills are more common than horizontal ones. A vertical mill is 2.5 to 4.0 ft. diam. and 6 to 10 ft. high; its shaft makes 20 to 25 r.p.m. The cylinder is heated by means of a jacket (dry-steam system), or the steam, preferably superheated, is injected in the paste (wet-steam

[1] Fischer, *op. cit.*, Vol. II, p. 47.

system); often both systems are in operation together. The paste when delivered to the press has a temperature of 70 to 90° C., and contains 3 to 5 per cent. H_2O and 6 to 10 per cent. pitch.

The pressure exerted by a press is 1,100 to 1,400 lb. per sq. in. of briquette surface; in exceptional cases it reaches double this amount. The power required ranges from 4 to 8 h.p. per ton briquette produced per hour.

Of the different presses, those of Couffinhal (Middleton, Yealdon) and Fouquemberg (Chisholm, Boyd and White) may serve as examples. The Couffinhal Press,[1] shown in Figs. 110 and 111[2] is a double compression machine, *i.e.*, the briquettes are pressed from top and bottom. The engine shaft *a* drives by means of a pinion wheel *b* the spurwheels *cc'*. At the opposite ends of the two spurwheel-shafts *dd'* are the crankwheels *ee'* tied by the connecting rods *ff'* to the crosshead *gg'* which communicates its motion to the upper double-cheeked lever *kk'* by means of the connecting rods *ii'* and the pins *hh'*. To the lever are attached the upper compression piston *l* and the ejecting piston *m* which are guided in their vertical motion by the fixed center *n*. A second double-cheeked lever *oo'*, below the molding table *w*, pivoted at *e*, carries the lower compression piston *q*. In the downward motion of piston *l* a moment arrives when the limit of compression of the upper surface of the briquette is reached through the resistance of the lower piston *q* and the friction between the paste and the walls of the mold. A reaction sets in, the fulcrum of lever *kk'* is changed from its original position on the left of piston *l* to the surface of the briquette. The continued downward motion of the upper lever near *m*, raises the lower lever *o'* by means of the rods *pp'*, and thereby the piston *q* which compresses the lower part of the briquette. Thus, during the compression of the upper surface of the briquette, the fixed point was near *t*, while during the compression of the lower portion it was at *e*. Rubber cushions *rr'* on rods *ff'* regulate the pressure. The lift of piston *q* is limited by the cast-iron cylinder *s* (attached to the pin connecting cheeks *o'*) striking a plate screwed to the rod *t* which is held in the frame of the machine. When the upper lever rises, the spring *u* brings the lower lever

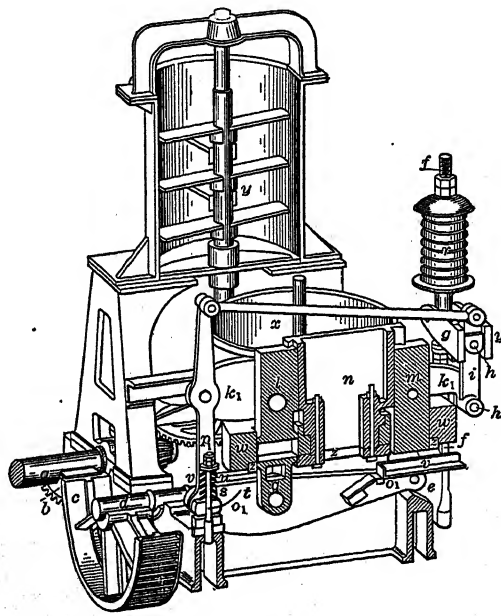


FIG. 111.—Couffinhal press.

[1] *Dingler Pol. J.*, 1883, CCXLVII, 159; 1884, CCLIV, 244.

[2] Beckert, "Leitfaden zur Eisenhüttenkunde," Springer, Berlin, 1898, Vol. I, pp. 49 and 50.

back to its original position. While the compression piston *l* molds a briquette, the ejecting piston *m* forces out a finished briquette on to the rocking table *v* which tips it on to an endless belt. The paste is brought by a screw conveyor to the pug mill *y* to be discharged from the bottom (or the side) in to the distributing pan *x*, the arms of which force it successively into the 12 molds of the table *w* as they pass underneath the pan. The rate of discharge is regulated by means of the hand-wheel *C*. The shaft of cylinder *y* is driven from the engine shaft *a* by a bevel-pinion and a gear-wheel; the shaft of the pan *x* is geared at the top with the shaft of cylinder *y*. The table *w* is steadied by the base-plate *z* and the center-piece *n*; it is rotated by the bottom pins *B* which engage in grooves cut in the cylinder *A*, keyed to shaft *dd'*. The grooves resemble those of a worm-gear;

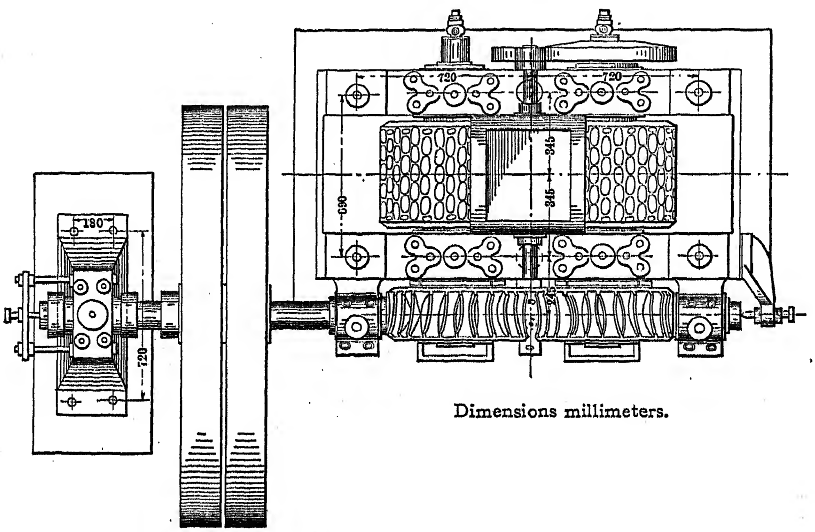


FIG. 112.—Fouquemberg press.

they are, however, cut in such a manner that the table comes to a standstill and is held firmly by 3 pins when the molds have arrived under the pistons. It then travels on at an accelerating, followed by a retarding, speed until it again comes to a stop.

The machines on the market are of various sizes. They make briquettes weighing 2.2, 6.6, 11 and 22 lb., and have accordingly a capacity of 20, 50, 80 and 160 tons of briquettes in 10 hr., consuming 18, 40, 60 and 90 h.p. The cost,[1] in Germany, of a plant having a daily capacity of 100 tons of 5-lb. briquettes is $13,000; of 200 tons 10-lb. briquettes, $19,600; of 300 tons 20-lb. briquettes, $25,300. According to J. Bordollo, Kingsbridge, New York City, a machine producing in 22 hr. 50 tons of briquettes of 1 1/2 lb. or 150 tons of 7 lb., requires 45 h.p., weighs 70 tons, and costs, f.o.b., New York, $18,000.

[1] Dumble, *op. cit.*, p. 225.

The Duplex Couffinhal Machine of Blanzy,[1] France, compresses 2 briquettes weighing 15.4 lb. with each piston stroke. With 26 strokes per min. its capacity for 10 hr. is 218 tons. A picture[2] of a Couffinhal plant is given in some of the references on briquetting.

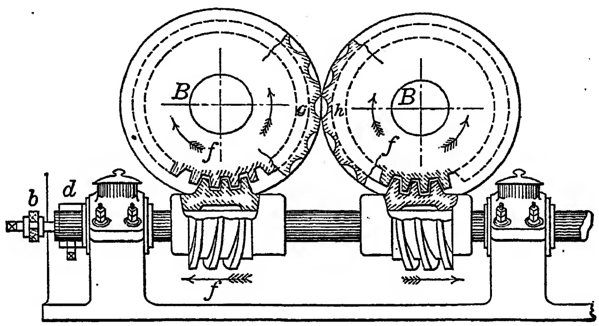


FIG. 113.—Fouquemberg press.

The Middleton[3] and Yealdon presses,[4] both of English make, are also double compression machines, but their compression table is upright and revolving upon a horizontal axis, while two levers, one on a side, compress the paste

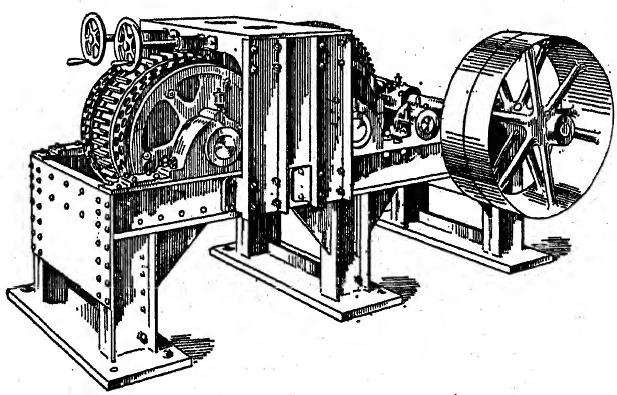


FIG. 114.—Chisholm, Boyd and White eggette press.

to a briquette. The briquettes are uniform in size, but are unequally compressed, as the effective force exerted depends upon the amount of paste fed, and this is likely to be uneven. Other makes[5] are given in the foot-note.

[1] *Compt. rend. Soc. Ind. Min.*, 1896, p. 107, 3 drawings.

[2] *Cass. Mag.*, 1896–97, XI, 26.

[3] *Eng. Mag.*, 1910, XXXIX, 200.

[4] *Freiberg. Jahrb.*, 1907, p. 35.

[5] *Iron Age*, 1909, LXXXIII, 291 (Hale); see also Wright (*Eng. Mag.*, 1910, XXXIX, 195), Parker (*Tr. A. I. M. E.*, 1907, XXXVIII, 581), Franke (*op. cit.*, pp. 139–191).

The Fouquemberg Press,[1] Figs. 112 and 113, also called tangential or circumferential or Belgian press, makes oval briquettes or eggettes. Two rolls containing on their surfaces oval cavities revolve in contact with one another in opposite directions; the cavities in one roll correspond exactly with those in the other so that closed molds are formed as the rolls revolve. The fuel mixture fed into the hopper on top is compressed in the cavities; the briquettes formed are discharged below on to a small screen which separating the waste (4–10 per cent.) delivers on to a traveling belt. With pasty material a pug mill takes the place of the hopper. The rolls are driven by worm-gears running in oil; they are steam heated if pasty material is to be agglomerated. Depending upon the length of the rolls, a machine will make per hour, 3, 7, 9 or 12 tons of eggettes, weighing 3 to 4 oz. each. In the Chisholm, Boyd and White machine (Eggette Press), Fig. 114, the rolls have spur-gears meshing with one another between the cupped faces; one of the spur-gears is driven by a pinion-wheel keyed to the pulley-shaft. The machine occupies a floor-space 10×5 ft., weighs 10 tons and requires 30 h.p. to operate it; its capacity is 5 tons of briquettes per hr. weighing on the average 0.3 lb. each.

Other machines of this type are those of Bilan,[2] Loiseau,[3] Mashek,[4] New Jersey Briquetting Co.,[5] United Gas Improvement Co.,[5] Renfraw,[5] Zimmermann,[5] Devillers.[6] The cost of briquetting a ton of slack coal in the western part of the United States is given by Schorr[7] at $2.166:

Labor, inclusive of stacking	$0.16
Oil and grease	0.006
Sundry stores	0.01
Steam-fuel	0.04
Depreciation	0.05
	$0.266
8 per cent. of pitch at $12 per ton	0.96
1,840 lb. of coal-slack at $1 per ton	0.94
Total cost per ton of briquettes	$2.166

According to Blauvelt[8] the cost at the plant of the Solvay Process Co., and the Semet-Solvay Co. at Detroit, Mich., is

[1] *Rev. Un. Min.*, 1891, XVI, 161.
[2] *Dingler, Pol. J.*, 1882, CCXLV, 109.
[3] *Tr. A. I. M. E.*, 1877–78, VI, 216.
[4] *Iron Age*, 1906, LXXVII, 1330.
[5] Parker, *Tr. A. I. M. E.*, 1907, XXXVIII, 589.
[6] Wright, *Eng. Mag.*, 1910, XXXIX, 207.
[7] *Tr. A. I. M. E.*, 1905, XXXV, 101.
[8] *Tr. A. I. M. E.*, 1910, XLI, 255.

	Cost of one ton of briquettes	
	All coal	Equal parts of coal and breeze
0.455 ton coal, at $2	....	$0 91
0.925 ton coal, at $2	$1 85	....
0.455 ton breeze, at $1	... .	0 455
9 per cent. of pitch, at $8	....	0.72
7.5 per cent. of pitch, at $8	0 60	.. ..
Cost of briquetting, as above	0 40	0 60
Total	$2 85	$2.685

Mashek[1] gives the operating cost at the New Jersey Briquetting Co.'s 15-ton plant, Brooklyn, N. Y., exclusive of fixed charges and coal dust, as 92.5 cents per ton of briquettes, viz., coal-tar pitch 44 cents, labor 12 cents, fuel 20.5 cents, lubricating oil 1 cent, running repair 10 cents, re-briquetting dust from briquettes 5 cents. Barber[2] estimates the cost for western America to be $0.87 per ton briquettes exclusive of coal dust; labor exclusive of stacking $0.16, oil and grease $0.06, sundry stores $0.01, steam fuel $0.04, interest and depreciation $0.05, pitch $0.60 (8 per cent. mixture, pitch $7.50 per ton).

Briquetting sawdust[3] and charcoal[4] can be mentioned only by name. Briquetting of ores is discussed in §280.

112. Carbonized Fuels in General.—The object of carbonizing a fuel, *i.e.*, heating without or with limited access of air, is to separate the non-volatile from the volatile matter in order to obtain a fuel of high calorific value and strong reducing power, and one that will not alter its form while it is being burned. The non-volatile matter forming the charcoal or coke consists of fixed carbon, the ash-giving constituents of the natural fuel and a small amount of H, O and N; the volatile matter shows the greatest variety of composition; it can be collected in part as a liquid, in part as a gaseous by-product and utilized. Carbonizing is a heat-consuming process and therefore wasteful; it will be resorted to only when charcoal or coke, *i.e.*, heating by contact, is an absolute necessity. Where a flame will do the work, the natural fuel will always be used, barring a few minor exceptions.

[1] *Iron Age*, 1906, LXXV, 1332.

[2] *The Engineer*, Chicago, 1906, XLIII, 749.

[3] *Berg. Hüttenm. Z.*, 1896, LV, 143; *Thonind. Z.*, 1900, XXIV, 2042; *Jernkont. Ann. Bihang.*, 1900, 345.

[4] *Eng. Min. J.*, 1882, XXIII, 198; *Stahl u. Eisen*, 1894, XIV, 441; *Iron Age*, Jan. 30, 1902.

113. **Wood-charcoal.**[1]—Wood subjected to destructive distillation undergoes the changes shown in Table 98:

TABLE 98.—TEMPERATURES AND PRODUCTS IN THE DESTRUCTIVE DISTILLATION OF WOOD

Temperature of distillation, ° C.		Products
Fixed matter	Volatile matter	
....	100–150	Water.
150–230	....	Torrefied (browned wood), 65 per cent. C.
....	150–280	Pyroligneous acid: Watery acid products.
230–250	....	Red charcoal, 73 per cent. C.
....	280–350	Hydrocarbons, mainly gaseous.
350 and over	....	Black charcoal (400° C.–80 per cent. C., 1500° C.– 90 per cent. C.
....	350–430	Hydrocarbons, liquid and solid (tar).

PYROLIGNEOUS ACID is dark red-brown, has a peculiar empyreumatic odor and includes acetic acid ($C_2H_4O_2$) and wood naphtha. The latter contains beside methyl alcohol (CH_4O), some acetone (C_3H_6O), methyl acetate ($C_3H_6O_2$), and oily substances; it is worked for wood naphtha containing methyl alcohol and methyl acetate to be used in the arts, and for wood vinegar, the crude acetic acid. The oily substances are burnt as fuel.

The GASES obtained are a mixture of H_2, CO_2, CO, CH_4, C_2H_4, C_6H_6 (benzole) and other illuminating C_xH_y.

The TAR is composed of benzole, paraffine (C_nH_{2n+2}), naphthalene ($C_{10}O_8$), carbolic acid or phenole (C_6H_6O) which forms the leading constitutcn of the ordinary creosote, pyroligneous acid, other liquid, C_xH_y, and resin. It is used raw for painting wood on account of its antiseptic properties, or is distilled for acid water, light oils, heavy oils and pitch. Table 99 of Violette[2] shows the chemical changes black alder (alder buckthorn) underwent in destructive distillation.

TORREFIED WOOD is a by-product in the distillation of wood (especially of beech) for the production of methyl alcohol, acetone and creosote. It is usually burnt to heat the retorts.

RED CHARCOAL is produced in the preparation of charcoal for the manufac-

[1] Svedelius, G., Anderson, R. B., and Nicodemus, W. J. L., "Handbook for Charcoal Burners," Wiley, New York, 1875.

Dromart, E., "Traité de carbonisation en forrêts," Lacroix, Paris, 1880.

Denz, F., "Die Holzverkohlung und der Köhlereibetrieb," Perles, Vienna, 1910.

Klar, M., "Technologie der Holzverkohlung," Springer, Berlin, 1910.

Börnstein, *J. Gasbel. Wasserversorgung*, 1906, XLIX, 648.

[2] *Ann. chim. phys.*, 1851, XXXII, 322.

TABLE 99.—DECOMPOSITION OF BLACK ALDER IN DESTRUCTIVE DISTILLATION

Material	Heated to deg. C.	C, per cent.	H, per cent.	O, N, and loss, per cent.	Ash, per cent.
Dried wood	150	47.51	6.12	46 29	0 08
Torrefied wood	260	67.85	5 04	26 49	0 56
Red charcoal	280	72.64	4.70	22.10	0.57
Brown charcoal	320	73.57	4.83	21.09	0 52
Dull-black charcoal	340	75.20	4.41	19 96	0 48
Lustrous-black charcoal	432	81.64	1.96	15 25	1.16
Charcoal produced at extreme white heat.	1,500+	96.52	0.62	0 94	1.95

ture of black powder. According to Fernow,[1] 100 parts wood give 26 parts charcoal, 30 pyroligneous acid and water, 7 tar, 37 CO_2,CO, C_xH_y and H_2O vapor. The brands, *i.e.*, imperfectly charred wood, obtained in making charcoal, very much resemble red charcoal. They form an excellent fuel for the slow warming of a blast-furnace crucible.

ORDINARY CHARCOAL, when made in the right way, is black, lustrous, hard, sonorous, has a glossy conchoidal fracture, soils the fingers only a little, and burns in small pieces without flame or smoke. If the wood has been imperfectly charred, the color may be brownish, the coal dull and soft, crushing readily between the fingers and soiling them; it will often burn with a flame. Charcoal retains the structure of the wood from which it has been made, it occupies a smaller volume and often shows radial cracks. It is very porous, as it consists of a large number of small elongated cells separated from one another by porous walls. This is clearly brought out by the photomicrograph of Thörner, Fig. 115, which is a cross-section through spruce charcoal having in 100 g., 200 c.c. cell-spaces and 61 c.c. cell-walls. Juon[2] found in charcoal from birch 74.82 per cent. cell-spaces, from pine 79.42 per cent., and from spruce 83.45 per cent.

The porosity of charcoal causes it to be readily permeable by gases. According to the celebrated researches of Saussure[3] in 1812, 1 vol. boxwood charcoal absorbs at 11 to 13° C., 90 vol. NH_3, 85 HCl, 65 SO_2, 55 H_2S, 40 NO, 35 C_2H_4, 35 CO_2, 9 CO, 7.5 N, 5 CH_4, 1.75 H, *i.e.*, large volumes of gases that are readily liquefied. Later data are those of Smith,[4] Hunter[5] and Hempel-Vatu.[6] Charcoal condenses offensive vapors, decolorizes solutions, etc.

It is generally held that charcoal will absorb 6 to 7 per cent. H_2O; this figure,

[1] *Tr. A. I. M. E.*, 1879, VI, 200.
[2] *Stahl u. Eisen*, 1904, XXIV, 1233.
[3] Gilbert, *Ann. Philos.*, 1814, LVII, 113.
Thomson, *Ann. Philos.*, 1815, VI, 241, 331.
[4] *Proc, Roy. Soc.*, 1862–63, XII, 424.
[5] *J. Chem. Soc.*, 1865, XVIII, 285; 1867, XX, 160; 1868, XXI 186; 1870, XXIII, 73.
[6] *Zt. Electrochem.*, 1912, XVIII, 724.

however, is not fixed. It has been shown[1] that it will absorb in the first 24 hr. after having been drawn, 4 to 8 per cent. H_2O, and in a few days 12 per cent., which appears to be the limit. The absorbing power seems to decrease as the charring temperature rises. The apparent specific gravity (charcoal + air-spaces) for hard-wood charcoal is 0.35 to 0.50, for soft-wood charcoal 0.20 to 0.40; the actual (true) specific gravity of charcoal powder free from air is 1.5 to

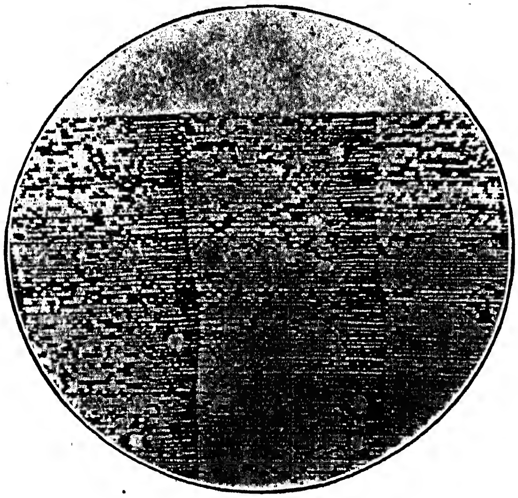


FIG. 115.—Photomicrograph of spruce charcoal. (Thörner).

2.0. One cu. ft. of charcoal from hard wood weighs[2] 14 lb., from pine wood 8 lb.; the commonly accepted range is 12 to 13 and 8 to 12 lb. Charcoal is usually measured by the bushel. This varies[3] from 2,250 cu. in. in Maryland to 2,844 cu. in. in Oregon. It is accepted now that a bushel shall be equal to 2,748 cu. in. and that the average weight of a bushel of charcoal is 20 lb.[4] Charcoal is a poor conductor of electricity.

In storing, 5 per cent. is lost by crumbling and oxidation. The ignition-point rises with the temperature the charcoal has been subjected to; thus Violette[5] gives the data assembled in Table 100.

The average ultimate composition[6] of charcoal according to Bunte is given in Table 101. The calorific power is 6,000–7,000 cal.

TABLE 100.—RELATION OF CHARRING AND IGNITION TEMPERATURES

Charring temperature, deg. C.	260–280	290–432	1,000–1,500	1,760
Ignition temperature, deg. C.	340–360	360–370	600–800	1,250

[1] *Foundry*, 1896–97, VIII, 143.
[2] *J. U. S. Assoc. Charc. Iron Workers*, 1882, III, 286–287.
[3] *Foundry*, 1900, XVI, 140.
[4] *Trans. A. I. M. E.*, 1880, VIII, 384.
[5] *Ann. chim. phys.*, 1853, XXXIX, 323.
[6] Analyses of Charcoal, *J. U. S. Assoc. Charc. Iron Workers*, 1884, V, 33, 194.

TABLE 101.—ULTIMATE COMPOSITION OF CHARCOAL

Condition	C	H	O	Ash	H_2O
Air-dry....	84.5	2.5	4 3	1 2[1]	7.5
Water-free..........	91.3	2.7	4.8	1.2[1]	....

114. **Charring in General.**—The woods commonly used for making charcoal are the evergreens: spruce, pine, fir, larch, and the deciduous trees, oak, beech, ash, elm, birch. The yield depends much upon the kind of tree. Thus Violette[2] charring at 300° C. 72 varieties of wood dried at 150° C. found the percentage by weight to range between 30.86 (horse-chestnut) and 62.80 per cent. (ebony), and Bull[3] in testing woods ordinarily used in charring obtained from dry wood 19 (white birch, American hornbeam) to 25.74 (white ash) per cent. by weight of charcoal. Trees of medium age are preferred to young and old trees, as they give a denser and harder charcoal. According to Scheerer[4] the best charcoal is obtained from red and white beech when 120 years old, from pine at 80 to 100, spruce at 70 to 80, fir at 60, larch at 50, oak at 50 to 60, elm at 20 to 30, alder at 18 to 20, birch at 20. Stem-wood half air-dried (cut in winter, stacked, air-dried in summer) and charred in the fall gives the densest charcoal; damp wood is difficult to kindle; dry wood makes much waste; rotten wood unsound charcoal. The wood is usually cut into 4-ft. lengths and split if over 6 in. diam. The yield depends not only upon the kind and quality of wood, but also upon the time given to charring. The 21 tests of Karsten[5] on different kinds of air-dried wood gave the facts shown in Table 102.

TABLE 102.—EFFECT OF TIME OF CHARRING UPON THE YIELD OF CHARCOAL

Charring	Charcoal, per cent.		
	Minimum	Maximum	Average
Quickly........................	12.2 (old birch)	16.225 (young spruce)	14.4
Slowly........................	24.2 (lime)	27.7 (young spruce)	25.6

Quick charring at an elevated temperature is thus seen to decrease the yield

[1] The percentage of ash is nearer 3 than 1.
[2] Stohmann-Kerl, *op. cit.*, p. 394.
[3] *J. U. S. Assoc. Charcoal Workers*, 1882, III, 93.
[4] "Lehrbuch der Metallurgie," Vieweg, Brunswick, 1848, p. 212.
[5] "System der Metallurgie," Reimer, Berlin, 1831, Vol. III, p. 34.

of charcoal and to increase the amount of volatile matter; the reverse is the case with slow charring. Quick charring will be practised in chemical plants in which the volatile matter is the main product. The reason for this difference in yield is that volatile matter is still being evolved after some charcoal has been formed, and acting upon the glowing coal forms CO, CO_2 and H. With slow charring, at least most of the H_2O will have been driven off before the temperature has risen to the charring stage. Charcoal charred too quickly is light; if the temperature is too low, the charcoal will remain too rich in volatile matter.

Charring is carried in heaps, stalls and retorts.

115. Charring in Heaps[1] (Meilers, Mounds).—This, formerly the leading method of operating, is at present confined in the United States to the production of charcoal for domestic use. In charring by this method, cord wood is stacked to form a slightly conical pile and covered with sods, sand, breeze, etc., to protect it from direct contact with air. The pile is ignited at the center, the admission of air is regulated in such a manner by holes punched into the cover that the combustion shall be just sufficient to do the charring. The plan of work is to have, (1) the air entering the heap pass as much as possible through fresh to charring wood; (2) to remove the products of imperfect combustion in a manner to prevent their coming in contact with charred wood, and, (3) to char slowly in making furnace charcoal.

The *site* for a heap should be near the wood-supply; it must be well sheltered, have plenty of water, and permit easy transportation of wood and charcoal.

The *ground* for building must be firm, even, dry and slightly porous. Cold, moist, impermeable soil makes charring impossible; porous, sandy ground admits too much air and causes an excess of breeze. Suitable ground is cleared and covered with about 1 ft. of breeze tamped down firmly; if breeze is not available, leaves, straw or similar material takes its place. Unfavorable ground can be made suitable by raising it 9 to 12 in. with stones and sticks, and tamping down firmly sand and earth; if need be, it is drained by a shallow ditch. The center of the floor is usually made 6 in. higher than the rim. An old floor works better than a new one.

A heap is 16 to 26 ft. in diam., 10 to 16 ft. high and holds 14 to 28 cords of wood.

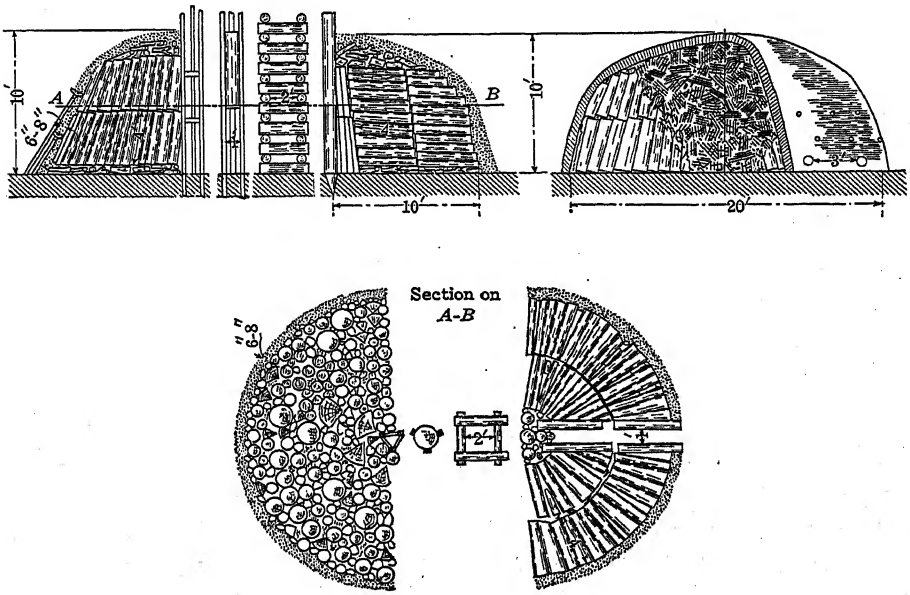
Erection of Heap.—In erecting a circular or standing heap, Figs. 116 to 120, three stakes, Fig. 116, are driven into the ground to form the corners of a triangular chimney; they are secured in their positions by cross pieces or by a central block. Sometimes a square chimney takes the place of the usual triangular form. On the bottom of the chimney are placed a few dry boards to form a temporary grate for the kindling.

In some heaps a single post is driven into the center of the bed to take the place of the chimney. If the heap is small, the post may be pulled after the wood has been stacked; with large heaps it has to be burnt out. For this purpose a radial channel is built along the floor, Fig. 119, by means of a log, reach-

[1] Bergström, *Jernkont. Ann.*, 1904, XLIX, 207.
Denz, F., "Die Holzverkohlung und der Köhlereibetrieb," Perles, Vienna, 1910.

ing from the center to the periphery, which is removed when the heap has been built. Through this channel the wood (kindling, brands) piled around the base of the post is readily ignited.

The cord wood is now stacked around the central chimney or post in one of two ways. With *vertical heaps*, Figs. 116 and 117, it is stacked upright with the edges facing the center and the thicker ends resting on the bottom; the inner sticks are placed vertically, the others with a slight inward slant. On top of the first row of sticks follows a second one similarly arranged.



FIGS. 116 to 120.—Standing circular heaps.

With *horizontal heaps*, Figs. 118 and 119, the sticks are stacked in horizontal radial layers, their thicker ends pointing outward; a few brands, or other material which is readily ignited, are placed vertically around the center to assist the kindling. In both cases, the heavier wood goes toward the center; it is packed closely to leave open as few spaces as possible, which are closed with small sticks. The top of a heap is covered with small wood placed horizontally and radially, and is thus rounded.[1] The heap is now ready to receive its cover.

The *cover* is made up of two layers. The inner layer, 4 to 6 in. thick, consists of sods beaten down with a shovel with the grassy side inward that the roots pointing outward may give support to the outer layer. Or, the heap is encircled with branches or marsh-hay and then covered with leaves, moss, etc. The outer layer is earth or sandy loam mixed with breeze spread to a thickness of 4 to 6 in., the thicker part being near the top. With breeze alone, the layer

[1] Illustration, *Foundry*, 1896, VIII, 141; *Pacific Coast Miner*, 1904, IX, 155.

may have to be 8 to 12 in. thick. The cover may reach to within 6 in. from the ground, Fig. 116 (the open ring to be closed later), or down to the ground, Fig. 120, in which case holes are punched into it near the base to furnish the air the necessary means of access.

In order to prevent the cover from slipping to the ground, crutches, Fig. 116, are put in place, or small stone pillars are erected.

Lighting.—The central chimney, Fig. 116, is filled with kindling, or the horizontal log, Fig. 119, is withdrawn to permit igniting the center-post, Fig. 118. A clear day is desirable for lighting, as on a dull or wet day the fire is likely not to start well. The heap is usually fired early, say 2 a. m., as the first stage of the process requires much attention. If the heap has a chimney, kindling is ignited, thrown into it, and the fire kept going by feeding small wood; if it has a center-post, kindling is fastened to a long stick, ignited and brought through the flue along the bottom to the base of the post which burns leaving a small chimney. In whichever way the heap has been lighted, the fire is kept going until a strong flame has been issuing forth from the top of the heap for 45 to 60 min., when the center is considered to have been satisfactorily kindled. The opening at the top is now securely covered with sods and earth, and the horizontal flue closed, if there was any.

Two stages are distinguished in the process, that of sweating and of charring.

Sweating Stage.—In the sweating stage, the heat generated by the burning of the wood in the chimney vaporizes the H_2O and begins to set free the V.H.-C. of the sticks close to it. The vapors will escape near the top, if openings have been left for them to pass off; or near the bottom, if the cover does not reach to the ground. In either case, fumes will pass off through the cover which becomes moist; it sweats. The V.H.-C. set free at first, may form explosive mixtures with the air of the open spaces in the heap and blow out parts of the cover. Such explosions may be expected, if the wood was too dry or in too small pieces, as this would cause the fire to progress too rapidly and the temperature to rise too quickly. If the wood, however, was half air-dried and of normal length, sufficient water-vapor will be evolved to dilute the gases and render the mixture harmless. Later on in the process, when most of the H_2O has been driven off, the amount of air allowed to enter is too small to make an explosive mixture; further, it enters at the bottom while the gases and vapors pass off at or near the top. The vapors at the beginning of the sweating stage are whitish, they then turn yellow, become dense and gradually change to a light-gray, when the sweating stage is over, having lasted from 18 to 24 hr. Now the hollow, "sink," in the center has to be filled, as the sucking-in of air at that point would interfere with charring. The cover is removed, the chimney filled with sticks, brands, breeze, the whole rammed down tightly, and the cover replaced and beaten or trodden down firmly. Any crevices in the cover are closed and any hollows in the heap, discovered by probing with a pole, are filled. If the cover did not extend to the ground, the open space is closed with sods, loam, breeze . . .

Charring Stage.—The cover having been repaired and made nearly imper-

vious to air, the heap is left to itself for from 2 to 4 days. Air enters mainly at the foot, rises through uncharred wood and near the center comes in contact with charring wood. It may be necessary to punch a few vent-holes to draw the fire to a part in which the charring did not proceed as rapidly as it did in others, the difference being recognized by the smaller shrinkage. The products of imperfect combustion pass off through the cover, and perhaps these vent-holes. The cover of the part that charred too quickly will have to be made thicker; it may be necessary to sprinkle it with water. Toward the fourth day the central portion of the heap will have been charred to the form of an inverted cone, Fig. 120. In order to draw the fire outward and downward, a horizontal row of holes about 1 in. in diameter is punched 4 to 6 ft. apart and about 5 ft. above the ground, *i.e.*, above the bottom layer of wood. First, white vapors will escape through the holes, then yellowish-gray smoke, and lastly colorless gas; gases becoming lead-colored, show that charcoal is being burnt by air entering through crevices in the cover. The temperature of the issuing volatile matter is 230 to 260° C. When colorless gases pass off through the first row of vent-holes, these are closed and a second row half-way down made; by means of a third row finally the fire is drawn to within 6 in. of the ground. In order to insure complete charring of the wood on the floor, this row is kept open until a flame issues as otherwise the wood would be only half-charred or simply browned. The dense vapors from a vent-hole gave Ebelmen[1] per liter 1 g. H_2O and tar, the light vapors 0.5 g.[2] The gases drawn from a vent-hole, with the vapors passing off abundantly, were whitish, had the composition *a*, Table 103; this changed to *b* when they were passing off sparingly, and were bluish to colorless. The values *a′* and *b′*, representing the percentage-composition with N omitted, show that CO_2 and CO diminish as the charring progresses, CO_2 more quickly

TABLE 103.—GAS FROM A HEAP OF CHARRING WOOD

Gas	*a*	*b*	*a′*	*b′*
CO_2	27.23	23 09	40.3	29.6
CO	7.67	6.04	23.6	21.1
H	11.64	14.11	36.1	49.3
N	53.40	55.77	....	....

than CO, and that H increases. As the heap shrinks and settles from 15 to 25 per cent., it is necessary at intervals to press down tightly the cover by stamping with the feet or by beating with a wooden hammer, to close up hollows that would otherwise form; occasional probing with a stick for hollows is necessary. The shrinkage of the heap combined with the hardening of the cover often causes breaks in the cover which have to be closed. If the cover becomes sufficiently

[1] *Ann. Min.*, 1843, III, 274.

[2] Recovery of By-products, see Percy, *op. cit.*, p. 372; also *Jernkont. Ann.*, 1907, LXII, 334.

hard and strong (shaly) so as not to follow the shrinking wood, it has to be broken into and then repaired.

Drawing of Charcoal.—When the charring is finished, the heap is covered with moistened loam to cool it and to smother the fire. In a day or two an opening is made into the cover at the base, logs of red-hot charcoal are drawn out with a heavy iron rake and quenched with water or buried in damp sand or breeze. When the air begins to act too energetically upon an exposed part, this is covered, and a new opening made. Drawing is thus continued until the heap has been pulled down.

Three grades of charcoal are obtained: (1) Well-burnt hard coal from the size of a billet to small pieces; (2) over-burned soft coal from near the chimney, and (3) under-burnt brands, especially from near the floor. Billets and middle-size are separated by forking (prongs 1 in. apart) from the fine coal; the former goes to market, the latter forms breeze, or brands which are to go into the next heap.

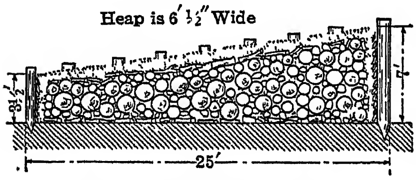


Fig. 121.—Rectangular or lying heap.

Results.—A heap 16 to 26 ft. diam., 10 to 16 ft. high, holding 14 to 28 cords of wood, burns 7 to 12 days. The yield by volume is 50 to 60 per cent. of the wood used, or[1] 1 cord gives 30 to 35 bushels;[2] by weight it is 22 to 23 per cent. The calorific value of the charcoal is 60 per cent. of that of the wood used.

Estimates of *costs* of charring in heaps range from 2.1 to 5.0 cents per bushel of charcoal.[3]

Rectangular or lying heaps, Fig. 121, are built in parts of Silesia, Austria and Sweden. They are oblong: the wood is piled as in cording. Such a heap, *e.g.*, will be 3.5 ft. high at the lower and 7 ft. at the upper end; it will be about 25 ft. long and 6.5 ft. wide. It is covered with sods and breeze as are standing heaps. Details are given by Karsten[4] and Percy.[5]

116. Charring in Stalls (Kilns).[6]—Replacing the movable cover of the heap by a permanent structure of masonry gives the closed stall used for charring wood. In Europe stalls are in operation mainly in Scandinavia[7] and the Ural Mountains;[8] in the United States they are used almost exclusively where much charcoal is produced, *e.g.*, in the smelting of charcoal iron, while heaps are of

[1] Egleston, *Tr. A. I. M. E.*, 1880, VIII, 374.

[2] One bushel equals 2748 cu. in.: *J. U. S. Assoc. Charcoal Iron Workers*, 1880–81, II, 128, 256

[3] *J. U. S. Assoc. Charcoal Iron Workers*, 1885, VI, 207.

[4] "System der Metallurgie," Vol. III, p. 71.

[5] "Fuel," p. 377.

[6] Egleston, *Tr. A. I. M. E.*, 1880, VIII, 373.
Jernkont. Ann., 1905, IX, 600; *op. cit., Bihang*, 1906, II, p. 61.

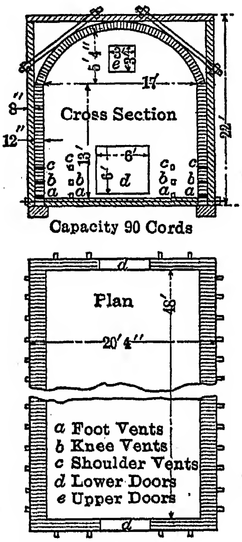
[7] *Jernkont. Ann.*, 1904, 1905.

[8] Gouvy, *Mém. Soc. Ing. Civ.*, 1901, I, 728.
Juon, *Stahl u. Eisen*, 1904, XXIV, 1230; Heck, *op. cit.*, 1906, XXVI, 193; 1907, XXVII, 733.

secondary importance. The latter have been discussed with more detail than their restricted use appears to call for because the different stages in charring can be made clearer with heaps than with stalls.

Stalls are rectangular, cylindrical or conical; volatile constituents either go to waste or are recovered as by-products; stalls are run intermittently or continuously. A stall has a stone foundation which often is laid in cement; the floor is a clay-mixture tamped down firmly; the walls are of red brick laid in lime, occasionally in clay mortar; they are whitewashed on the outside to make them air-tight and to aid in disclosing any leaks.

A RECTANGULAR STALL, Figs. 122 and 123, is a brick chamber 40 to 50 ft. long, 12 to 18 ft. wide and 12 to 18 ft. high to spring of arch which has a rise of about 5 ft. In the end-walls are two doors with air-holes, in the side-walls three rows of vent-holes, 12 in. apart vertically and 16 in. horizontally, to be closed by loose brick. The side-walls, 15 in. thick, are strengthened by buttresses and braced by buckstays and tie-rods. A stall holds from 55 to 90 cords of wood. The drawback of this stall is the difficulty of keeping the long side-walls air-tight. This is overcome to some extent by the circular stall.



FIGS. 122 and 123.—Rectangular stall.

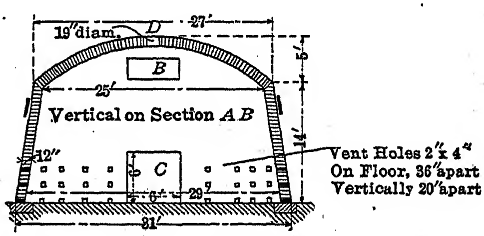
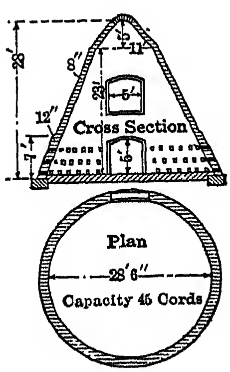


FIG. 124.—Circular stall.

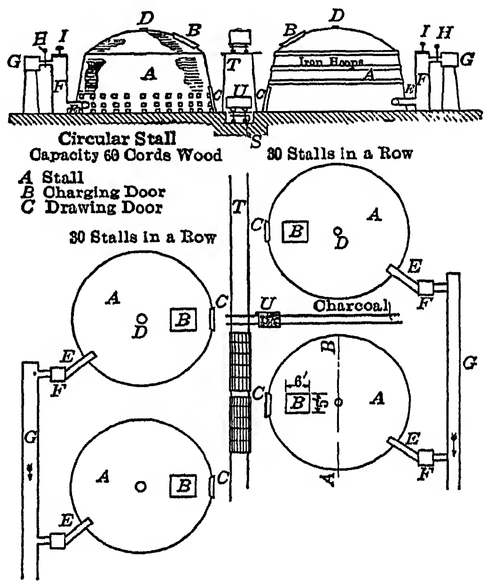
CIRCULAR STALL (Fig. 124).—This is 26 to 28 ft. diam., 12 ft. high to spring of arch which has a rise of 6.5 ft. The sides are braced by broad iron bands. The stall has a rectangular *C* door with air-holes on the ground level, a circular one in the roof, and three rows of vent-holes. It holds about 50 cords of wood.

CONICAL STALL.—This stall, Figs. 125 and 126, is 26 to 30 ft. diam. at the base and 20 to 28 ft. high. The walls have sufficient batter so as not to require any bracing, and remain air-tight longer than the preceding two forms. There are two doors, 5 ft. sq., built in iron frames; the one at the front on the floor has air-holes, the other generally at the back and near the top is solid; the 3 rows of vent-holes often are of cast-iron. The cord-wood is charged in horizontal layers around a central space serving as chimney; sometimes there is left open along the floor a horizontal kindling flue. The wood is packed closely and any

open spaces are carefully filled with sticks and brands. The mode of charring is similar to that in a heap. The wood is kindled from the central chimney or through the channel along the floor, and the fire is drawn outward and downward by means of the vent-holes, the necessary air entering through the holes in the lower door. When the central part of the wood has been well kindled, and the upper door closed and luted, bricks are placed in the two lower rows of vent-holes. The smoke issuing from the upper row will be white for several days, then turn yellow and finally become light-gray to colorless. The upper row of holes is now closed and the middle one opened; later follows the lower one, and when, in 6 to 10 days after kindling, the smoke from the lower holes has become light-colored to blue, the charring is finished. The lower holes are now closed, and the stall is allowed to cool for four to six days. The time of cooling can be



Figs. 125 and 126.—Conical stall.



Figs. 127 and 128.—Charring plant of the Cleveland Cliff Iron Co., Gladstone, Mich. *D*, Kindling door; *E*, flue; *F*, stack; *G*, flue leading to chemical plant; *H*, damper; *I*, lid to close stack.

reduced one-half by spraying water on the inside, but only at the expense of the quality of the charcoal and the life of the furnace.

It takes four to six men to charge a 35- to 45-cord stall in one day; charring lasts six to ten days, cooling four to six days; one man will watch six to ten stalls in a 12-hr. shift; two men discharge a stall in one day; 1 cord wood yields 45 to 50 bushels of charcoal.

By-product Recovery.—The stalls used with a partial recovery of by-pro-

ducts usually have the form shown in Figs. 127 and 128, *i.e.*, they are circular in plan; the side wall has some batter to withstand any outward thrust and is tied with heavy iron bands. The vent-holes are closed, and the gases and vapors are sucked off by means of a fan through branch flue *E* and stack *F* into the main flue *G* which leads them to the condensation (chemical) plant. Some stalls, *e.g.*, those of the Ashland Iron and Steel Co., Ashland, Wis.,[1] used to have a fireplace beneath the floor to assist in kindling and heating the wood on the floor.

The plant of the Cleveland Cliff Iron Co.,[2] shown in Figs. 127 and 128, consists of two rows of 30 stalls, represented in vertical section in Fig. 124. The trestle-work *T* carries the track on which the timber-cars arrive; beneath it run small cars *S* for carrying the charcoal buggies *U*. The volatile matter is carried off through the branch-flue *E*, chimney *F* and connecting flue *H* into the wooden gas-main *G* leading to the chemical works, 2,000 ft. away.

The stall, shown in section in Fig. 124, has a capacity of 60 cords of wood, principally maple and beech cut into 4-ft. lengths, split into pieces 6 to 7 in. thick, and seasoned eight to nine months. Sticks under 3 in. are not used. In filling, there is first placed on the floor a wood grating made up of rows 4 ft. apart, the sides facing the door. Next come the regular layers placed at right angles to the grating. Charging is begun from the lower and finished from the upper door. All open spaces are well filled with sticks and brands. When a stall has received its charge, doors *B* and *C* are closed and luted, lid *I* is removed from the chimney *F*, damper *H* having been lowered when the preceding charge was ready to be drawn. Kindling is introduced through *D*, ignited and the fire allowed to burn for from 2 to 3 hr.; the opening *D* is closed while the chimney *F* is left open until the gases are rich enough in V.H.–C. to be turned into the condensation plant, *i.e.*, after from 4 to 5 hr. with dry and from 10 to 12 hr. with wet wood. Lid *I* is now replaced on chimney *F* and damper *H* raised. Charring lasts six to eight days, cooling five to six days without watering; the charge shrinks to 8 ft. from the crown of the arch; charging requires 54 hr., and discharging 36 hr. The 60 cords of wood charged yield six cords (10 per cent.) brands, 48 bu. charcoal, 200 gal. acid liquor with 2 per cent. methyl alcohol, 250 lb. tar, 12,000 cu. ft. gas. Charcoal >1 in. in size goes to the blast-furnace, smalls, <1 in., to the boilers. The gases and vapors from the two rows of stalls are drawn by means of two fans through condensers which liquefy the tar and acid liquor, while the non-condensed gas is burnt under the boilers. The general scheme of treatment is given in the accompanying tree.[3]

[1] *Stahl u. Eisen*, 1896, XVI, 266; *Berg. u. Hüttenm. Z.*, 1896, LV, 266; *Jernkont. Ann.*, *Bihang*, 1907, VIII, 165.

[2] *Iron Trade Review*, Jan. 2, 1896; *Iron Age*, Jan. 24, 1901; *Min. World*, 1906, XXV, 10. Plant of Iron Dale, Mich., *Iron Trade Rev.*, 1907, XLI, 463.

[3] Details of treatment: S. P. Sadtler, "Handbook of Industrial Organic Chemistry," Lippincott, Philadelphia, 1912.

Klar, M. N. "Technologie der Holzverkohlung," Springer, Berlin, 1910.

15 wooden condensers
(5×5 in., 15 in. high)
(3 rows of 5 with copper water-circulating
tubes, 12 ft. long ×1/2 in. out. diam.)

- Gas
  - boilers
- Tar and acid liquor
  - 2 wooden settling tanks (10 ft. diam. 6 ft. high)
    - Tar
      - boilers
    - Acid liquor
      - 4 wooden storage tanks (12 ft. diam., 6 ft. high)
        - Distillation I (10 common copper stills, 9 ft. diam., 7 ft. high)
          - Acid liquor
            - 2 wooden lime-neutralizing tanks (10 ft. diam., 2 ft. high) with hand stirrers
              - 2 wooden settling tanks (10 ft. diam., 6 ft. high)
                - Some heavy oils
                  - Waste
                - Clear neutral acetate of lime solution
                  - Evaporate to gray acetate of lime
                    - Market
          - Distillate (4 to 5 per cent. methyl alcohol)
            - Distillation II (Column rectifying apparatus)
              - Residue
                - Waste
              - distillate (40–50 per cent. methyl alcohol)
                - Distillation III
                  - Low spirit liquor
                    - Distillation II
                  - Distillate
                    - Methyl alcohol 97+ per cent. pure

The charcoal plant of Pioneer Furnace No. 2, Marquette, Mich.,[1] has a battery of 86 stalls of 80 cords capacity arranged in two double rows, each of which flanks the sides of standard railroad trestles.

Continuous Kilns.—All the kilns so far described work intermittently. Ljundberg[2] has constructed a continuous stall which consists of blocks of four

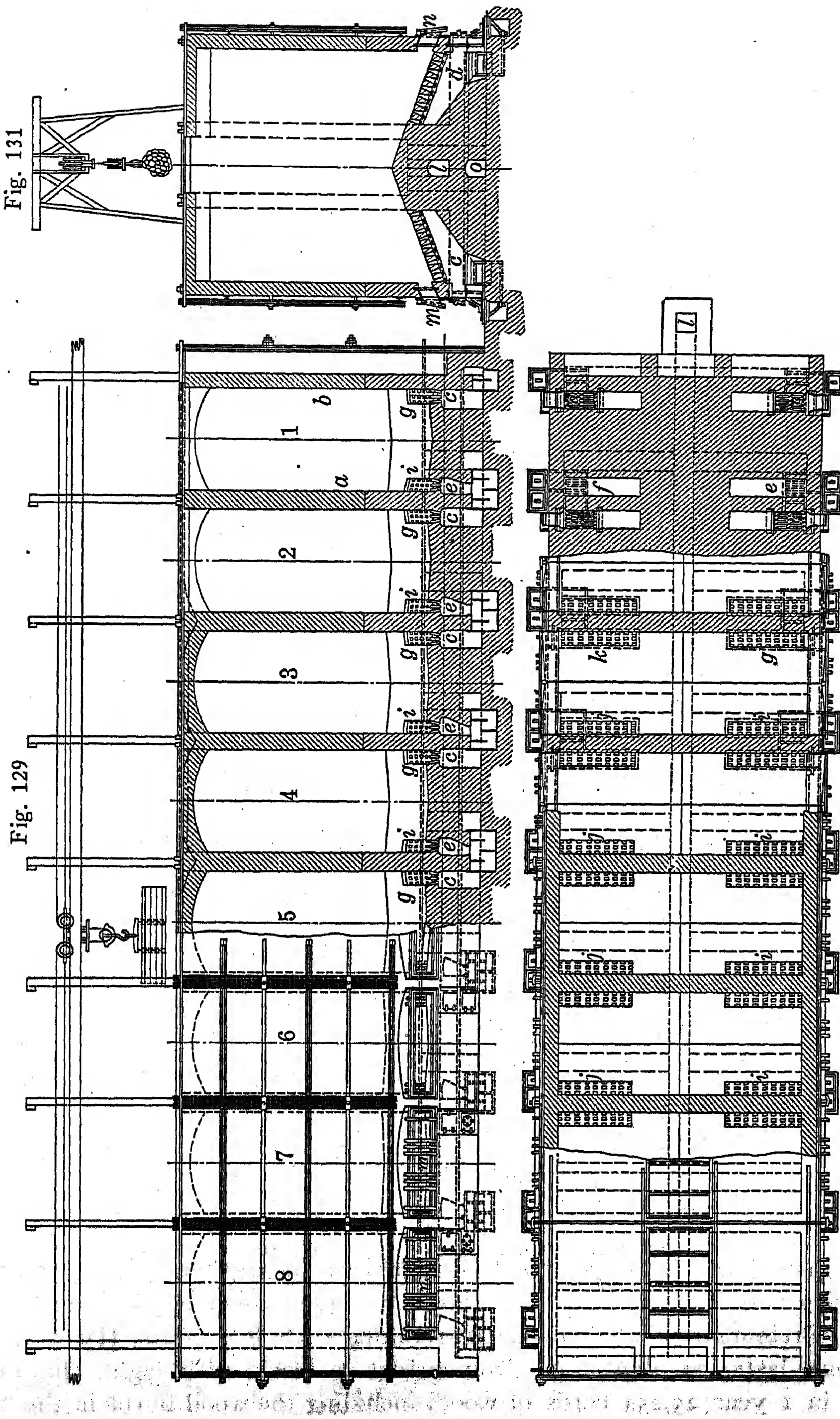
[1] *Lake Superior Min. Inst.*, 1903, IX, 89–91, plan of works; *Eng. Rec.*, 1907, LVI, 411.

[2] *Jernkont.*, 1897, 311; transl., *Iron Age*, July 28, 1898; *Tr. A. I. M. E.*, 1898, XXVIII, 103, 814; *Eng. and Min. J.*, 1898, LXVI, 309; *Colliery Guardian*, 1897, LXXIV, 1116; *Berg. Hüttenm. Z.*, 1898, LVII, 330.

Fig. 131

Fig. 129

Fig. 130



FIGS. 129 to 131.—Ljungberg continuous stall.

chambers connected with one another in which the operations of charging, drying, preliminary and final charring are carried on successively. The gases from the chamber for final charring are burnt in that for preliminary charring, and its products of combustion pass through the drying chamber to the fan furnishing the draft. Figs. 129 to 131 represent two blocks with chambers Nos. 1 to 4 and 5 to 8. The wood, of a length equal to the width of the chamber, is charged by means of a differential pulley, suspended from a traveler running on an overhead track, and the charcoal discharged at the sides through doors *m* and *n*, each consisting of three castings so as to be light and easy to manipulate. The inclined bottom of a chamber is trough-shaped, that the condensed water (30 per cent. of that of the wood) may collect and flow off at the center. Below the bottom there are on one side two fireplaces *c* and *d*, on the other two flues *e* and *f*; the products of combustion from the fireplaces enter the chamber through the ports *g* and *k*, and after circulating through the wood pass off through similar ports *i* and *j* into flues *e* and *f* from which they are directed by means of water-seal valves either into the adjoining fireplaces *c* and *d* or toward the main flue *l*. The necessary draft is furnished by a suction fan. Let us suppose the stall to be in operation, and the final charring to be carried on in chambers 1 and 5. Their fireplaces will be cold, but their hot gases will be directed under the grates of the preliminary charring chambers 2 and 6 on which wood is being burnt. The gases from 2 and 6 will dry and warm the wood in chambers 3 and 7, no fire being maintained on their grates. Cooling and discharging of charcoal followed by filling with fresh wood will take place in chambers 4 and 8. When after five days the final charring in chambers 1 and 5 is finished, they are cut off, cooled, discharged and refilled, which takes four days. The several operations are advanced one number as shown in Table 104.

TABLE 104.—ORDER OF OPERATIONS IN LJUNBERG CONTINUOUS STALL

Final charring		Preliminary charring		Warming and drying		Cooling, discharging and filling	
1	5	2	6	3	7	4	8
2	6	3	7	4	8	5	1
3	7	4	8	5	1	6	2
4	8	5	1	6	2	7	3

The gases from chamber 8 pass to chamber 1 through flue *o*. As cooling, discharging and filling takes only four hours, six hours are given to warming and drying. Preliminary and final charring taking each five hours, the cycle of operations lasts 20 days. At Domnarfvet a block with eight chambers charred in 1 year 25,253 cords of wood, including the wood burnt in the fireplaces, and yielded 72.56 per cent. by volume of charcoal included. Beside

the charcoal there were obtained per cord of wood 22.0 lb. tar, 21.4 lb. acetic acid, 7.7 lb. methyl alcohol and 176,192 cu. ft. gas (18° C., 760 m.m.).

117. **Comparison of Heaps and Stalls.**—A heap must be constantly watched; charring can be carried on only part of the year, and the result depends upon the state of the weather. In a stall the process is under much better control; one man can watch a number of stalls; charring is carried on winter and summer and is independent of the weather. The operation with the heap is slower and gives a smaller yield of charcoal. The stall gives a cleaner product, part of the V.H.–C. may be and is recovered, and there is little transportation loss of charcoal with the inherent breakage, as the plant is built near the blast-furnace. The disadvantages of stalls are: Transportation[1] of wood (five times the weight of the charcoal); cost, interest and maintenance of plant; and liability to large percentage (6+) of brands.

118. **Charring in Retorts.**[2]—Until within probably fifteen years retorts were used for charring only when the main object of the operation was the recovery of by-products.[3] As quick charring is favorable for the recovery of a high percentage of V.H.–C., the time given to working a charge is short and the charcoal obtained of a corresponding inferior quality not suited for blast-furnace work. It is used for domestic purposes, the manufacture of black powder, for foundry facings, etc.

The behavior of wood when subjected to destructive distillation is shown by the following examples. Gillot[4] carbonized in a closed iron vessel 5.25 ft. diam. and 3.28 ft. high a charge of 115 lb. of sticks of air-dried oak and elm, 12 in. long and 2 in. thick, in 4.25 days at a temperature rising gradually to 380° C., and obtained charcoal 32.63 per cent. wt., acetic acid 7.169 per cent., methyl alcohol 1.9 per cent., tar 2 per cent., water 40.128 per cent., gas 16.167, per cent., total 99.999 per cent.

Fischer[5] examined the gases from a plant in which air-dried beech wood was charred in large-size retorts. The gas analyses are recorded in Table 105.

There was obtained from 100 kg. wood: 45 kg. liquor (4 kg. acetic acid, 1.1 kg. methyl alcohol), 23 kg. charcoal, 4 kg. tar, 28 kg. gases of 20–25° C. containing on the average 20 kg. CO_2, 7 kg. CO, 0.5 kg. CH_4, 0.05 kg. H, 0.45 kg. H_2O, with a calorific power of 24,000 calories.

[1] Logging Industry, *Cass. Mag.*, 1906, XXIX, 443.

[2] Bergström, *Jernkont. Ann.*, 1908, LXIII, 301.

[3] Thenius, G., "Das Holz und seine Destillationsproducte," Hartleben, Vienna, 1880.
Klar, M., "Technologie der Holzverkohlung," Springer, Berlin, 1910.
Klar, *Chem. Ind.*, 1897, XX, 152, 176, 191, 219; *J. Soc. Chem. Ind.*, 1897, XVI, 667, 722.
Campbell, "The Wood Distilling Industry," *Met. Chem. Eng.*, 1910, VIII, 155.
Geer, "Wood Distillation," U. S. Dept. Agriculture, *Circular* 114, Nov. 5, 1907.
Yuon, *Chem. Trade J.*, 1909, XLIV, 87.

[4] "Carbonisation du Bois," Lacroix, Paris, 1873, pp. 43 *et seq.*

[5] Dingler, *Pol. J.*, 1880, CCXXXVIII, 55.
Wagner, *Jahresber.*, 1880, XXVI, 417.

TABLE 105.—GASES FROM CHARRING WOOD IN A RETORT

Sample taken after hours	CO_2	CO	CH_2	C_2H_4	H	N	O
3	58.99	31.29	3.94	trace	3 87	1.21	trace
4[1]	61.9	30.1	....	....	......	....	trace
5[1]	62.0	32 5	....	....	......	....	....
7[1]	59.8	31.9	....	....	......	....	....
8	57.48	34.52	4.67	0.45	2.78	trace	....
9[1]	58.7	33.8	....	....	......	....	....
10[1]	57.9	35 1	....	....	......	....	....
11	55 60	34.38	5.12	0 32	3 99	trace	....
15	64 68	30.32	2.43	0.69	1 21	trace	....

Sartig[2] records the following data: charring in 15 hr. air-dry beech wood at a temperature of 350° C. in vertical retorts (10 ft. long, 4 ft. diam., 0.5 in. thick) holding 3.07 cbm. wood gave per cbm. (=0.237 cord) or 395.9 kg. wood; 121.8 kg. charcoal; 157.2 kg. acid liquor (11.78 kg. acetic acid, or 27.0 kg. acetate of lime 80–82 per cent. pure, and 4.63 methyl alcohol); 23.9 kg. tar (sp. gr. 1.080); and 93.0 kg. gas. The coal consumption was 45.5 kg. The yield reduced to 100 kg. wood gives in round numbers 30 kg. charcoal, 40 kg. acid liquor (4.67 per cent. acetic acid, 1.17 per cent. alcohol), 6 kg. tar, 24 kg. gas. The author gives similar data for birch wood. A similar paper is that of Börnstein.[3] The present European practice is described in detail by Bühler.[4] Horizontal and vertical iron retorts 10.82 ft. long, 3.28 ft. diam., are in general use. They are externally heated by the non-condensed gases. Heating up a freshly charged retort requires 70–110 lb. coal. The charge of 2.3 cbm. (0.63 cord) air-dry wood (beech and birch, conifers give too little acid and furnish much tar rich in resin and turpentine) is charred in 12–16 hr. The dark-red charcoal is quickly drawn into an air-tight sheet-iron (0.12–0.16 in.) box to cool in 36 hr. In a few cases large-size vertical retorts internally heated through cast-iron tubes are in operation. They are 10.82 ft. sq. and 15.42 ft. long; the sides are of 0.25-in. boiler iron, top and bottom are 0.39 in. thick; there are 15 cast-iron heating tubes. A charge of 15 cbm. (3.55 cords) is worked in 24 hr. The yield of 100 lb. beech wood absolutely dry, is charcoal 24 per cent., acetate of lime 8 per cent., methyl alcohol 1 per cent., tar 24 per cent., gas 59 per cent.

The first retort-plant for making blast-furnace charcoal in the United States

[1] Examined on the spot for CO_2, CO and O only.

[2] *Chem. Z.*, 1893, XVII, 1269.
Fischer, *Jahresber.*, 1893, XXX, 15.

[3] *J. Gasbel. Wasserversorgung*, 1906, XLIX, 648.

[4] *Zt. angew. Chem.*, 1900, 155, full drawings.

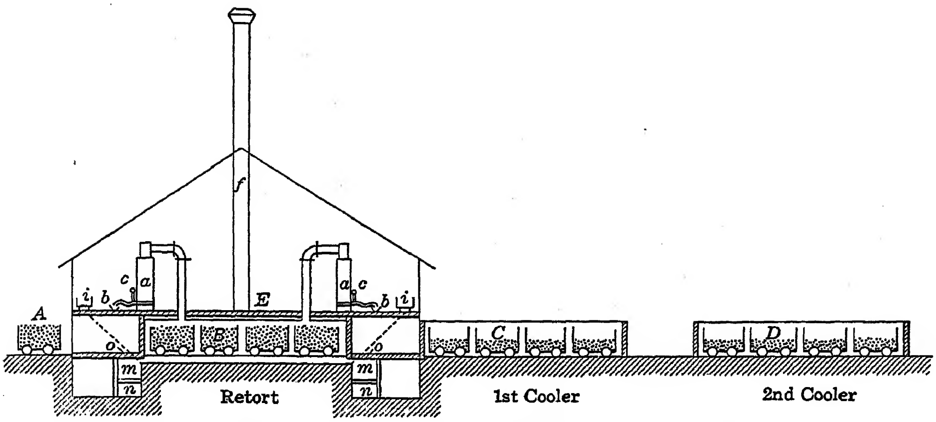
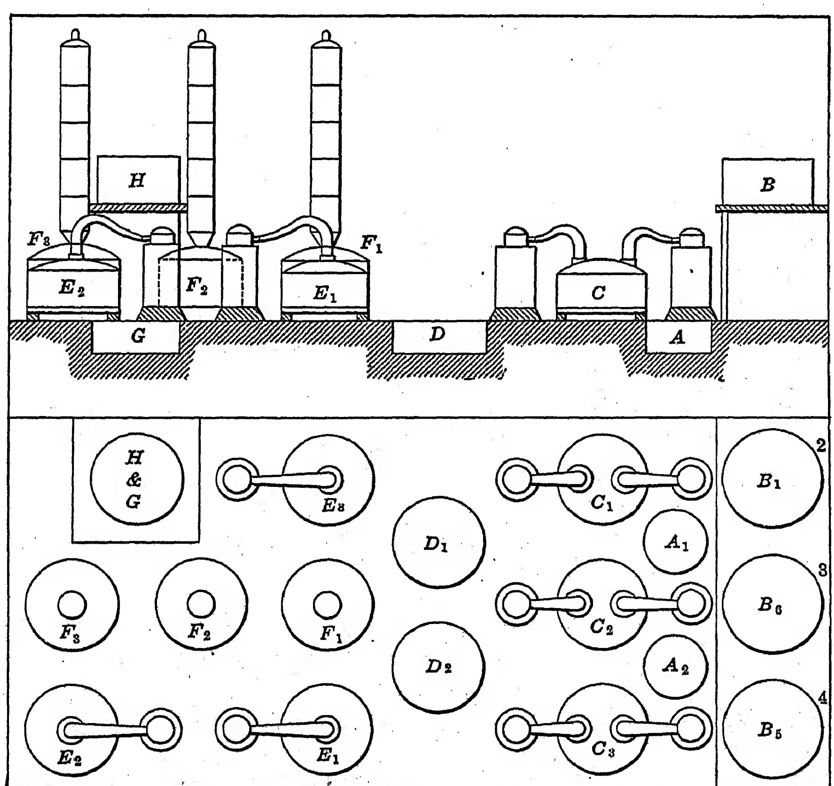


FIG. 132.—Retort charring plant, Algoma Steel Co., Sault Ste. Marie, Ontario.



FIGS. 133 and 134.—Chemical plant, Algoma Steel Co., Sault Ste. Marie, Ontario.

is probably that erected by Mathieu[1] at Luther, Mich., in 1882–83. The retorts were made of boiler iron and held 1 cord of wood. The body of the retort consisting of three convex and one concave plane was cylindrical; the ends were conical. Two inclined retorts facing one another with the concave planes were heated from a central fireplace. This form has fallen into disuse.

The plant of the Algoma Steel Co., Sault Ste. Marie, Ont.,[2] with a daily capacity of 160 cords wood, is typical for the present mode of operating. Fig. 132 represents a longitudinal section through one of the 20 units of the charring plant. It consists of a retort *B* with its first and second coolers *C* and *D*. The retort is a horizontal steel (1/4 in. bottom, 3/8 in. sides) shell, 46 ft. long $\times$6 ft. 3 in. wide$\times$8 ft. 4 in. high, closed air tight at the ends by a double set of swinging iron doors. It is set in brick-work as would be the case with a boiler. The fireplaces *m* with ash-pits *n* at either end burn mill refuse and furnish the heat necessary for charring. The refuse arrives in conveyor troughs *i* and is discharged by the swinging spouts *o*. The products of combustion travel around the retort and then pass off through the chimney *f*. A standard gauge track runs through the retort; it carries four iron trucks *A*, each of which has a skeleton-iron frame holding two cords of wood. Charring at a temperature of 370–400° C. takes 24 hr. The volatile matter ascends in two vertical pipes leading to condensers *a* from which the condensible liquor is collected in troughs *b* and run into collecting tank *A* for raw liquor of the chemical plant, Figs. 133 and 134, while the non-condensible part passes off through pipes *c* to the boiler plant which also burns the tar. When the wood is charred, the door of the retort is opened and the train of four cars hauled by means of a rope-drive into the first cooler *C*, similar in shape and size as the retort, but built of lighter material. The retort is filled again with four cars charged with wood. When the second charge has been charred, the partly cooled charcoal is transferred from *C* to *D*, the fresh red-hot charcoal from *B* to *C*, and *B* is re-charged. After remaining 24 hr. in *D*, the charcoal is sufficiently cool to go to the blast-furnace.[3] The scheme of operating the condensation plant, Figs. 133 and 134, is given in the accompanying tree.

[1] *J. U. S. Assoc. Charcoal Iron Workers*, 1884, V, 9, 385; 1885, VI, 60; *Eng. Min. J.*, 1884, XXXVII, 160; *Berg Hüttenm. Z.*, 1885, XLIV 9 (Åkerman).

[2] Sjöstedt, *Iron Age*, Jan. 28, 1904. Plant of Meshek Chemical and Iron Co., Wells, Mich., *Iron Trade Rev.*, 1912, LI, 619; *Eng. Min. J.*, 1912, XCIV, 747; Lake Superior Iron and Chemical Co., *Iron Age*, 1912, XC, 1211.

[3] Zwillinger, A New Charcoal Cooling Process, *Iron Age*, January 11, 1903.

- Collecting tank *A*
  - Raw liquor tanks B_{1-6}
    - Tar
      - Boiler
    - Liquor[3]
      - Tar stills C[1] $_{1-2}$ (3, reserve)
        - Distillate
          - Neutralizing tanks D_{1-2}
            - Lime stills $E^1{}_{1-3}$
              - Methyl alcohol (8–10 per cent)
                - Storage tank *G*[1]
                  - Alcohol still *F*[2]
                    - Methyl alcohol (82 per cent.)
                      - Storage tank *H*[3]
                        - Barrel room
                    - Residue
                      - Waste.
              - Lime acetate, Water
                - Evaporate in double-bottom steam pan, and then dry
        - Tarry waste

119. Comparison of Stalls and Retorts.—The leading advantages of retorts over stalls are: A higher yield in charcoal, methyl alcohol, and acetic acid; a more uniform quality of charcoal, and a lower cost of charring. Wherever there is a market for methyl alcohol and calcium acetate, the retort will be preferred to the stall.

120. Peat Charcoal (Coke).[4]—Peat, when subjected to destructive distillation, is decomposed at a temperature ranging between 200 and 300° C., and on the average[5] yields by weight, charcoal 35 per cent., gas liquor 40 per cent., tar 8 per cent., gas 17 per cent. The tar upon fractional distillation gives 2 per cent. light oils, 2.5 per cent. heavy oils, 1.5 per cent. lubricating oil, 0.5 per cent. paraffine, and 1.5 per cent. asphalt. Heine[6] gives as an average the following data: charcoal 40 per cent., gas liquor 33 per cent., tar 6 per cent., and gas 21 per cent. As peat is rich in ash, the charcoal cannot form a very valuable

[1] Tanks and stills are 14 ft. diam. and 6 ft. high.

[2] Distillate, originally brown, upon addition of $Ca(OH)_2$ becomes green (first pale, then olive), claret-color (neutral point), orange-yellow (slight excess of $Ca(OH_2)$).

[3] One cord of wood yields 275 gal. liquor with 5 per cent. $C_2H_4O_2$ and 3.5 per cent. CH_4O.

[4] Heber, *Braunkohle*, 1910, VIII, 744.

[5] Thenius, "Die Technische Verwerthung des Torfes, Hartleben, Vienna, 1904, p. 331. Börnstein, *Zt. Gasbel. Wasserversorgung*, 1906, XLIX, 649.

[6] *Zt. angew. Chem.*, 1896, LXI; see also *Jl. Assoc., Eng. Soc.*, 1907, XXXVIII, 233.

fuel unless the peat is first washed. Peat charcoal retains the structure of the raw peat, hence only peat compressed into briquettes is used in charring. The specific gravity of peat charcoal varies greatly with the purity and compactness of the peat. The weight of 1 cu. ft. peat charcoal ranges from 15 to 20 lb., depending upon the percentage of ash which is 4 to 5 for high-grade material, but generally exceeds 10. The absorbing power for gases, the ignition temperature, etc., are similar to charcoal. The calorific power ranges between 2,700 and 6,900 cal.; average 3,600. Heine[1] gives the ultimate analyses of high-grade German peat charcoal, shown in Table 106.

TABLE 106.—ULTIMATE ANALYSES OF GERMAN PEAT CHARCOAL

Locality	C	H	O	N	S	P	Ash	H_2O
Oberfranken	89.90	1.70	....	2.40	....	....	4.20	1.80
Oldenburg	83.06	0.91	4.18	......	0.27	0.075	3.86	7.65
Gifhorn	77.46	3.86	11.45		....	....	3.53	3.45

The usual range of composition is given[2] as C 60 to 86, H 2.2 to 4.4, O+N 6 to 17, H_2O 10 to 15, ash 10 to 20 per cent.; this shows that peat charcoal contains about 25 per cent. volatile matter. Peat charcoal is too weak to be used in a blast-furnace of any considerable height; it is satisfactory for domestic purposes, for raising steam, firing reverberatory furnaces and similar purposes.

The gas liquor is generally ammoniacal; very young peat may furnish an acid liquor owing to the excess of acetic acid evolved. The ammonia salts of the liquor are the carbonate and acetate; the liquor often contains some methyl alcohol. The tar has a light-brown color, a penetrating odor, and is rich in paraffine which causes it to solidify at 9° C.

Peat can be charred in heaps, stalls and retorts; electric charring has been tried in Norway.[3] As in charring peat shrinks more than wood, and the management of a heap becomes more difficult. Using peat briquettes, however, allows packing more closely than is possible with wood. The yield is about 30 per cent. by weight. Stalls show the same advantages over heaps as they do in charring wood; they should be erected at the bog which furnishes the raw material. The details about charring in heaps and stalls given in the literature are of ancient date. They need not be gone into, as in all probability no peat is charred at present in either heaps or stalls. Charring in retorts has taken a new lease of life by the advent of the Ziegler furnace.[4]

The Ziegler furnace consists of a pair of vertical oblong iron retorts heated

[1] *Loc. cit.*

[2] Kerl, "Metallhüttenkunde," p. 117.

[3] Hausding, *op. cit.*, p. 391; *Eng. Mag.*, 1902, XXIV, 210; *Berg. Hüttenm. Z.*, 1904, LXIII, 64.

[4] Hausding, *op. cit.*, p. 392; *Thonind. Z.*, 1901, XXV, 810; *Eng. Mag.*, 1902, XXIV, 209; *Eng. Min. J.*, 1907, LXXXIII, 143.

Zwingenberg, *J. Am. Peat Soc.*, 1908, I, 53.

with the non-condensed gases of the destructive distillation; enough excess gas is available for heating the boilers of the plant. The retort, after having been heated up, works continuously; some coke is discharged at intervals from the bottom and a corresponding amount of briquette charged at the top. The temperature of the flues near the bottom of the retort reaches 1,110° C.; that of the upper flues is 400° C.; the products of combustion pass off at 300° C.; the temperature in the retort does not exceed 600° C. The volatile matter is drawn off at the middle and near the top of the retort. A furnace (dimensions not given) treats 18 tons in 24 hr. Schönelding[1] states that in Oldenburg air-dry peat briquettes give by weight: peat charcoal 35 per cent., gas liquor 20 per cent., tar 4 per cent., water 20 per cent., gas and loss 20 per cent. The composition of the charcoal is: C 84.9, H 1.2, O 2.5, ash 2.8, H_2O 8.6, calorific power 7,500 cal. The gas liquor from 100 lb. peat gives $(NH_4)_2SO_4$ 0.4 per cent., $(C_2H_3O_2)_2Ca$ 0.6 per cent., CH_4O 0.2 per cent. A working test on 500 tons Oldenburg peat in five furnaces gave Wolf[2] from 100 lb. peat briquettes, charcoal 27.3 per cent., gas liquor 31.2 per cent., tar 4.5 per cent., gas 37.0 per cent. By fractional distillation the gas liquor yielded CH_4O 0.34 per cent., NH_3 0.16 per cent., $C_2H_4O_2$ 0.44 per cent.; and the tar: light oils 2 per cent., heavy oils 0.7 per cent., paraffine 0.3 per cent., lubricating oil (phenols) 1.3 per cent., asphalt 0.2 per cent. According to Bleymüller,[3] 1/4 to 1/3 of the charcoal in the iron blast-furnace of Schmalkalden, Germany, has been replaced by Ziegler peat charcoal. In 1907 only three plants were in operation in Germany.[4]

121. Brown-coal (Lignite) Coke.[5]—Many experiments have been made to produce from brown coal a coherent coke suitable for metallurgical purposes, but practically all have failed, as brown coal upon heating breaks up into small fragments. It has been mixed with good coking coal and has given a satisfactory coke.[6] Briquetting brown coal with pitch bond and coking the briquettes promises well.[7] It may be inferred from Schorr's tests[8] that brown coal briquetted with oil residues and coked in a modified Appolt oven[9] will give good furnace coke. Schnablegger's experiments are promising.[10]

Brown coals near Halle, Germany, are subjected to destructive distillation in retorts.[11] From the tar[12] are produced paraffine, paraffine oil, solar oil, benzine, creosote and pitch; the coke, called "Grude," 40 to 50 per cent. of the weight of the coal, forms a domestic fuel and is used in gas producers.

[1] *Gluck Auf*, 1900, XXXVI, 796.

[2] *Verh. Verein Beförd. Gewerb.*, 1903, LXXXII, 295.

[3] *Stahl u. Eisen*, 1905, XXV, 1326.

[4] *Eng. Min. J.*, 1907, LXXXIII, 143.

[5] Dumble, E. T., "Brown Coal and Lignite," Geol. Survey of Texas, 1902, pp. 69, 74.

[6] von Tunner, *J. I. and St. I.*, 1882, I, 96; Friderici, *Oest. Zt. Berg. Hüttenw.*, 1882, XXX, 45.

[7] Coal Testing Plant, U. S. Geol. Survey, St. Louis, 1904, *Prof. Paper* No. 48, Part III, p. 1454.

[8] *Eng. Min. J.*, 1905, LXXIX, 322.

[9] *Eng. Min. J.*, 1905, LXXX, 1115.

[10] *Oest. Zt. Berg. Hüttenw.*, 1902, L, 195.

[11] *Berg. Hüttenm. Z.*, 1890, XLIV, 308; *Min. Ind.*, 1900, IX, 170.

[12] Graefe, Ed., "Die Braunkohlenteer-Industrie," Knapp, Halle, 1906, p. 104.

122. Bituminous-coal Coke.[1]—The first attempts at coking bituminous coal were made in the sixteenth century near Cassel, Germany.[2] In 1589[3] coking experiments were carried on in England. In the eighteenth century coke began to replace charcoal as a metallurgical fuel; at present, excepting a few districts, it is the sole carbonized metallurgical fuel.

The coking power of coking coals varies greatly (§103). Assuming 8 per cent. ash as an average, the coking coals of the U. S. contain 18 to 32 per cent. V.M. and 74 to 60 per cent. F.C. A coking coal subjected to destructive distillation[4] gives off some H_2O and a little gas at 200° C. At about 300° C. it begins to soften, fuse and swell; at 400° C. tarry vapors with illuminating gases are set free (paraffines C_nH_{2n+2}, olefines C_nH_{2n}, phenols $C_nH_{2n-2}OH$); with increase of temperature to 500 to 600° C. the remaining H_2O is decomposed, the amount of heavy hydrocarbons (paraffines and olefines) set free decreases (being in part decomposed into C, light C_xH_y and H) and the volume of gases decreases; at 900° C. practically gases only pass off, and at 1,200 to 1,400° C. the decomposition of the coal is completed. The character and amount of products obtained vary with the kind of coal, the form of furnace and the manner of operating. The volatile matter is collected as gas, gas-liquor and tar; the residue forms the coke. One net ton of coking coal[5] will yield 9,000 to 9,600 cu. ft. gas containing H 50 to 55 per cent., CH_4 35 to 40 per cent., C_mH_n (heavy hydrocarbons) 2.5 to 3 per cent., CO 0.5 per cent., CO_2 1.5 per cent., and small amounts of N; 200 lb. gas-liquor with 2 to 3 per cent. NH_3 present as free NH_3, and in combination with Cl, CO_2, S, CN; 65 lb. black coal tar consisting of liquid and solid hydrocarbons, phenols and asphalt-forming constituents; and 70 to 75 per cent. coke containing the fixed carbon and ash. Table 107 by

[1] Dürre, E. F., "Die Anlage und der Betrieb der Eisenhütten," Baumgärtner, Leipsic, 1881, I, pp. 239-275; Supplement, 1892, p. 164.

Fulton, J., "Coke," International Text Book Co., Scranton, Pa., 1905.

Simmersbach, O., and Anderson, W. C., "Chemistry of Coke," Hodge and Co., Glasgow-Edinburgh, 1899.

Weeks, J. D., "Report on the Manufacture of Coke," *Tenth Census*, U. S., 1880, Vol. x ("Petroleum, Coke and Building Stone").

Byrom, T. H., and Christopher, J. E., "Modern Coking Practice," Crosby, Lockwood & Son, London, 1910.

Lewes, V. B., "Carbonization of Coal," John Allan & Co., London, 1912.

Percy, C. M., "Improvements in Coke Manufacture in Europe," *Coll. Eng.*, 1888–90, Vols. 8–10.

Rossigneux, "Fabrication der Coke," *Bull. Soc. Ind. Min.*, 1891, v, 387, 505; *Coll. Guard.*, 408, LXV, 1893, 457, 501.

[2] Say., *Bull. Soc. Ind. Min.*, 1909, x, 157, 365, 463, 551.

K., *Stahl u. Eisen*, 1894, XIV, 202,255.

[3] Beck, L., "Geschichte des Eisens," Vieweg, Brunswick, 1892, II, 752, 784, 1268; 1897, III, 306, 928.

[4] Börnstein, "*Braunkohle*," 1906, v, 415.

Porter and Ovitz, *Eng. Min. J.*, 1908, LXXXVI, 720; *Bull.* 1, Bureau of Mines, 1910.

Parr, S. W., and Olin, H. L., "Coking of Coal at Low Temperatures," *Bull.* 30, Univ. Ill., June 3, 1912.

[5] Lunge, *Min. Ind.*, 1896, v, 182.

Schniewind[1] gives the yields in coking some American coals in Otto-Hoffmann (§131) ovens. The data may serve as guides for other ovens.

TABLE 107.—YIELD OF COKE FROM AMERICAN COALS IN OTTO-HOFFMANN COKING OVENS

Source of coal	Location of plant	Coke, per cent.	Tar, per cent.	Ammonium sulphate, per cent.	Gas per net ton coal, cu. ft.
Cape Breton, N. S.	Everett, Mass.	72.83	4.99	1.010	9,000
Youghiogeny, Pa.	Glassport, Pa.	75.60	5.07	1.100	9,000
Connellsville, Pa.	Working tests	76.34	6.14	1.223	8,924
Pittsburgh, Pa.	Working tests	68.25	4.38	0.908	8,884
Eastern Pa.	Working tests	85.00	2.00	0.800	8,400
Virginia	Working tests	66.01	4.70	1.070	10,090
Kanawha, W. Va.	Working tests	73.60	6 40	1.000	10,289

The treatment of the violatile matter is discussed in §134. The solid product of carbonization is either gas-coke, made in coal-gas retorts, or metallurgical coke, made in a variety of ovens. Only the latter is treated in this text.

123. Properties of Coke.—Blast-furnace coke[2] must have the right color and luster, and a well-developed structure; it must be hard, sonorous and strong, in lump form and contain little breeze, and lastly it must come up to certain standards of purity.

COLOR AND LUSTER.—The color ranges from silver-gray to black, usually iron-gray; the luster from silky and metallic to dull, beehive-coke having a larger amount of graphitic C than retort-coke, and being quenched with exclusion of air, has a lighter color and is more lustrous than retort-coke. The irridescence of occasional pieces of coke is said to be due to films of FeS.

STRUCTURE.[3]—The surface of coke is rough; here and there pieces are smooth and glossy like graphite; hair-like threads are sometimes found on a piece of coke. Coke is a porous vesicular mass consisting of a large number of irregular cells joined together by walls that are impervious to gases. This imperviousness is seen clearly in the photomicrographs prepared by Thörner. Fig. 135,[4] representing Rhenish coke made in a Coppée oven (§131), shows the two constituents, white cell-spaces and black cell-walls (compare with photomicrograph of charcoal, Fig. 115).[5] The porosity, therefore, depends solely upon the number and size of the cells. Coke from moderately fine coal of uniform size is denser

[1] *Iron Age*, Nov. 28, 1901.
[2] Thau, *Glück auf*, 1907, XLIII, 277.
[3] Wedding, *Zt. Berg. Hütten. Sal. Wesen i. Pr.*, 1876, XXIV, 409.
Dewey, *Tr. A. I. M. E.*, 1883–84, XII, 111.
Kutscher, *Stahl u. Eisen*, 1885, V, 794.
Thörner, *op. cit.*, 1886, VI, 72.
[4] *Stahl u. Eisen*, 1886, V, Plate V, Fig. 5.
[5] Roush, *Metallurgie*, 1912, IX, 160; *J. Ind. Eng. Chem.*, 1911, III, 368.

and heavier than if the coal used is in lump form or of uneven size (crushing, screening 1/6 to 3/8 in., compressing coals). The greater the ratio V.H.–C.: F.C. in a coal, the larger will be the size and the number of cells. But the same coal will make a lighter coke if the initial coking temperature is high than if low; the higher the fuel-column, the higher the temperature after most of the volatile matter has been driven off, and the longer the time given to coking the stronger will be the coke. The size of cell varies greatly. With gas-coke the cells are frequently large enough to give the coke the appearance of a coarse sponge; with blast-furnace coke, they are very small. Dewey[1] gives the percentage of cell-spaces of American blast-furnace coke made in the beehive as ranging between 42.96 and 61.12 per cent.; Fulton[2] gives 44 per cent., the figure of Connellsville beehive coke, as the most desirable volume of cells for American iron blast-furnace practice. European blast-furnace coke made in retort-ovens has 40 to 50 per cent. cell-spaces, while foundry coke has 25 to 40 per cent. Coke to be used in lead, copper and other non-ferrous blast-furnaces which are run with a low pressure blast ought to be more porous than the coke of the iron blast-furnace. The percentage of cells in coke which is to be burnt as a domestic or a boiler fuel should be larger than that in the coke of lead and copper furnaces; such coke should also contain a larger amount of volatile matter. Coke from retort-ovens shows less cell-spaces than beehive coke, as is shown by the data on Connellsville coke, Table 108, given by Fulton.

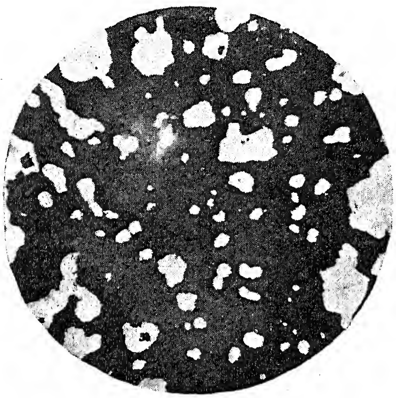


FIG. 135.—Photomicrograph of coke. (Thörner).

TABLE 108.—CELL-SPACES IN BEEHIVE AND RETORT-OVEN COKE FROM CONNELLSVILLE COAL

Connellsville coal coked in the	Weight of 1 cu. ft.-lb.		Volume per cent.		Ultimate compressive strength lb. per sq. in.	Hardness	Specific gravity	Proximate chemical analysis					
	Dry	Wet	Cell-walls	Cell-spaces				F.C.	H_2O	Ash	S.	P.	V.M.
Beehive oven, 24 hr....	47.69	82.20	43.93	56.07	1,088	3	1.74	86.88	0.79	11.54	0.695	0.005	1.31
Huessener oven.......	57.13	89.45	48.24	51.76	1,548	3.1	1.89	86.38	0.03	13.08	0.630	0.015	0.51
Otto-Hoffmann oven At side.	55.79	80.07	61.13	38.87	1,860	....	.....	.....	.....	.....	.....	.....	.....
At bottom..........	78.07	92.30	77.22	22.78	3,760	....	.....	.....	.....	.....	.....	.....	.....
Average...........	66.93	86.18	69.17	30.83	2,808	3.3	1.90	85.60	0.12	12.26	0.520	0.006	2.02

[1] *Loc. cit.*

[2] *Op. cit.*, p. 332.

HYGROSCOPICITY.[1]—Coke is hygroscopic. Dry coke exposed to moist air absorbs about 2.5 per cent. H_2O; when immersed in water, it takes up 20 to 50 per cent., but readily gives up again the major part, retaining 10 to 15 per cent. H_2O. Retort blast-furnace coke usually contains 4 to 6 per cent. H_2O; beehive coke quenched in the oven 2 to 3 per cent., but it absorbs enough H_2O, when exposed for some time to moist air, for its moisture content to reach 4 to 6 per cent.

SPECIFIC GRAVITY.[2]—This is governed by the percentage of cell-walls and of ash. According to Dewey[3] the true or actual specific gravity (relation between coke-substance and equal volume of water) of American beehive coke varies from 1.49 to 1.83, and the apparent specific gravity (relation between entire volume of coke and equal volume of water) from 0.713 to 0.935. Retort-oven coke gives higher figures for the apparent specific gravity, as it has a smaller percentage of cells than beehive cokes. Thus Muck[4] gives 1.2 to 1.9 as the range of the true specific gravity of European coke. One cu. ft. of American furnace coke with an apparent specific gravity of 0.9 weighs when in lumps about 30 lb., and in the form of breeze 32 lb.; gas-coke is lighter, it weighs 23 to 30 lb. per cu. ft.; 1 bushel of coke weighs about 40 lb. European coke weighs 22 to 23 lb. per cu. ft.

HARDNESS.—The hardness of coke depends upon the character of the cell-walls, and this upon the general character of the coal, the percentage of ash (especially of SiO_2), the kind of furnace used and the time given to coking. The hardness varies from 2.1 to 3.6. Fulton[5] gives the hardness of American beehive coke as ranging from 3.0 to 3.6. Blast-furnace coke must be hard, as soft coke is friable and readily attacked by CO_2 at a red heat. Fronheiser[6] conducted CO_2 over anthracite and coke heated in a glass tube; the percentage of CO in the gas leaving the tube gave the means of comparing the hardness. His data are given in Table 109.

TABLE 109.—RESISTANCE OF ANTHRACITE AND COKE AT ELEVATED TEMPERATURE TO CO_2

Kind of fuel	CO_2	CO	Scratch hardness	Ultimate compressive strength, lb. per sq. in.	Coking oven
Anthracite	96.0	4.0	2.5	3,000	
Coke, Connellsville	94 5	5.5	3.7	2,260	Otto-Hoffmann.
Coke, Connellsville	91.9	9.0	3.0	1,204	Beehive.
Coke, Morris Run	88.8	11.2	2 6	1,360	Semet Solvay.
Coke, Bennington	86.1	13.9	2.4	848	Beehive.

[1] Report of Tests, *Stahl u. Eisen*, 1908, XXVIII, 800, 997, 1325; 1909, XXIX, 28.
Steingroever, *Glück auf*, 1908, XLIV, 1601.
Crisfield, *J. Frankl. Inst.*, 1911, CLXXII, 495.

[2] Thörner, *Stahl u. Eisen*, 1888, VIII, 591.

[3] *Tr. A. I. M. E.*, 1883–84, XII, 111.

[4] "Chemie der Steinkohle," Engelmann, Leipsic, 1891, p. 200.

[5] *Tr. A. I. M. E.*, 1883–84, XII, 220.

[6] Fulton, *op. cit.*, 359.

TABLE 110.—PHYSICAL PROPERTIES OF SOME KINDS OF AMERICAN COKE

Locality	Weight, lb. per cu. ft.		Volume, per cent.		Ultimate compressive strength, lb. per sq. in.	Hardness	Specific gravity	Proximate analysis						Oven
	Dry	Wet	Cell-walls	Cell-spaces				F.C.	H_2O	Ash	S.	P.	V.M.	
Connellsville, Pa.....	47.69	82.20	43.93	56.07	1,160	3.0	1.80	86.880	0.790	11.540	0.695	0.05	1.310	Beehive, 48-hr.
Caledonia, Pa.......	46.12	83.07	40.83	59.17	596	3.0	1.80	87.890	0.130	9.420	1.570	0.0240	0.990	Beehive, 48-hr.
Caledonia, Pa........	51.69	85.38	46.07	53.93	836	3.0	1.81	......	......	......	.....	....	.....	Beehive, 72-hr.
Pocahontas, Va.......	59.68	89.64	52.07	47.93	944	2.5	1.83	92.640	0.345	5.822	0.738	0.0063	3.41	Beehive.
Big Stone Gap, Va...	45.31	80.70	43.34	56.66	980	2.5	1.67	92.050	0.290	5.600	0.740	0.0090	1.32	Beehive, 48-hr.
Blocton, Ala.........	54.56	85.80	49.97	50.03	1,636	2.6	1.75	92.760	0.153	6.940	0.740	0.0066	0.81	Beehive.
Pineville, Ky........	53.73	84.73	50.37	49.63	908	2.6	1.77	94.660	1.140	3.780	0.590	0.0070	0.41	Beehive.
Powelton, W. Va.....	53.88	87.33	46.48	53.52	1,524	2.6	1.86	91.048	0.017	7.548	0.626	0.0070	2.90	Beehive.
Cumberland, Tenn...	56.68	85.00	50.39	49.61	1,924	2.9	1.81	87.150	0.910	10.320	0.970	0.0140	1.62	Beehive.

The figures for Connellsville show that when coked in the retort-oven, the coke resists oxidation by CO_2 better than when in the beehive. Other data are those by Bell,[1] Åkerman[2] and Lévèque.[3]

SONOROUSNESS.—Good coke always gives a clear metallic ring when struck; this property furnishes a simple means of approximating the quality.

STRENGTH.—The strength of coke depends, beside the porosity and hardness, upon the percentage of ash and the temperature at which coking was carried on. It is, on the whole, the smaller the higher the ash-content, and the higher the percentage of SiO_2 in the ash.[4] There are cases on record in which the ash being readily fusible has given strength to the cell-walls. At an elevated temperature coke is weaker than at ordinary temperature. Thus Bell[5] found in one instance that the resistance of coke when hot was 33 per cent. less than when cold; Holtzer[5] in another found the loss in strength to be only 7 per cent. Table 110 gives the physical properties of some kinds of American coke.[6]

According to Simmersbach-Anderson,[7] Westphalian coke has an average ultimate compressive strength of 1,700 lb. per sq. in. Tests with different ovens at the Bessége works, Terrenoire, gave Jougouet[8] the data shown in Table 111.

TABLE 111.—ULTIMATE COMPRESSIVE STRENGTH OF COKE MADE IN DIFFERENT OVENS

Oven	Carvès, width of retort, in.			Beehive	Belgian (Smet)	Coppée
	$27\frac{9}{16}$	26	$19\frac{11}{16}$			
Ultimate compressive strength, lb. per sq. in.	945	1,135	1,315	625	755	1,145

In order to improve the structure and with it the strength of coke, finely crushed and wetted coal is sometimes compressed into a cake,[9] and this resting

[1] "Principles of the Manufacture of Iron and Steel," Spon, London, 1884, p. 287.

[2] *Oest. Jahrb.*, 1889, XXXVII, 127.

[3] *Bull. Soc. Ind. Min.*, 1906, V, 433.

[4] Simmersbach-Anderson, *op. cit.*, p. 120.

[5] *Bull. Soc. Ind. Min.*, 1891, V, 396.

[6] Fulton, *op. cit.*, p. 334.

[7] *Op. cit.*, p. 119.

[8] *J. I. and St. I.*, 1880, I, 144.

[9] Quaglio, *Verh. Verein. Bef. Gewerb.*, 1892, LXXI, 416.

Simmersbach, *Stahl u. Eisen*, 1898, XVIII, 1078 (*Iron Age*, Dec. 29, 1898); 1905, XXV, 1058, 1347 (*Eng. Min. J.*, 1906, LXXXII, 208).

H. W., *Coll. Guard.*, 1899, LXXV, 1115.

Ellison, *Tr. Inst. Min. Eng.*, 1900–01, XXI, 89 (*Berg. Hüttenm. Z.*, 1892, LXI, 69).

A, *Stahl u. Eisen*, 1900, XX, 1248.

Brink and Hübner, *op. cit.*, 1901, XXI, 73.

Editor, *Rev. Un. Min.*, 1901, LIII, 308.

on a plate or peel, is charged into a horizontal retort-oven and the peel then withdrawn. The compression decreases the volume of the coal by 45 per cent.; deducting 15 per cent. for necessary clearance space and 20 per cent. increased coking time, leaves an increase of coke output of 10 per cent.[1] This advantage is more than balanced by the cost of repairs in machinery and of additional labor, and by the difficulty of operating in cold weather. Poorly coking coals high in volatile matter may require this treatment; coals that swell much in coking may not be compressed, as compression of course increases this tendency.

The strength of blast-furnace coke has been and often still is measured by determining the resistance it offers to compression. A true cubic inch[2] (U. S.) or 1 c.c. (European Continent) is prepared by sawing and then grinding on an emery-wheel to the required size, and then subjected to pressure in a suitable machine until it begins to crack. As sawing wears out a blade quickly,[3] it is preferable to shape a specimen roughly with nippers. The apparatus for compression used in Germany is described by Simmersbach-Anderson[4] and Thörner.[5] These compression tests cannot well give the average strength of a lot of coke, as it is impossible to obtain an average sample and as the average of even a large number of tests can only be approximately correct. One way out of the difficulty is to confine compression tests to picked samples and base comparisons on them. But in the blast-furnace there is not so much danger of coke being broken by the weight of the charge as by its torsional, abrasive and grinding effect. Wedding[6] calculated that in an iron blast-furnace 98 ft. high the compression exerted upon the square inch of coke was 4.29 lb. and of charcoal 1.42 lb. The strength and friability of coke has been determined for years at the Creuzot works, France,[7] by charging 110 lb. coke into a tumbler, 3.28 ft. diam. with 3 radial arms 8 2/3 in. long, allowing the tumbler to revolve 4 min. at the rate of 25 r.p.m., and then to screen through a sieve with holes 1.5748 in. sq. A good coke should not give more than 30 per cent. fines. A similar mode of procedure, the rattler-test, has given satisfactory results in testing paving material and brick.[8]

Darby, *J. I. and St. I.*, 1902, I, 26.
Ernst, *Iron Age*, 1906, LXXVII, 663.
Lackawanna Steel Co., *Zt. Berg. Hütten u. Sal Wesen i. Pr.*, 1906, LIV, 205.
Ernst Coal-charging Machine, *Iron Age*, 1907, LXXX, 1682.
Thau, *Glück Auf*, 1907, XLIII, 925.
Crooke, *Stahl u. Eisen*, 1908, XXVIII, 1350.
Schreiber, *op. cit.*, 1911, XXXI, 1896.

[1] Atwater, *Tr. A. I. M. E.*, 1903, XXXIII, 767.
[2] Fulton, *Tr. A. I. M. E.*, 1883–84, XII, 214.
[3] Phillips, *Eng. Min. J.*, 1898, LXV, 67.
[4] *Op. cit.*, p. 121.
[5] *Stahl u. Eisen*, 1886, VI, 74.
[6] "Handbuch der Eisenhüttenkunde," Vieweg, Brunswick, 1906, Vol. III, p. 780.
[7] Rossigneux, *Bull. Soc. Ind. Min.*, 1891, V, 394; further Weill, *Rev. Mét.*, 1905, II, 557; *Metallurgie*, 1905, III, 36.
[8] Orton, Ohio Geol. Survey, 1893, VII,[1] 189, 247; *Tr. Am. Ceramic Soc.*, 1911, XIII, 792. Brown, *op. cit.*, 1910, XII, 270.

A method is used in Pennsylvania which may be called a drop test, a wooden box, 18×18 in., and 12 in. deep, having a hinged bottom, is filled with about 50 lb. coke. The coke is made to drop three times from a height of 6 ft. onto a cast-iron plate, and then sifted with a screen having openings 2 in. sq.; with a good coke the oversize is not less than 70 per cent. and usually reaches 82 per cent.

CHEMICAL COMPOSITION.—Coke consists mainly of C and ash; it contains small quantities of H, O, N, S and some V.M., and also some water.

TABLE 112.—ULTIMATE ANALYSES OF EUROPEAN COKE

Locality	C	H	O	N	Ash	S	Reference
Germany, Ruhr District...	91.772	1.225	0.040	.. ..	6	933	Muck, *op. cit.*, p. 203.
England, Dukinfield.......	85.840	0.520	1.	380	11.400	0.860	Percy, *op. cit.*, p. 417.
England, Durham.........	91.580	0.230	1.310	.....	6	860	Weeks, *op. cit.*, p. 72.
Belgium, Mons........ ...	80.850	0.510	2.130	.....	16.	510	Weeks, *op. cit.*, p. 72.
Belgium, Séraing..........	91 300	0.330	2.	17	6.200	.... ..	Weeks, *op. cit.*, p. 72.

In the United States ordinarily only the proximate analysis is made giving F.C., V.M., Ash, S, P, H_2O. In Tables 108 and 110 such analyses were given.[1]

CARBON.—The carbon content of coke ranges from 83 to 93 per cent.; it is composed of the F.C. of the coal, the C. set free by the decomposition of some of the V.H.–C., and the C. of the V.M. still retained by the coke.

ASH.—The percentage of ash varies from 4 to 15 per cent.; good coke should contain less than 12 per cent. As the ash has its source in the inorganic matter of the coal and remains practically unchanged in coking, it has the same composition as that of the coal (Table 80).

GASES.[2]—The gas present in coke can be only partly attributed to volatile matter. In blast-furnace coke its amount varies from 0.5 to 4 per cent. Of the elements H, O and N, H is the most important, reaching 2 per cent. by weight. Parry[3] believes that H is present as occluded gas, while Muck[4] holds that H and O form stable compounds with C. Thus, *e.g.*, coke-hairs gave Platz: C 95.729, H 0.384, comb. O 3.887, free O, 0.715, ash none. Phillips[5] found needle-coke to be composed of C 97.55, H 1.12, O 1.23, N 0.00, ash 0.10; total 100.00, S 0.27. The amount of O may reach 8 per cent. According to Knublauch,[6] 31 to 36 per cent. of the N in the coal remains with the coke, which retains about 0.7 per cent. Rossigneux[7] found 1.84 per cent. N in an English coke.[8]

[1] See also Weeks, *op. cit.*, p. 22; Moldenke, *Bull.* 3, Bureau of Mines, 1910.
[2] Thörner, *Stahl u. Eisen*, 1886, VI, 77.
[3] *Chem. News*, 1872, XXV, 98; *Dingl, Pol. J.*, 1872, CCIV, 470.
[4] "Steinkohlen-Chemie," Engelmann, Leipsic, 1891, p. 204.
[5] *Eng. Min. J.*, 1897, LXIV, 760.
[6] *Stahl u. Eisen*, 1885, V, 436.
[7] *Bull. Soc. Ind. Min.*, 1891, V, 413.
[8] See also Short, *J. Soc. Chem. Ind.*, 1907, XXVII, 581.
Bertelsmann, "Der Stickstoff der Steinkohle," Enke, Stuttgart, 1904, p. 86.

Sulphur.[1]—The amount varies from 0.5 to 2.5 per cent. Good iron blast-furnace coke should contain less than 1 per cent.; best coke less than 0.5 per cent. In smelting sulphide non-ferrous ores, the percentage of S in coke is of minor importance. As a rule the coke contains less S than the coal from which it has been made for the reason that its main source is FeS_2 which gives up part of its S upon heating. There are, however, cases[2] in which the percentage of S in the coke is higher than in the coal. The cause of this is that while the volatile matter is expelled, Fe_xO_y, CaO and MgO may combine with S which otherwise would be volatilized. Hence coal rich in Fe_xO_y, CaO and MgO cannot give coke low in S. Most of organic and sulphate sulphur of the coal remain with the coke; some S is removed in quenching the coke. As a rule about 40 per cent. of the S of the coal is expelled in coking.

Phosphorus.—The percentage of P in coke is low, 0.01 to 0.03 per cent. All the P of the coal remains with the coke.

Kindling Temperature.—Coke begins to kindle at a dull-red heat (about 700° C.). In order to burn it requires a deep layer and a strong draft; single pieces taken from the fire soon grow cold. Coke burns without flame and is, therefore, an excellent fuel when heating by contact or radiation is required. The calorific power of coke, after deducting ash and water, is about 8,100 cal.

124. Modes of Coking.—The manner of coking has to be adapted to the coal. In general[3] it may be said that it is difficult to obtain a good blast-furnace coke from coal containing over 28 per cent. volatile matter. From coal low in volatile matter a satisfactory blast-furnace coke may be obtained by coking quickly at a high temperature,[4] from coal high in volatile matter by coking slowly at a low temperature; often different grades of coal are mixed to obtain the desired result. Coal is frequently crushed,[5] and washed to reduce the percentage of ash and S. The bulk of the water of washed coal is removed by draining, while the coal is stored in pockets, but the drained coal still retains 5 to 7 per cent. H_2O. This water is disadvantageous for coal requiring quick coking, and may have to be removed; it is advantageous for slow coking, and dry unwashed coal is sometimes wetted before charging into the oven. Different grades of coal are more easily mixed when moist than when dry; moist coal also lies more compactly in the ovens than dry; the water finally takes up space otherwise occupied by air.

Coking is carried on either in heaps, open and closed stalls, when air has limited access to the coal, or in retorts when it is excluded. In the former case, part of the fixed carbon is consumed to furnish the necessary heat, and the volatile matter as a rule is burnt as much as possible in contact with the coal; in

[1] Wuest-Wolff, *J. I. and St. I.*, 1905, I, 406.
Lodin, *Am. Min.*, 1906, IX, 87.
Short, *J. Soc. Chem. Ind.*, 1907, XXVII, 581.

[2] Simmersbach, *Stahl u. Eisen*, 1898, XVIII, 18.

[3] Sheldon, *Iron Age*, 1908, LXXXI, 197.

[4] Hartman, *Met. Chem. Eng.*, 1912, X, 522.

[5] Hanford, *Mines and Min.*, 1907, XXVII, 396, coal-crushing plant.

a few cases part of the volatile matter is saved and converted into a marketable product (waste-heat coking-ovens). With retort-ovens, volatile matter only is burnt in flues surrounding the chambers which contain the coal, and furnishes the heat necessary for coking. While at first all the gases and vapors issuing from a retort were burnt, now part of them are first extracted as by-products (tar, ammonia) and the more permanent gases or only part of them are returned to the oven and used as fuel for coking,[1] the rest being available for heating or illuminating purposes. The by-products have become the basis of an important industry, and especially so in Europe. In England and the United States, where coking with access of air has been almost exclusively used, retort-ovens are replacing the older apparatus.

Coking Ovens.—The two types of ovens are given as follows:

I. Coking with air having limited access to coal.
  A. Heaps.
  B. Open Stalls.
  C. Beehives: Ordinary Beehives. Modifications of Adams, Thomas, Aitken, Jameson, Newton-Chambers, Pernolet, Hemingway, etc.

II. Coking with air having no access to coal.
  1. Intermittent Ovens:
    A. Retort-ovens with vertical chambers: Ovens of Appolt, Appolt-Schorr, Bauer, etc.
    B. Retort-ovens with horizontal chambers.
      a. Vertical Flues: Ovens of François-Rexroth, Coppée-Otto, Bernard, Otto-Hoffmann, Otto, Otto-Hilgenstock, Schniewind or United-Otto, Brunck, Bauer, Ruppert, Collin, Coppée, Kopper, Poetter, Müller, Kros.
      b. Horizontal Flues: Ovens of Smet-Büttgenbach, Simon-Carvès, Seibel, Huessener, Festner, Collin, Bernard, Semet-Solvay, Rothberg.
  2. Continuous Ovens:
    A. Ovens of Lürmann, Sheldon, Woodall-Duckham.

125. Coking with Air Having Limited Access to Coal (Heaps).[2]—Coking in

[1] Blauvelt, "By-product Coke-ovens," *Min. Ind.*, 1895, IV, 215; 1897, VI, 171; *Proc. Am. Soc. Mech. Eng.*, 1908, XXX, 247; *Tr. A. I. M. E.*, 1912, XLIV, 196.

Lunge, "Modern Coke-ovens and their By-products," *Min. Ind.*, 1896, V, 179.

Schniewind, "The Manufacture of Coke in the U. S. with Special Reference to the Markets for By-products," *Min. Ind.*, 1901, X, 135; 1902, XI, 158; *Tr. A. I. M. E.*, 1912, XLIV, 186.

Firket, "Fabrication du Coke," *Ann. Min. Belg.*, 1903, VIII, 390.

Hurez, "Les Fours à Coke à Récuperation," *Bull. Soc. Ind. Min.*, 1903, II, 777.

Byrom, T. H., and Christopher, J. E., "Modern Coking Practice," Crosby, Lockwood & Son, London, 1910.

Hartman, *Proc. Eng. Soc. West. Pa.*, 1912; *Iron Age*, 1912, XC, 85; *Iron Trade Rev.*, 1912, LI, 309; *Met. Chem. Eng.*, 1912, X, 521.

Herbst, *Internat. Congr.*, Düsseldorf, 1910, Vol. Praktisches Hüttenwesen, p. 115.

Rau, *ibid.*, p. 280.

Lecocq, *Rev. Mét.*, 1912, IX, 681.

[2] Percy, *op. cit.*, p. 422.

Weeks, *op. cit.*, p. 82.

heaps is the earliest method. It has become obsolete on account of the fact that it requires lump-coal and gives a small yield of coke (50 to 55 per cent. *vs.* 60 to 70 per cent. in ovens) that is dirty and not uniform in size and quality. The operation in circular heaps consists in piling coal around a central perforated brick chimney, kindling the coal from the foot of the chimney, checking the fire at the surface from the base to the top as soon as the volatile matter has burned off by giving a cover of coke-breeze and earth, and finally smothering it. Heaps 18 to 30 ft. in diam. and 5 to 6 ft. high, holding about 20 tons of coal, burn 5 to 8 days, cool 3 to 4 days, and are then watered and drawn. Oblong heaps were once used by the Cambria Iron Co.[1]

126. Open Stalls (Kilns).[2]—A coking stall is 44 to 60 ft. long, 8 ft. wide and 5 ft. high, enclosed by brick walls having air openings in the sides, and charging and discharging openings in the ends. This method is an improvement on coking in heaps, as stalls permit the use of small coal, and as the temperature is more easily regulated, but the yield is still small and the quality of the product uneven. The method has only historical interest.

127. Closed Stalls or Beehives.[3]—Beehives were first used in England about 1750, and were introduced in the United States in 1835.

The *Ordinary Beehive*, Figs. 136 to 139,[4] commonly used in the United States, is a dome-shaped structure (11 to 12 ft. in diam., 6 to 7 ft. high, walls 9 in. thick) of siliceous fire-brick[5] with a flat sloping (4 to 5 in.) bottom (tiles 12 in. sq. and 3 in. thick), holding 5 to 6 tons of coal. It has two openings; one in the crown (12 in. in diam. lined with silica-brick) which serves as charging-opening for the coal and as outlet for the waste gases; the other, an arched door, 30×30 or 30×32 in., in the side at the bottom which serves as inlet for the air necessary for combustion, and as a drawing-door for the coke.[6] During the coking, it is closed to the spring of the arch by a half course of brick laid

[1] *Second Geol. Survey of Pa.*, Vol. L, p. 122.

[2] Percy, *op. cit.*, p. 426.
Weeks, *op. cit.*, p. 85.

[3] Fulton, *op. cit.*, p. 148.
Weeks, *op. cit.*, p. 86.
Phillips, *Eng. Min. J.*, 1897, LXIV, 726, 760; 1898, LXV, 67.
Keighly, *Am. Mfr.*, 1898, LXIII, 481, 952; 1899, LXIV, 14 and 543.
Catlett, *Tr. A. I. M. E.*, 1899, XXIX, 84; 1903, XXXIII, 272.
Bennen, *Mines and Minerals*, 1901, XXI, 387; 1901–02, XXII, 110.
Mullen-Fulton, *op. cit.*, 1903, XXIV, 4.
Editor, *Eng. Min. J.*, 1906, LXXXI, 267.
Eavens, *Appal. Eng. Assoc.*, 1906; *Coll. Guard.*, 1906, XCII, 597; *Mines and Minerals*, 1906, XXVII, 80.
Judd, *Eng. Min. J.*, 1906, LXXXII, 877; *Industrial World*, 1906, XL, 711; *Mines and Minerals*, 1907, XXVII, 278; *Tr. Eng. Soc. West. Pa.*, 1906, XXII, 327.
Galloway, Historical Notice, *Eng. Min. J.*, 1909, LXXXVIII, 11.
Campbell, *Eng. Min. J.*, 1909, LXXXVIII, 120.
Beldin, *Eighth Internat. Congr. Appl. Chem.*, New York, 1912, Vol. X, p. 53.

[4] *Eng. Min. J.*, 1906, LXXXI, 267.

[5] McKinley, *Mines and Minerals*, 1907, XXVII, 313; Campbell, *op. cit.*, 1908, III, 456.

[6] Peters, Door, *Eng. Min. J.*, 1910, XC, 180.

dry and plastered over with clay. Sometimes the bricks are enclosed in an iron door-frame which is hinged or suspended.

The ovens are built in single banks, Fig. 137, or in double banks, when the ovens are placed either back to back or are nested, Fig. 138. The ovens, as many as 150 in a row, are enclosed by retaining walls, the spaces between the ovens and walls are packed with loam which, storing up heat, reduces the loss by radiation and keeps the oven walls at an even temperature.[1] The space underneath the bottom should be tamped with at least 3 ft. of clay or good selected loam to insure the bottom being kept dry and hot. Nesting gives compactness to the structure. This arrangement is believed by some coke-men not to work as evenly as that in which the ovens are placed opposite one another, back to back, as less heat is stored up in the smaller amount of loam required to fill the smaller space between the side- and the oven-walls. Along the center line of a double row of ovens is placed a drain (not shown) to draw off any moisture. Over the furnaces is a track (70- to 80-lb. rails) of a 4- to 6-ft. gauge having a grade of 1 in.:100 ft. It is laid with iron tie-pieces and carries the coal cars (larries) which hold a fuel charge. The wharf ("yard level" in Fig. 136) on the front is 2 ft. 9 in. to 3 ft. below the bottom of an oven, and is 20 to 36, average 30, ft. wide to permit storage of coke. It is 7 to 21 ft. above the track of the railroad; 8 ft. in Fig. 136. Along the front of a row of furnaces, about 3 ft. underground, is a 3- to 4-in. cast-iron water-pipe, from which the water for quenching the coke is delivered to 1 in. stand-pipes situated between the ovens. The estimated cost of the plant of 118 single-row ovens of the Joseph Wharton Coke Works,

FIG. 138.

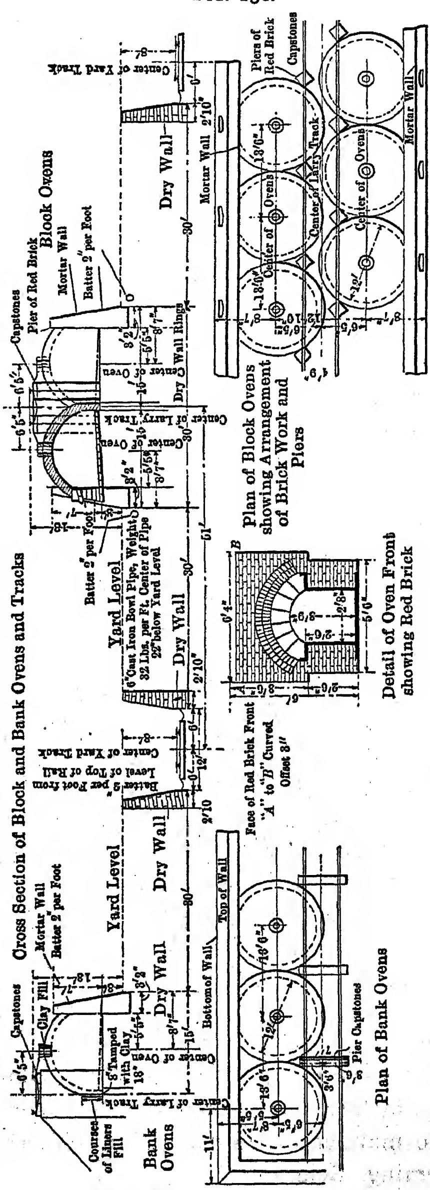


FIG. 136. FIG. 137.

FIGS. 136 to 139.—Beehive coke-ovens.

[1] Lee, Use of Concrete for Stonework, *Mines and Minerals*, 1910, XXX, 429.

Indiana Co., Pa., erected in 1903, was $311.70 per oven.[1] A double row would be somewhat cheaper.

Only coal which is high in volatile matter and gives with a low temperature a dense strong coke is suited for the beehive; coal that is low in volatile matter and requires a high temperature, furnishes a coke that is porous and weak.

In starting a new plant, either all the ovens are dried and warmed in 8 to 10 days by a wood fire, adding lump coal from time to time, or only a dozen ovens are dried; coking is begun in them and the imperfect coke from the first charges used to warm the other ovens. The first charge for the warmed oven is about 100 bu. (7,600 lb.) coal. Hot coke is placed in front of the oven door; the lump, coal from the dropped charge is drawn toward the coke and followed by the fine coal; the door is closed with brick, leaving an opening near the floor. The first charge will be finished ("off," "around") in 2 to 3 days; the coke is drawn hot and quenched on the wharf. It is an inferior product ("off-coke") and is sold or reserved for warming ovens. The second charge, 115 bu. (8,740 lb.) coal, is kindled as was the first; it is sometimes leveled in the oven; coking lasts 48 hr.; the coke may be slightly watered in the oven, but is preferably quenched on the wharf. It is better than the first charge, but not yet normal. Enough heat has now been stored in the oven to ignite the third charge of 125 to 130 bu. (9,700 to 9,880 lb.). Coking lasts 48 hr. and may furnish a fair blast-furnace fuel. The fourth charge should be normal, *i.e.*, 5 to 6 tons of coal.

Supposing an oven to be working normally. When the coke has been drawn, a new charge of 5 to 6 tons is dropped through the opening in the crown, and leveled; the front is bricked up to within about 2 in. from the top and daubed. In leveling, the coal is not only spread, but lump and fine coal are so disposed as to be evenly distributed and to leave no open spaces. The charge, about 23 in. deep, cools somewhat the parts with which it is in contact; the dome of the oven, however, remains hot; it radiates heat and starts the distillation on the surface of the coal. For 2 or 3 hours a black smoke issues from the tunnel-head; it burns with a reddish-yellow flame. This increases in size as the oven grows hotter and extends about 4 ft. out of the tunnel-head; it gradually becomes paler and in about 10 hr. after charging it will have become whitish; after 10 hr. more it will be thinner and shorter, and at the end of the next 15 hr. it will just reach to the tunnel-head and then draw back so that it cannot be seen; small blue jets (candles) will issue here and there from the coke to the end of the 48-hr. coking period. The composition of the gases varies with the time at which they are taken, the character of the coal, the dimensions of the oven and the manner of operating. Ebelmen[2] analyzed the gases from a beehive at Seraing, Belgium:

Temperature measurements by Campbell[3] made at different points of beehive ovens are shown in Table 114.

[1] *Mines and Minerals*, 1903–04, XXIV, 5, drawings and specifications.

[2] *Ann. Min.*, 1851, XIX, 139; *Berg. Hüttenm. Z.*, 1851, X, 468.

[3] *Eng. Min. J.*, 1909, LXXXVIII, 120.

TABLE 113.—COMPOSITION OF GAS FROM BEEHIVE COKING OVEN

Components	2 hours after charging	7.5 hours after charging	14 hours after charging	Average of preceding analyses
	Smoke black and dense	Flame reddish, bright, no smoke	Flame clear, small in volume	
CO_2	10.13	9 60	13.06	10.93
CO	4.17	3.61	2.19	3.42
CH_4	1.44	1.66	0.40	1.17
H	6.28	3.67	1.10	3.68
N	77.98	81.16	83.25	80.80

TABLE 114.—TEMPERATURES IN BEEHIVE OVENS

Type of oven	Kind charge	Time burnt	Point where temperature was taken	Deg. C.
	Hours	Hours		
Bank Beehive	48	24	Front of oven, directly inside of arch.	875
Bank Beehive	48	24	Top of charge, 4 ft. inside of arch	1,010
Bank Beehive	48	24	Crown of oven, 3 ft. below tunnel	1,290
Bank Beehive	48	33	Front of oven, directly inside of arch.	830
Bank Beehive	48	33	Top of charge, 4 ft. inside of arch	1,065
Bank Beehive	48	33	Crown of oven, 3 ft. below tunnel	1,230
8-ft. Rect.	48	30	Front of oven, directly inside of arch.	1,050
8-ft. Rect.	48	30	Crown of oven, 3 ft. below tunnel	1,120
6-ft. Rect.	48	26	Front of oven, directly inside of arch.	980
6-ft. Rect.	48	26	Crown of oven, 3 ft. below tunnel	1,180
Block Beehive	48	30	Front of oven, directly inside of arch.	845
Block Beehive	48	30	Top of charge, 4 ft. inside of arch	1,050
Block Beehive	48	30	Crown of oven, 3 ft. below tunnel	1,370
Block Beehive	48	30	Intersection of outside arch with front wall.	430 to 540
Bank Beehive	72	0	Bottom of oven, under tile at center of oven.	260
Bank Beehive	72	72	At point midway between tunnel and over arch.	830
Bank Beehive	72	72	Maximum temperature inside tunnel	1,260
Bank Beehive	72	72	Bottom of oven, under tile at center of oven.	825
Bank Beehive	72	72	Bottom of oven, top of tile, after 16 min. watering.	65

A short time after the ignition of the vapors and gases, the coal will frit on the surface, fuse, swell and begin to crack downward, thus enabling the vapors from below to pass off. The surface becomes more or less uneven and the cracks increase in size as the coking progresses downward. The coke near the top becomes red-hot, the vapors set free in the lower part of the charge ascend and passing through the porous coke are in part decomposed and give up some of their C; the bottom becomes hotter until finally it has been converted into red-hot coke broken into basaltiform columns[1] (needles) by the vertical cracks shown in Fig. 140. While coking, the swelling of coal may cause the original 23-in. depth of the charge to increase to 31 in. When most of the volatile matter has been driven off, the charge will shrink and the coke be only 21 in. deep, showing a slight depression below the tunnel-head.[2]

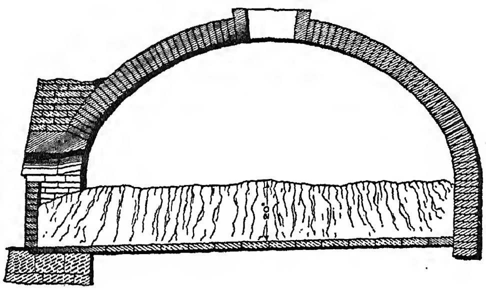


FIG. 140.—Needle structure of coke.

The coke is now quenched in the oven for 15 to 35 min., requiring about 105 gal. of water per net ton; the tunnel-head is closed, the coke drawn, forked and loaded into cars by two men in 2.5 hr.; the time between quenching and re-charging amounts to about 3.5 hr. Mechanical coke-drawers[3] have been constructed in order to hasten the discharge.

The coke from the top and bottom, and from near the door of the oven is softer and more porous than that from the body. On the top of the bed and near the door are found the ashes of the coke which has been burnt. By proper drafting,[4] the burning of coke can be reduced to 0.5 per cent. of the F.C. found by the crucible assay. A top covered with a film of fused coke (black top) is a sign of excessive heat in the crown which melted the coal too quickly and, forming a hard crust impermeable to gases, was not glazed later on. A bottom showing an inch or more of fused uncoked coal (black butt) is caused by drawing

[1] Schreiber, *Stahl u. Eisen*, 1904, XXIV, 521.

[2] Fulton, *op. cit.*, pp. 159 and 161 illustrates these changes.

[3] Cook, *Tr. A. I. M. E.*, 1896, XXVI, 347.

MacFarren, *Proc. Eng. Soc. West. Pa.*, 1907, XXIII, 451; *Iron Trade Rev.*, 1907, XLI, 995, 1035.

Wickes, *Tr. A. I. M. E.*, 1906, XXXVI, 353.

Nixon, *Eng. Min. J.*, 1907, LXXXIV, 119.

Marmac, *Iron Age*, 1907, LXXX, 1807.

Catlett, *Tr. A. I. M. E.*, 1903, XXXIII, 272.

Affelder, *Mines and Minerals*, 1908, XXIV, 484; *Eng. Min. J.*, 1909, LXXXVII, 510; 1910, XC, 180; Hebb, *Iron Age*, 1908, LXXXI, 1299.

Covington, *Iron Age*, 1908, LXXXI, 1241; 1910, LXXXV, 24; *Eng. Min. J.*, 1910, LXXXIX, 578; *Coll. Guard.*, 1910, XCIX, 186.

Fulton, *Mines and Minerals*, 1909, XXX, 247.

Stauft, *Iron Trade Rev.*, 1912, LI, 884.

[4] Catlett, *Tr. A. I. M. E.*, 1903, XXXIII, 272.

the oven too quickly or by overcharging it or by running it too hot. Coke with a dull, dark, smoky, dirty appearance is obtained when there is insufficient supply of air.

In drawing, the coke is forked (1.5-in. spaces between tines) to separate fine coke or breeze (2 to 3 per cent.) from lump coke.

Coking commonly takes 48 hr. As no work is done on Sundays, the ovens charged on a Friday are discharged only on the following Monday; they thus make 72-hr. coke which, being denser, harder and stronger, usually goes to foundries.[1] The yield of coke varies with the percentage of F.C., the form of the oven and the manner of working. Fulton[2] made some tests in a beehive (12 ft. diam. × 6 ft. 9 in. high) with Connellsville coal having the following composition: H 1.25, V.M. 31.27, F.C. 59.79, ash 7.16, S 0.53, P 0.024 per cent. Assuming 40 per cent. S to be volatilized, the theoretical yield of coke would be: 67.27 per cent. He obtained from 43,650 lb. of coal, with four 48-hr. charges, 66.17 per cent. marketable coke, 2.30 per cent. fine coke and 0.82 per cent. ash, or a total of 68.47 per cent. The increase in yield of 1.75 per cent. over the theoretical more than makes up for the loss by combustion. A second test with four 72-hr. charges gave from 41,650 lb. of coal 59.66 per cent. marketable coke, 3.28 per cent. fine coke and 0.82 per cent. ash, or a total of 62.94 per cent., corresponding to a loss of 6.45 per cent. of the theoretical amount. In retorts the same coal gave 79.20 per cent. coke or a gain of 11.64 per cent. upon the 48-hr. charges in the beehive. Phillips[3] records details of tests with 48-, 72- and 96-hr. coke made in a beehive oven at Birmingham, Ala. The oven was 12 ft. in diam., and 26 in. high to spring of arch. The coal of one of the 48-hour tests, deducting 5 per cent. H_2O, weighed 11,024 lb. and showed the following composition: V.M. 32.43, F.C. 60.91, ash 6.66, S 1.91 per cent. The coke was sorted with a fork, 21 in. wide with 1.5-in. spaces; it gave 53.30 per cent. marketable coke (V.M. 1.51, F.C. 88.90, ash 9.59, S 1.37) and 2.92 per cent. breeze (V.M. 1.47, F.C. 56.00, ash 42.53, S 1.14). The black ends gave upon analysis V.M. 1.82, F.C. 89.00, ash 9.18, S 1.29. The leading results are summarized in Tables 115 and 116.

TABLE 115.—CHEMICAL CHANGES OF ALABAMA COAL WHEN COKED IN BEEHIVE OVENS

	V.M., per cent.	F.C., per cent.	Ash, per cent.	S., per cent.	Yield of coke from dry coal, per cent.	Yield of breeze and ashes, per cent.	Increase of C from coal to coke, per cent.	Increase of ash from coal to coke, per cent.	Decrease of V.M. from coal to coke, per cent.	Decrease of S from coal to coke, per cent.
Coal	32.48	60.80	6.72	1.91	....	....	....	....	....	....
Coke, 48-hr.	1.51	88.90	9.59	1.37	58.78	2.92	46.21	42.71	95.35	28.27
Coke, 72-hr.	1.71	88.35	9.94	1.31	59.77	2.58	45.31	48.51	94.73	31.41
Coke, 96-hr.	1.06	89.63	9.31	1.34	57.51	2.17	47.41	38.54	96.73	29.84

[1] Moldenke, R., "Coke Industry of the U. S. as Related to the Foundry," *Bull.* 3, Bureau Mines, 1910.

Wüst-Ott, *Stahl u. Eisen*, 1906, XXVI, 841.

Simmersbach, *op. cit.*, 1909, XXIX, 1551.

[2] *Op. cit.*, p. 158.

[3] *Eng. Min. J.*, 1897, LXIV, 726, 760; 1898, LXV, 67.

TABLE 116.—PHYSICAL PROPERTIES OF ALABAMA BEEHIVE COKE AND TIME OF COKING

Time of coking	Apparent specific gravity	True specific gravity	Volume of cells, per cent.	Volume of cells in 100 parts by weight of coke	Compressive strength, lb. per sq. in.
Coke, 48-hr.	1.029	1.913	46.58	46.29	1,760
Coke, 72-hr.	0.875	1.785	52.22	61.25	2,200
Coke, 96-hr.	0.921	1.839	48 84	54.30	2,640

The irregularities in this table are probably due to the difficulty of obtaining true samples.

COST OF COKING.—Keighley[1] gives the cost of a ton of coke in 1896 as \$1.3338. He assumes a plant of 300 ovens, 12×7 ft., producing 175,000 tons coke per year from a coal yielding 66.6 per cent. coke; he allows 5 years for the life of an oven (cost \$300) and \$20 for repairs; 6 per cent. interest; a sinking fund of \$3000 per year assuming the field to be worked in 30 years:

TABLE 117.—COST OF COKING IN BEEHIVE OVEN (1896)

1.5 tons coal delivered at 60 cents	\$0.9000
Yard expenses	0.3266
Repair, wear and tear	0 0343
Interest at 6 per cent.	0.0308
Taxes, insurance, salesmen, etc.	0 0250
Sinking fund	0 0171
	\$1 3338
Coke-forking	0.0500
	\$1.3838

Fulton[2] gives the cost for 1899 as \$1.75; d'Invilliers,[3] assuming a plant of 500 ovens, estimates for the period 1899–1903, the cost of producing 1 ton of coke in the Connellsville region with a yield of 67 per cent. at \$1.216 (selling price \$2.304); and in the Reynoldsville region with a yield of 57 per cent. at \$1.53 (selling price \$2.06). The average prices of 1 short ton of coke at the ovens in the different states and territories of the United States are given in the U. S. Geol. Survey in the annual *Mineral Resources of the U. S.* Thus in 1910, the lowest cost was \$1.83, Virginia, the highest \$5.83, Washington, the average being \$2.39.

128. Modified Beehive Ovens.—In order to reduce the arduous and lengthy work of drawing coke, attempts have been made to change the form of the oven.

[1] *Am. Mfr.*, 1898, LXIII, 734.
[2] *Op. cit.*, p. 346.
[3] *Tr. A. I. M. E.*, 1905, XXXV, 44; *Mines and Minerals*, 1904–05, XXV, 313.

The *Adams Improved Beehive*[1] has a movable bottom which is raised and lowered by means of an hydraulic ram running on rails under a row of ovens. The *Thomas Coke-Oven*[2] is an oblong arched chamber with swinging doors at the ends; the coke is drawn out in a body at the front by means of chain and drag. The *Rainey Oven*[3] is similar.

With the ordinary beehive all the excess heat and by-products go to waste. The heat of the burning gases issuing from the tunnel-head has been utilized for the generating of steam.[4] Attempts have also been made to modify the beehive in such a manner as to permit the recovery of by-products. Internally fired ovens, however, furnish little NH_3 and the tar formed,[5] being a low-temperature product, resembles rather an oil; it is rich in paraffine, but yields no benzole or anthracene, and has a low market value. Ovens of this type are those of Aitken,[6] Jameson,[7] Newton-Chambers.[8]

The ovens of Pernolet,[9] Hemingway,[10] and others form the transition between the internally fired beehive and the externally fired retort.

129. Coking with Air Having no Access to Coal.—In coking with exclusion of air, the coal, contained in an oblong narrow retort of siliceous fire-brick or silica brick[11] is subjected to destructive distillation by burning all or only part

[1] McKelvey, *Coll. Engr.*, 1889–90, x, 130.
Fulton, *Coll. Engr.*, 1890–91, xi, 8.

[2] Hill, *Ala. Sc. Ind. Soc.*, 1891, ii, 86.

[3] *Eng. and Min. J.*, 1892, liv, 512; *Iron Age*, Oct. 27, 1892.
Fulton, *op. cit.*, p. 164.
Iron Trade Rev., Nov. 22, 1906.

[4] Ramsay, *Ala. Sc. Ind. Soc.*, 1893, iii, 40, 62.
Bowron, *Coll. Engr.*, 1897–98, xviii, 60.
Wood, *Eng. and Min. J.*, 1897, lxiv, 369.
Eavenson, *Mines and Minerals*, 1905, xxv, 587.
Judd, *Eng. and Min. J.*, 1906, lxxxii, 880; *Proc. Eng. Soc. West. Pa.*, 1906, xxii, 327.
Allard, *Mines and Minerals*, 1908, xxviii, 387.
Affelder, *op. cit.*, 1908, xxix, 195.
Elliot, *op. cit.*, 1909, xxix, 352.
Martin, *op. cit.*, 1909, xxx, 129.
Reynolds, *op. cit.*, 1912, xxxiii, 135.
Guiterman, *Eng. and Min. J.*, 1908, lxxxviii, 1011.
Hall, *Coal Age*, 1912, i, 936.
Sheridan, *Tr. A. I. M. E.*, 1909, xl, 354; *Mines and Minerals*, 1911, xxxii, 142.

[5] Lunge, *Min. Ind.*, 1896, v, 186.

[6] *North Engl. Instit. Min. Eng.*, 1879, xxix, 81; *Min. Ind.*, 1895, iv, 219.

[7] *Eng. and Min. J.*, 1883, xxxvi, 248; *J. I. and St. I.*, 1883, ii, 504; *Stahl u. Eisen*, 1883, iii, 349, 423, 515, 564.

[8] *Tr. A. I. M. E.*, 1896, xxvi, 340, 347; *Iron Age*, Oct. 3, 1895; *Min. Ind.*, 1905, iv, 219.

[9] *North. Engl. Instit. Min. Eng.*, 1872–73, xxii, 3; *Rev. Un. Min.*, 1883, xiv, 146; *Bull. Soc. Ind. Min.*, 1891, v, 454; *Coll. Engr.*, 1889–90, x, 147.

[10] *Eng. and Min. J.*, 1900, lxix, 142; *Iron Age*, Jan. 8, 1900; *Mines and Minerals*, 1901, xxi, 412; *Coll. Guard.*, 1901, lxxxi, 1078.

[11] Hartmann, *Proc. Eng. Soc. West. Pa.*, 1912, xxviii, 313.
Destruction of brick: Schreiber, *Stahl u. Eisen*, 1910, xxx, 1839; Kossmann, *op. cit.*, 1911, xxxi, 730.

of the combustible volatile matter of the coal in flues surrounding it. A number of retorts, 12 to 60, and in recent years 120, are united to a block to reduce the loss by radiation and to diminish the cost of plant and management. Coking in retorts had its origin in Belgium in 1842; the term Belgian oven was once used synonymously with retort-oven.

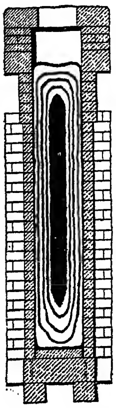
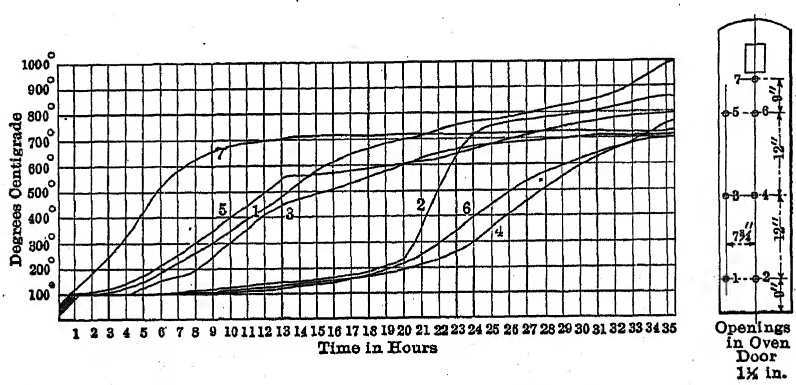


FIG. 141.—Progress of coking in retort oven.

The destructive distillation in a retort proceeds gradually from the hot walls to the center of the charge, as indicated[1] in Fig. 141, the largest evolution of gas occurring in the second and third hour. The rise of temperature in the retort is shown in Figs. 142 and 143[2] which represent measurements made in an Otto-Hoffmann oven (§131) at Sydney, N. S. The seven curves form three groups. Curves 1, 3 and 5, near the heating walls, show a gradual and continuous rise of temperature. In the retort a crust of coke forms quickly near the walls and the heat must penetrate this crust which, being a poor conductor of heat and increasing in thickness, retards the coking as it progresses toward the center. Curves 2, 4 and 6 in the center of the charge show that for one-half of the entire coking period a low temperature of from 100 to 200 ° C. prevails and prove that during that time the center will not only not lose any volatile matter, but will act as a condenser of heavy volatile hydrocarbons. As the two coking-zones approach each other, the temperature will rise, and meeting, will drive off the last of the volatile matter. Curve 7 shows that the temperature of the top of the charge, which is not in contact with any heating surface, rises comparatively quickly to a maximum and then remains constant. Similar data have been



FIGS. 142 and 143.—Rise of temperature in retort.

obtained by Hilgenstock[3] and Hüssener[4] in their ovens. A cake of retort

[1] *Circular*, United Otto Coke and Gas Co., 1906, p. 71.

[2] *Min. Ind.*, 1902, XI, 160; *Tr. A. I. M. E.*, 1903, XXXIII, 776.

[3] *Stahl u. Eisen*, 1904, XXIV, 449.

[4] *J. I. and St. I.*, 1904, I, 188; *Tr. Inst. Min. Eng.*, 1906, XXXI, 460.

coke[1] then shows, Fig. 144,[2] a vertical division plane through the center which is split into branches toward top and bottom; it will also be fissured more or less at right angles to the heating planes; again, the coke near the heating surfaces which has been exposed a longer time to a high temperature will be denser and harder than that near the center, where will be found the "black butts" made on the bottom of the beehive. If the temperature in coking is kept low, the heavy volatile hydro-carbons will not be satisfactorily decomposed; too large a portion will be condensed in the center and form there a soft and spongy coke. In coking, part of the H_2O in the coal passes off as vapor, especially at the beginning of the process; later, most of it is decomposed by C giving a mixture of H, CO and CO_2, the CO_2 of which is reduced by incandescent C to CO.

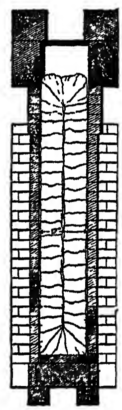


FIG. 144.—Cake of retort-oven coke.

The gases and vapors at first set free by the contact of coal with hot walls are rich in C_xH_y. They pass upward rapidly in the space left open by the shrinking of the coal and are prevented by the presence of moisture from being decomposed to any extent into C and CH_4. As coking progresses, they are more and more decomposed into C, CH_4 and a small decreasing amount of C_mH_n; a small portion, however, remains unchanged and is collected in the form of tar. This is formed especially during the first or cool period of coking. In general, a low temperature (800° C.) in coking gives much tar and little gas; the reverse is the case with a high temperature (1,200° C.). The N of the coal passes off either unchanged or partly in the form of N_3H; its diminution is quite uniform during the entire process. Mayer-Altmayer[3] found that NH_3 is slowly decomposed at elevated temperatures (>900°C.) and especially when present only in small amounts as is the case with coke-oven gas. The yield depends upon the character of the coal, the amount present in the gas, and the time NH_3 remains in contact with the incandescent charge or hot retort-walls; it is highest at about 800° C. The subject has been recently discussed by J. R. Campbell.[4]

From what has been said, it is clear that it is important to keep the temperature of a retort uniform, as only with an even, correctly adapted temperature will the coke and by-products show the desired physical and chemical properties.

In 1897 a very complete investigation into the process of coking was made by Schniewind[5] in an Otto-Hoffmann oven at Glassport, Pa.[6], while testing the coal of the Dominion Coal Co., Cape Breton, which was to serve as and was later[7]

[1] *Stahl u. Eisen*, 1904, XXIV, 521.
[2] *Circular*, United Otto Coke and Gas Co., 1906.
[3] *J. Gasbeleuchtung*, 1907, L, 25, 49.
[4] *Proc. Eng. Soc. West. Pa.*, 1912, Vol. XXVIII, p. 328; *Met. Chem. Eng.*, 1912, X, 524.
[5] Hofman, *Eng. and Min. J.*, 1898, LXVI, 428, 458.
Schniewind, *Progressive Age*, Oct. 15, 1901; *J. für Gasbeleuchtung*, 1902, XLV, 125, 141.
[6] *Iron Age*, Nov. 11, 1897.
[7] Part of the Cape Breton coal has been replaced (1913) by coal from W. Va.

the coal used by the New England Gas & Coke Co. for making illuminating gas for the city of Boston, and coke for boiler-, domestic-[1] and blast-furnace use. While the results obtained refer strictly to a special coal, form of oven and mode of working, the data furnish an excellent illustration of what generally takes place in a retort-oven. The block in which the tests were made contained 30 retorts, 33 ft. long, 5 ft. 10 in. high and 20.75 in. wide. Of these, one was reserved for the tests, being heated by the gas of the other 29. The coal had the following ultimate composition: C 75.10, H 3.75, N 1.51, O+S 13.80, ash 5.84; total 100.00. The proximate analysis gave: V.M. 34.60, F.C. 59.56, ash 5.84; total 100.00. The average weight of a charge was 14,591 lb., which, deducting 9.9 per cent. H_2O, corresponds to 13,602 lb. dry coal. The average coking time was 33 hr. 54 min. A summary of the results obtained per long ton of coal, the volumes having been reduced to weights, is given in Table 118.

TABLE 118—PRODUCTS FROM COKING IN A RETORT-OVEN

Products from 1 long ton of coal	Pounds	Per cent.
Coke, total { Large coke, >1″, 66.69 per cent. Small coke, ½—1″, 1.64 per cent. Breeze, <½″, 2.80 per cent. }	1,593.4	71.13
Tar	75.7	3 38
Ammonia (=1.373 per cent. sulphate)	7.6	0.34
Gas, total, 10,390 cu. ft. of 0.466 sp. gr.	368.0	16.43
Sulphur compounds in gas:		
Hydrogen sulphide (H_2S), 0.98 lb. per 1,000 cu. ft.	10 8	0.48
Carbon disulphide (CS_2), 0.13 lb. per 1,000 cu. ft.	1.6	0.07
Gas liquor and loss, by difference	182.9	8.17
Total	2,240.0	100.00

Of the 10,390 cu. ft. of gas, 49.5 per cent. was surplus gas, that is, gas not required for the heating of the oven. This had the following composition:

Olefines, C_mH_n,	5.2	Marsh gas, CH_4,	38.7	Hydrogen, H_2,	38.4
Carbon monoxide, CO,	6.1	Carbon dioxide, CO_2,	3.6	Oxygen, O_2,	0.3
Nitrogen, N_2,	7.7	Total,	100.0		

Its calorific power (H_2 burnt to liquid water) was 686 B.t.u., its candle-power 14.7, and its specific gravity (air=1) 0.510.

The coke, a fair blast-furnace fuel, showed: V.M. 1.27, F.C. and S 69.82, ash 8.91; total 100.00, P 0.0041, H_2O 3.67. The tar, sp. gr. 1.170, gave upon fractional distillation the products shown in Table 119.

[1] *Eng. and Min. J.*, 1906, LXXXI, 1143.

TABLE 119.—FRACTIONAL DISTILLATION OF COKE-OVEN TAR

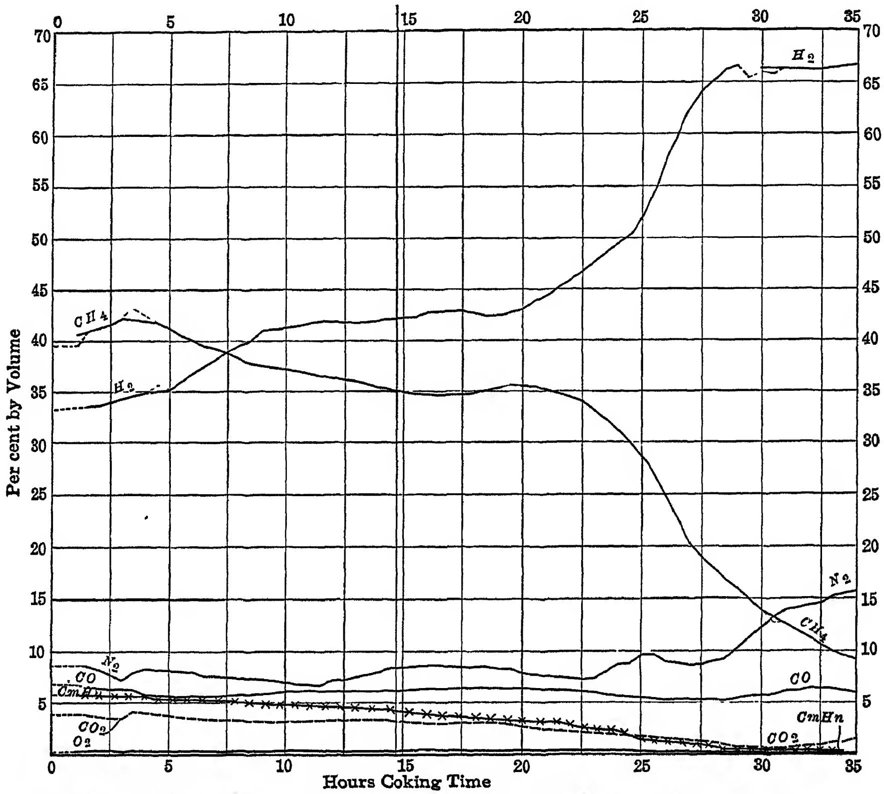
Fractions	Temperature, C°	International Coal
Light oil	80–170	3.7
Middle oil	170–230	9.8
Heavy oil	230–270	12.0
Anthracene oil	over 270	4.3 } 71.3
Pitch	......	67.0 }
Water	......	2.3
Loss	......	0.9
Total	......	100.00
Sp. gr.	......	1.170

Of the NH_3 in the gas liquor 96.6 per cent. was present as free NH_3. As the ultimate analysis had given 1.51 per cent. N, and as the 0.34 per cent. NH_3 recovered contains 0.28 per cent. N, 18.5 per cent. of the N of the coal had been converted into NH_3 (13 to 15 per cent. is a common figure). Of the three accompanying diagrams, Fig. 145 gives the average analyses of the gases of 4 charges, samples having been taken and analyzed every 2 hr.; Fig. 146 the calorific power, specific gravity and candle power; and Fig. 147 the total volume of gas obtained from one long ton of dry coal, its calorific power, and the relation of this calorific power to the total amount of heat required for coking.

Fig. 145 is a graphical representation of the average analyses of the gases from the four separate charges. The percentage of marsh-gas (CH_4) shows a rapid decline, and this is especially the case toward the end of the coking operation. It is accompanied by a corresponding increase of hydrogen (H), which reaches nearly 67 per cent., or nearly 80 per cent. if figured for gas free from oxygen and nitrogen. The curve of carbon monoxide (CO) shows the percentage of this gas to vary between the narrow limits of 5.4 and 6.8 per cent., a very low figure. The olefines (C_mH_n) remain at a constant figure, 6 1/2 per cent. for the first 5 hr., and then diminish at a uniform ratio to the end of the operation. The percentage of carbon dioxide (CO_2) is constant for the first 20 hr., ranging at from 3 to 4 per cent., and then decreases gradually to 1 per cent. Oxygen (O) and nitrogen (N) are accidental impurities. The oxygen gets into the gas through leaky joints and amounts to about 0.3 per cent. The average percentage of nitrogen during the first 14 hr. and 46 min. is seen to be 7.7 per cent.; that of the last 19 hr. and 10 min., 9.1 per cent. It remains pretty constant up to the twenty-third hour, and then increases rapidly. It has its origin in the destructive distillation, which accounts for 2–3 volumes, and in leakages which permit the entrance of air. The large increase toward the end is explained by the diminished volume of gas generated in the oven

and the consequent decrease of pressure. The percentage of nitrogen is high, and does not exceed 5 per cent. under normal working conditions.

Fig. 146 gives the calorific power, the specific gravity, and the candle power of the gas. The calorific power at the beginning is about 685 B.t.u. It rises quickly to 775 B.t.u., then drops quickly until the seventh hour, and more slowly until the twenty-second hour, after which it drops more quickly than at first until the end of the operation. The curve of the specific gravity runs almost



Surplus gas produced during the first 14 hours, 46 minutes. Volume per long ton, 5,143 cu. ft. = 40.5 per cent Average analysis

$CmHn$	CH_4	H_2	CO	CO_2	O_2	N_2	Total
5.2	38.7	38.4	6.1	3.6	0.3	7.7	100.0

Oven-heating gas produced during the remaining 19 hours, 10 minutes Volume per long ton, 5,247 cu. ft. = 50.5 per cent. Average analysis

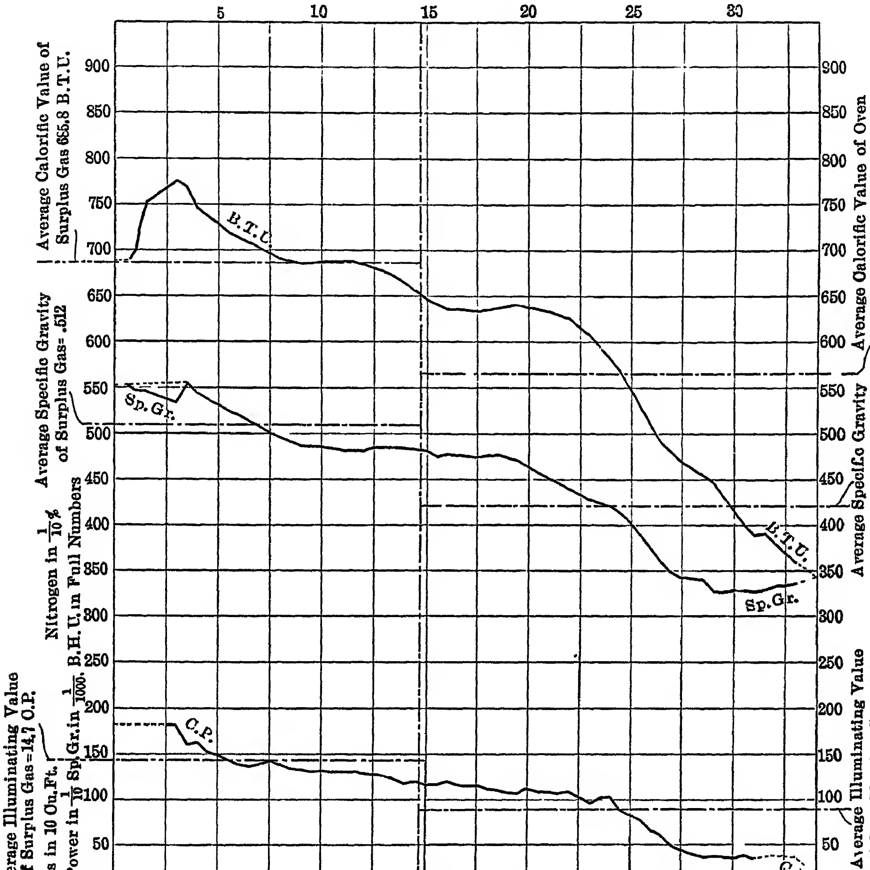
$CmHn$	CH_4	H_2	CO	CO_2	O_2	N_2	Total
2.4	29.2	50.5	6.3	2.2	0.3	9.1	100.0

FIG. 145.—Composition of gas evolved in coking

parallel to that of the calorific power, except that the drop is not quite so rapid. Toward the end the specific gravity is slightly higher, which is due to the increase in the percentage of nitrogen. The candle power is seen to drop rapidly during the first seven hours, then slowly to the twenty-fourth hour, when the decline is again very rapid, so that toward the end the gas has no illuminating value whatever.

Fig. 147 gives a graphical representation of the total volume of the gas

obtained from 1 long ton of dry coal, its calorific power, and the relation of this to the total amount of heat required for coking. The volume of gas obtained is seen to remain almost constant during the first 24 hr.; in the twenty-second and twenty-third hours there is an increase in volume, followed by a rapid



Surplus gas produced during first 14 hours, 46 minutes:

Average calorific value	685.8 B.T.U.
Average illuminating value	14.7 C.P.
Average specific gravity	0.512
Volume per long ton, 5,143 cubic feet =	49.5 per cent.

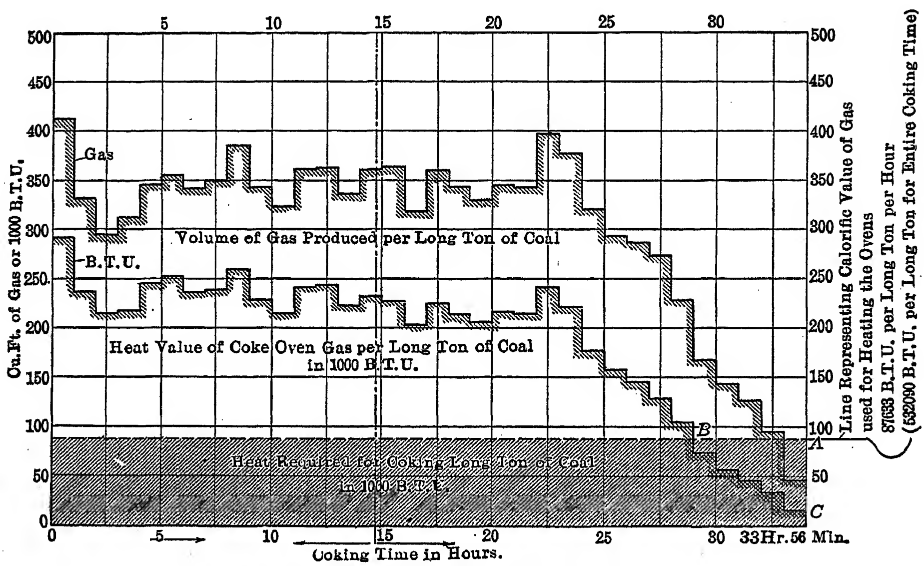
Oven-heating gas produced during remaining 19 hours, 10 minutes:

Average calorific value	566.7 B.T.U.
Average illuminating value	9.0 C.P.
Average specific gravity	0.421
Volume per long ton, 5,247 cubic feet	50.5 per cent.

FIG. 146.—Calorific power, specific gravity, and candle power of gas evolved in coking.

decline. The increase is due to the fact that the last thin layer of coal in the center of the oven is heated from both sides instead of only from one side, as was the case up to that time, the coal having received its heat chiefly from the

side wall contiguous to it. The sudden drop is explained, of course, by the fact that the last of the coal has been heated to the coking temperature. The heat value of the gas is seen to keep pace pretty evenly with the volume. Combining the results of the three diagrams, three periods in the process can be distinguished. The first period, lasting 9 hr., is one of declining value of gas. The percentage of marsh gas diminishes; that of hydrogen grows; the calorific power is reduced from 775 to 685 B.t.u.; the specific gravity drops from 0.550 to 0.490 and the candle power from 18 to 13 1/2. The second



Surplus gas produced during first 14 hours, 46 minutes.
Volume 5,143 cu.ft. = 49.5; per cent:- Calorific value 3,527,320 B.T.U. = 54.2 per cent.
Surplus gas produced during remaining 19 hours, 10 minutes.
Volume 5,247 cu.ft. = 50.4; per cent;- Calorific value 2,973,680 B.T.U. = 45.8 per cent.

FIG. 147.—Volume and heat value of gas evolved in coking, and heat required for coking.

period, reaching from the ninth to the twenty-second hour, produces gas of almost constant quality. The percentages of marsh gas and hydrogen are very little changed; the calorific power, specific gravity, and candle power are pretty constant. The third period is again one of declining value of gas. The percentage of marsh gas diminishes rapidly, with a corresponding increase in hydrogen; the calorific power, specific gravity, and candle power show a rapid decline. The decrease in volume is very gradual up to the twenty-second hour, when (for reasons given above) there is a small increase in the twenty-third hour which is followed by a rapid decline. The gas of the first period, after purification, can be advantageously used for illuminating purposes; that of the second period for heating the ovens, and that of the third period, after having been purified and enriched with benzole or oil vapors, or with oil gas, can be mixed with that from the first period. This last gas, being so

rich in hydrogen, is especially adapted to serve as a carrier of hydro-carbon vapors.

A question still to be settled is, How much of the gas is required to heat the ovens? It was stated at the beginning that the test oven had been heated with the gas from the other 29 ovens of the block. The amount of gas required per charge of 13,602 lb. dry coal was 36,169 cu. ft. As its calorific power was 499.2 B.t.u., the amount of heat required was $36{,}169 \times 499.2 = 1{,}805{,}564$ B.t.u. The heat consumption of 1 long ton for the entire coking time (33 hr., 56 min.) was therefore 2,973,680 B.t.u., or of 1 long ton for 1 hr., 87,633 B.t.u., further the heat consumption per oven per hour, 532,090 B.t.u. This is graphically represented in Fig. 147 by the shaded area. This figure shows that up to and including the twenty-ninth hour the heat value of the gas produced is greater than that required for coking, but after that time there is a lack of heat which would have to be supplied in case the ovens were exclusively heated by coke-oven gas and blast-furnace coke was to be produced. If the coke is to be used for domestic purposes or as a boiler fuel, a softer coke is more desirable and the coking can be advantageously interrupted at the twenty-ninth hour and the amount of gas represented by the area *A B C* saved. With ovens 18 in. wide instead of 20 3/4 in., this can be done even sooner.

Retort-ovens, as stated on page 237, can be divided into two classes. With one class (Non-Recovery Ovens) all the gases and vapors set free in coking pass with their initial heat directly into the heating-flues and are burnt. With the other class, the volatile matter is first cooled to recover the condensable part as a by-product (By-product Recovery Oven), and only the more permanent gases, as much as may be necessary, are returned to the heating-flues. A special method of handling the volatile matter with by-product ovens is that inaugurated by Schniewind in 1897 at the Everett gas works, Boston, in which the volatile matter set free during the first coking period (rich in illuminating gas C_xH_y and low in H) is purified by itself and furnishes illuminating gas, while that of the second period (rich in H and low in illuminating gas C_xH_y) after purification serves for heating the retorts. Cooling the gases to 15–20° C. represents a loss in heat amounting to 40 per cent.[1] of the total heat of the gases as they leave the oven. However, the by-products recovered and the additional yield of coke more than make up for this loss when there is a suitable market for tar and ammonia. Most modern retort-ovens have by-product plants. The additional yield of coke in by-product over non-by-product plants is due to the fact that the ovens of the former class are run with a slight internal pressure of gas which excludes any access of air by leakage, while those of the latter are run with a slight vacuum.

130. Retort-ovens with Vertical Chambers. THE APPOLT OVEN.[2]—The

[1] Lunge, *Min. Ind.*, 1896, v, 182.

[2] Percy, *op. cit.*, p. 444.

Weeks, *op. cit.*, p. 93.

Jordan, "Album to Course of Lectures on Metallurgy," Baudry, Paris, 1877, full drawings.

Appolt, *Ann. Min.*, 1858, XIII, 417.

Jullien-Lan, *Bull. Soc. Ind. Min.*, 1858, IV, 24, 45.

oven, represented in Fig. 148,[1] is an oblong block (49 ft. 4 in. long, 23 ft. wide and 23 ft. 7 in. high) containing 12 to 18 pyramidal vertical fire-brick retorts *a* (48×18 in. at base, 44×13 in. at top, 13 ft. long, walls 4.75 in. thick) which are spaced 7.75 to 9.75, av. 8 in. apart, strengthened by binders *d*, and surrounded by combustion chambers. The charging hoppers are closed by lids *c* and the discharging openings by hinged cast-iron doors *b*. The side-walls of a retort rest on cast-iron frames which are supported lengthwise by brick arches. In the vault underneath them run cars to receive the coke. The space between fire-

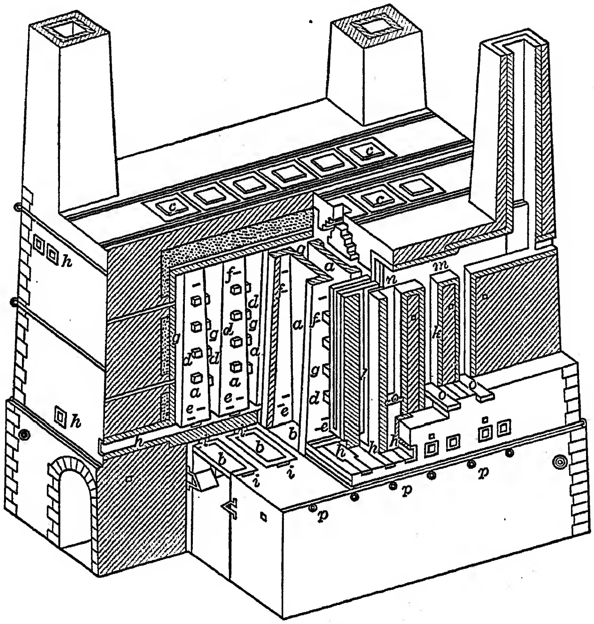


FIG. 148.—Appolt oven.

brick combustion chamber and outer wall of red brick is filled with sand or other loose poor conductor of heat to diminish the loss of heat by radiation and to provide for an expansion space. The volatile matter set free in coking passes out through slits *e* and *f* in the sides and ends of a retort into the flues *g*; it is burnt by air admitted from the bottom through *i* and from the side through *h*. The nine lower slits *e* (5×0.75 in.) on a side are situated 16 and 24 in. from the bottom; the three slits in the lower ends are similarly placed; there are three upper slits to a side. The products of combustion pass off through one or two sets of horizontal flues (not shown) into vertical flues *k* and *l*, ending in the main flues *n* and *m* leading to the four chimneys of a block. Rods *p*, square at one end to permit turning with a key, pass through iron pipes to the hinged doors *b*; dampers *o*

Rexroth-Ozier, 1859, v, 428, 593.
Baroulier, 1865, IX, 469.
Vériot, 1869, XV, 77.
Rossigneux, 1891, v, 450.

[1] Beckert, Th., Leitfaden zur Eisenhüttenkunde, Springer, Berlin, 1898, I, 62.

serve to regulate the draft; square peep-holes are placed above the air-inlets *h*. In some later furnaces[1] division walls have been introduced in order to permit a better control of the travel of the products of combustion and thus obtain a more even heating of the retorts. At Blanzy[2] the retorts have been built up with unburnt bricks and these then baked in place, an operation which takes about three weeks careful firing.

Firing a New Oven.—In starting a new oven, the retorts are gradually warmed and brought in eight to ten days to a temperature of 1,200 to 1,400° C. For this purpose a temporary grate is placed in the bottom of a retort and this lined with fire-brick for about 1 ft. to prevent clinkers from adhering to the sides. In firing, the gases are at first allowed to pass off through the charging door; later this is closed that the gases may pass through slits *e* and *f* into flues *g*, *m* and *n*, and heat them before passing off into the chimney. When the retorts are ready to be charged, the temporary grates and linings are removed and the iron trapdoors put in place, locked and covered with about 1 ft. of breeze to protect them and to prevent loss of heat. In order to have an even evolution of gas and thereby an even distribution of heat, it is important to charge the retorts alternately, which is characteristic of all retort-ovens with the possible exception of the Semet-Solvay oven (§132). In an 18-retort oven there are nine pairs of retorts; the order of charging would be, pairs 1, 3, 5, 7, 9, then paris 2, 4, 6, 8. Thus when pair No. 2 has been filled, it will have on the sides pairs Nos. 1 and 3 which, evolving gas freely, furnish all the heat necessary to start the coking until comes the turn of pair No. 2 to help out its neighbors. A charge of coal weighs about 3,000 lb. and reaches to within 3 ft. from the top. Coking lasts 24 hr. The coke is discharged by tilting the bottom. In order to prevent it from dropping with too much force into the iron receiving car, inclined cast-iron plates are built into the walls beneath the retorts which catch the red-hot coke as it glides out of the retort. It is then detached in pieces and run out of the vault and quenched. The retort is refilled from the top. The yield in coke is the same as with other non-by-product ovens.

The leading advantage claimed for this oven is a large heating-surface. A retort with a capacity of 68.1 cu. ft. has a total surface of 137.5 sq. ft. and a heating surface of 132.4 sq. ft. or 94.7 per cent., which is the largest figure extant. A charge of coal can, therefore, be coked quickly; further since the charge is high, there is considerable pressure which aids in making the coke dense. These facts make the oven suitable for coals that do not coke readily, as is the case with the coal of Belgium, Southwestern Prussia (Saarbrücken) and Southern France. On the other hand, the oven is expensive, as the cost of repairs is high owing to the high temperature and the necessity of cooling down the entire oven. The coke is not uniform as the retorts in the center are more strongly heated than those at the ends.[3]

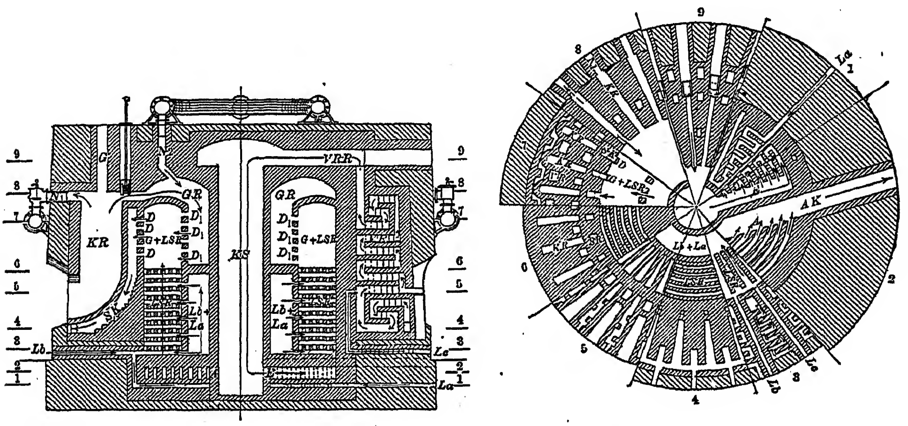
[1] Lürmann, *Stahl u. Eisen*, 1894, XIV, 383.

[2] Marle, *Bull. Soc. Ind. Min.*, 1893, VII, 317; *Coll. Guard.*, 1894, LXVII, 397; *Stahl u. Eisen*, 1894, XIV, 383.

[3] Remy, *Zt. Berg. Hütten. Sal. Wesen i. P.*, 1890, XXXVIII, 121.

Schorr[1] has constructed a by-product oven which in general arrangement and disposition of retorts resembles the Appolt type; two other forms are the Elliot-Jones[2] and the Woodall-Duckham.[3]

THE BAUER OVEN.[4]—This is a by-product oven. According to the form of the upright retort, three types are distinguished which have been constructed for treating coal having a high, medium and low percentage of volatile matter. The type for average coking coal is represented in Figs. 149 and 150.[5] Forty retorts *KR* are arranged around a central chimney; they are 10 ft. high, 6 ft. deep, 1 ft. 4 in. wide at bottom and 1 ft. wide at top; the bottom is curved to facilitate the discharge of the coke. A charge of two tons coal, to be coked in



FIGS. 149 and 150.—Bauer vertical coking oven.

20 hr., is dropped through hopper *G*; the volatile matter at first is drawn off through *N* into the main leading to the condensing plant; later it may be deflected wholly or in part to the right by raising the damper which shuts off the gas chamber *GR*. From the condensation plant the heating gas returns through six pipes N_2 to the annular gas chamber *GR*, passes through flues *D′* into the combustion chamber *G+LSR* where it meets air superheated in the checker work *LSR* and burns. The products of combustion pass through flues *D* and descend through *SK*. At the bottom of the retort the current is split into two branches. These ascend in the heating chambers *VR* on the sides of a retort, until they reach the flue *VRR*, where they combine and pass downward through the common central flue *KS*, enter at the right the circular horizontal flues *E* and, having made the circuit, pass off into the chimney. Cold air enters through flues *La*, *Lb* from which small ports lead to the checker work

[1] *Eng. and Min. J.*, 1905, LXXX, 1115.

[2] Futers, *Tr. Inst. Min. Eng.*, 1910–11, XL, 499.

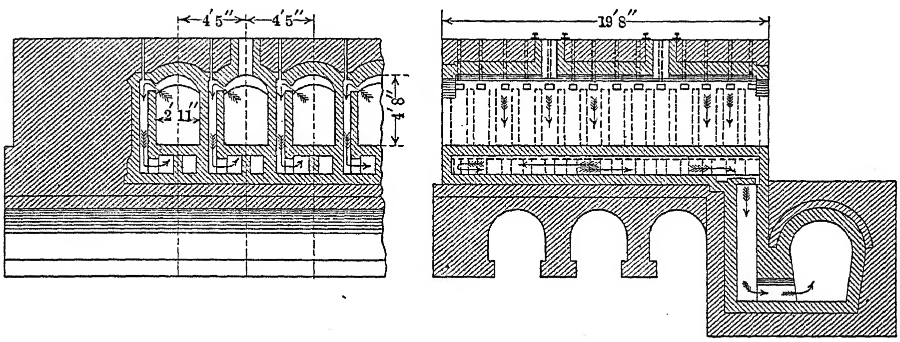
[3] Corresp., *Coal Age*, 1911, I, 314.

[4] *Oest. Zt. Berg. Hüttenw.*, 1885, XXXIII, 17; *Berg. Hüttenm. Z.*, 1887, XLVI, 379; *Glaser's Ann.*, 1887, XXI, 93; *Engineering*, 1887, XLIV, 500; *Eng. and Min. J.*, 1890, L, 72; *Bull. Soc. Ind. Min.*, 1891, V, 532.

[5] *Berg. Hüttenm. Z.*, 1887, XLVI, plate X, Figs. 19 and 20.

LSR of each retort; an additional air-inlet *Lc* is provided for the combustion chamber *VR*. The ovens are in operation in France,[1] England and Germany.[2]

131. Retort-ovens with Horizontal Chambers and Vertical Flues.[3] THE FRANÇOIS-REXROTH OVEN.[4]—This, the first oven with horizontal chambers and vertical heating flues, was built about 1850. The retort, Figs. 151 and 152 (20 to 30 ft. long, 4 to 5 ft. high and 24 to 36 in. wide according to the character of the coal) receives a charge of 5 to 8 tons of coal through two hoppers in the roof. In coking, all the volatile matter passes off at one side through openings just beneath the spring of the arch, where it meets the air drawn in through vertical flues in the roof and burns. The products of combustion, descending through the vertical flues, heat first the sides of the retort, then the bottom along which



FIGS. 151 and 152.—François-Rexroth retort-oven.

they make two passes, and are collected in the main flue leading to the chimney. The coke is discharged by means of a pusher similar to the one shown in Figs. 161 and 165. The coking time varies from 24 to 48 hr. according to the weight of the charge and the character of the coal. The furnace has found favor mainly in the Saar district, Rhenish Prussia, where in 1905 over 1,000 ovens were in operation. In other districts it has been largely replaced by the Coppée oven which, although more complicated in construction, forms an improvement.

THE COPPÉE OTTO OVEN.[5]—The first Coppée oven was erected in Belgium

[1] *Engineering*, 1887, XLVII, 500.

[2] *Chemiker Z.*, 1898, XXII, 993.

[3] Herbst, *Internat. Congr. Min. and Met.*, Düsseldorf, 1910, vol. Praktisches Hüttenwesen, p. 115.

[4] Huessener, *Stahl u. Eisen*, 1883, III, 399.
Remy, *Zt. Berg. Hütten. Sal. Wesen i. Pr.*, 1890, XXXVIII, 123.
Simmersbach, *Stahl u. Eisen*, 1905, XXV, 1349.
Fähndrich, *Zt. Berg. Hütten. Sal. Wesen i. Pr.*, 1911, LIX, 472.

[5] *Zt. Berg. Hütten Sal. Wesen i. Pr.*, 1887, XXXV, 303 (Simmersbach); 1890, XXXVIII, 123 (Remy); 1891, XXXIX, 203 (Deicke); 1896, XLIV, 409 (Simmersbach); *Stahl u. Eisen*, 1883, III, 399 (Huessener); *Bull. Soc. Ind. Min.*, 1891, V, 445 (Rossigneux); 1903, II, 796, 802 (Hurez); *J. I. and St. I.*, 1894, II, 211 (Soldenhof); *Min. Ind.*, 1895, IV, 221 (Blauvelt); Evence-Coppée Oven, *Coll. Guard.*, 1910, XCIX, 267.

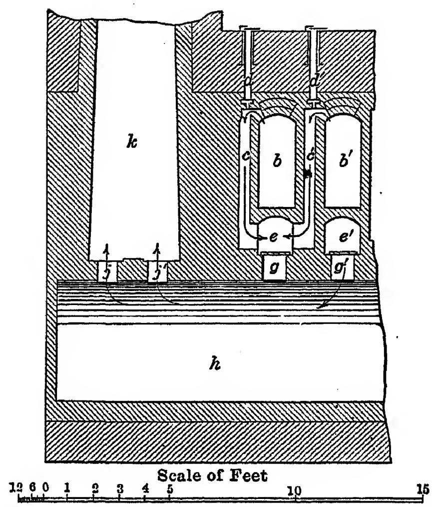
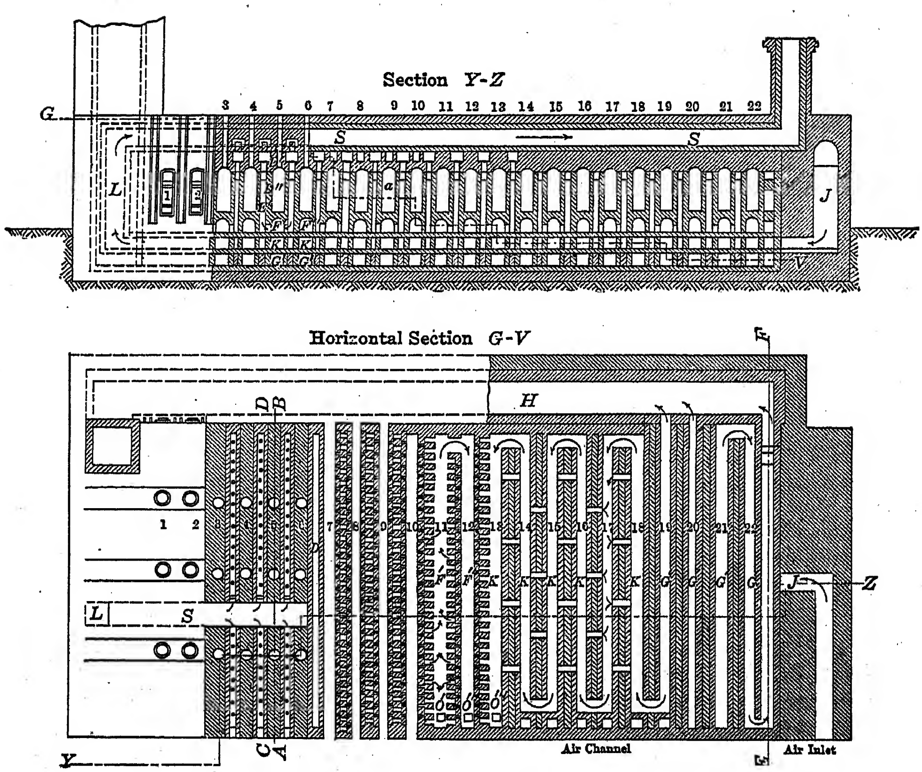


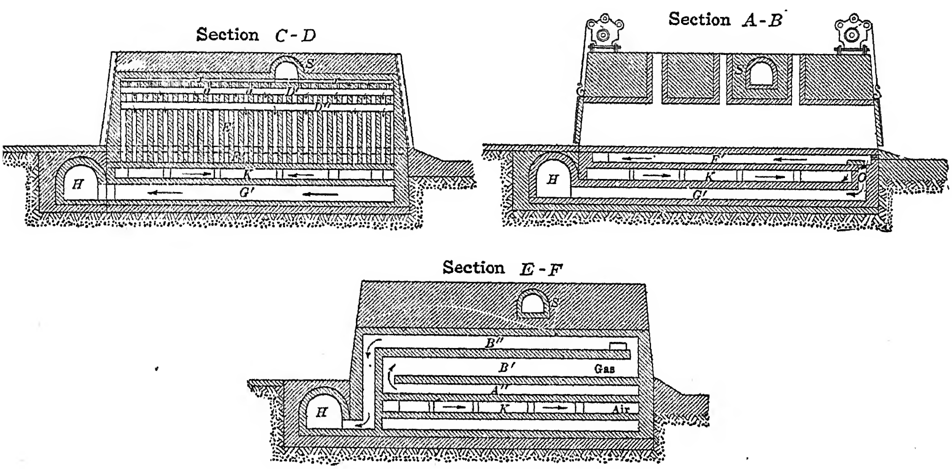
FIG. 153.—Principle of the Coppée retort-oven.



FIGS. 154 and 155.—Coppée retort-oven.

about 1860[1] for coking the lean coals of that country. To-day it is probably the leading vertical-flue type of oven in which the by-products are not recovered. In the 50 years of its existence it has undergone many changes. The leading characteristic of the oven is that the retorts are worked in pairs, 15+ to a block. One retort is charged when the content of the other is half-coked; the gases from the retorts bb', Fig. 153,[2] are burned in their respective vertical flues cc', combine, travel along flue e heating the bottom of b, return along e' heating the bottom of b', and pass off into the main flue h and through ports j into the stack k. By thus combining the gases of a pair of retorts, the heating becomes more uniform.

An improved form of the Coppée-Otto oven[3] is shown in Figs. 154 to 158. It consists of a block of 22 horizontal retorts, Fig. 154 (30 to 33 ft.



FIGS. 156 to 158.—Coppée retort-oven.

long, 5 ft. 6 in. to 6 ft. high, 23 to 27 in. wide), holding 5 to 6.5 tons of coal which are coked in 48 hr. The fronts and backs of the retorts are closed by heavy cast-iron brick-lined doors, Fig. 157, raised and lowered by means of a traveling windlass. The end-retorts have strong buttress-walls. The space between the tops of the retorts and the top-course of red brick is filled with a layer of sand or ashes to diminish any loss of heat. A retort has three charging hoppers fed from coal buggies running on corresponding tracks. The volatile matter passes off near the top of the retort through 28 openings on one side into the horizontal flue D', Figs. 155 and 156, where it meets superheated air from flue S, Fig. 156, distributed by branch-flues x and x', and is burnt in D' and D''. The products of combustion descend in the vertical flues E' into the bottom flue F' of the odd-numbered retorts and return through the others, F'', having an even number. In the flues F' F'' . . . are openings

[1] Steavenson, *North Engl. Inst. Min. Eng.*, 1872–73, XXII, 82.
[2] Percy, "Fuel," part of Fig. 6, plate IV.
[3] *Zt. Berg. Hütten Sal. Wesen i. Pr.*, 1896, XLIV, 409.

$O' O''$ . . . , Figs. 155 and 157, having dampers. The gases pass from $F' F''$ . . . through $O' O''$ . . . into the flues, $G' G''$ leading into the main flue H, Figs. 156 and 158, which leads either to the stack or to a set of boilers to utilize the rest of the available heat. The air required for burning the gases is admitted through the flue J in one of the buttress-walls, Figs. 154 and 155, passes zig-zag through the recuperator chambers K and leaving these heated to 320 to 430 ° C., ascends the hot-air flue L and enters the horizontal flue S referred to above. It will be noted that by this arrangement the outer side-wall of retort No. 22 is not heated. In order to accomplish this, the gas returning under retort No. 22, instead of passing downward, Fig. 157, into flues G' and G'', ascends, Fig. 158, in the horizontal zig-zag flues A'' B' B'' and then descends into the main H.

A new oven is dried[1] and warmed with a coal fire, about 3 ft. 3 in. deep and 20 in. high, built at the front and back of each retort. The necessary air enters through open spaces made by supporting the doors with brick placed on edge. The oven is dry when no more vapor escapes through flue H, *i.e.*, after about ten days. Lump coal is used in firing. It is charged through the feed-hoppers, and the depth of the bed is raised daily about 4 in. The ashes are removed every other day. The position of the doors is the same as in drying and warming, only the doors are carefully luted at the sides. The draft is checked as much as possible so as to keep the heat in the oven. When the side walls have become bright red, a charge filling two-thirds of the retort is given, and the amount of coal gradually increased to the full charge of 6.5 tons. Coking takes 48 hr. In regular work one of the two retorts working together is charged 24 hr. ahead of the other.

The charge is leveled[2] with long-handled rakes through openings (see retorts 1 and 2 in Fig. 154) in the doors, which are then luted. Black smoke issues from the top of the coal and passes off into D' and D'' where it is burnt. When the charge has been coked, the doors (Fig. 157) are raised with the windlasses and the coke is discharged by a ram onto the wharf, where it is quenched (5 gal. water per ton of coke) and pulled apart along the median line, if it does not fall apart of its own accord. The variations in temperature with an oven not having flues $G' G''$ have been found to be only 200° C., viz., 1,100 to 1,300° C. in the side-flues and 1,300 to 1,500° C. in the bottom flue. With ovens having flues G' G'' the variations are smaller. In the Ruhr district, Prussia, a charge of 6.5 tons of coal with 15 per cent. H_2O is coked in 48 hr. giving 75 to 77 per cent. blast-furnace coke containing about 4 per cent. H_2O. The advantages of the Coppée[3] oven are the shortness of vertical flues which permits making the retort walls thin (3 1/2 in.), the simple construction of the flues, the high temperature, and

[1] Deicke, *Zt. Berg. Hütten Sal. Wesen i. Pr.*, 1891, XXXIX, 203.

[2] Mechanical Levelers: Thau, *Glückauf*, 1908, XLIV, 1149.
Seaver, *Iron Age*, 1908, LXXXII, 302; *Iron Trade Rev.*, 1908, XLII, 203.
Herbst, *op. cit.*, p. 145.

[3] Fulton, *op. cit.*, 294, 379.
Rossigneux, *Bull. Soc. Ind. Min.*, 1891, V, 449; *Eng. News*, March 5, 1908.

the large heating surface (64 per cent. of the total). The Bernard oven is similar to the Coppée oven of 1890.[1]

THE OTTO-HOFFMANN OVEN.[2]—This by-product oven is essentially a Coppée-Otto oven to which Hoffmann has added Siemens regenerative chambers in order that the heat lost by the cooling of the gases and by the withdrawal of the by-products may be made up by preheating the air necessary for burning the gases. The products of combustion are passed through the regenerators instead of their sensitive heat being utilized under boilers as is the common practice with non-regenerative systems. Since the first ovens went into operation in 1883,[3] the Otto-Hoffmann system has undergone several changes in the constructive details which resulted in 1891 in a type which has become the standard. The modifications of 1896[4] have been given up again. In the United States the first plant with 60 ovens was built in 1894 at the works of the Cambria Steel Co., Johnstown, Pa. The unit plant consists of two blocks of ovens, each with 30 retorts; this number was raised to 50 in 1905, and to a higher number since then. The oven of 1891[5] is represented in Figs. 159 and 160. The Siemens regenerators R and R' are placed in vaulted chambers beneath the fronts and backs of a row of retorts; the space between them is supported by pillars and arches. As the regenerators are usually built on the ground floor, the ovens will be elevated 13 ft. or more above it. A regenerator, R or R', is divided into two parts by a double arch having ports t t'; the lower part has a longitudinal partition wall which forms the two flues S S'; the upper part R R' is filled with checkerwork. The flues s s' unite outside of the oven into a main flue which leads to a reversing valve directing the products of combustion into the stack and fresh air into the flue.

The coking chamber is 33 ft. long, 6 ft. 6 in. high and 17 to 22 in. wide with a taper of 1 to 4 in. toward the discharge end, and holds 6 to 7 tons of coal which are coked in 30 ± hr. It has three charging openings c, two ports d for withdrawing the volatile matter, and is closed front and back by cast-iron brick-lined

[1] *Zt. Berg. Hütten. Sal. Wesen i. Pr.*, 1891, XXXIX, 203.

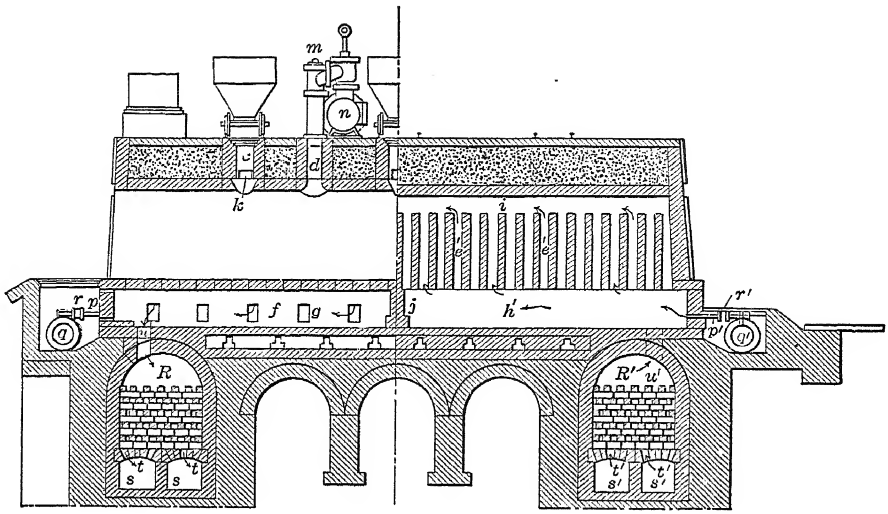
[2] *Stahl u. Eisen*, 1882, II, 311, 360, 505 (Otto); 1883, III, 397 (Huessener); 1884, IV, 396 (Otto); 1892, XII, 819 (Leistikow); 827 (Festner); 1894, XV, 1105 (Huessener); 1126 (Otto); 1895, XVI, 30 (Otto); *Zt. Berg. Hütten Sal. Wesen i. Pr.*, 1891, XXXIX, 203 (Deicke); 1896, XLIV, 411 (Simmersbach); *Bull. Soc. Ind. Min.*, 1891, V, 505 (Rossigneux); *Min. Ind.*, 1895, IV, 223 (Blauvelt); 1901, X, 143 (Schniewind); *Tr. A. I. M. E.*, 1903, XXXIII, 760 (Atwater); *Iron Age*, Sept. 6, 1894 (Editor); *Am. Mfr.*, 1898, LXII, 446 (Phillips); *Circular*, United-Otto System of By-product Ovens, New York, 1906.

Plants: Glassport, Pa., *Iron Age*, Nov. 11, 1897; Everett, Mass., *Tr. A. I. M. E.*, 1903, XXXIII, 766, 775; *Progressive Age*, Aug. 15, Sept. 1, 1899, Jan. 1, 1900; *American Gas Light Jl.*, 1901, LXXIV, 445; Buffalo, N. Y., *Mines and Minerals*, 1904, XXIV, 362; Camden, N. J., *op. cit.*, 1905, XXV, 610; *Iron Trade Rev.*, Feb. 9, 1905; *Mines and Minerals*, 1907, XXVII, 253; *Circular*, United-Otto System of By-product Ovens, New York, 1906.

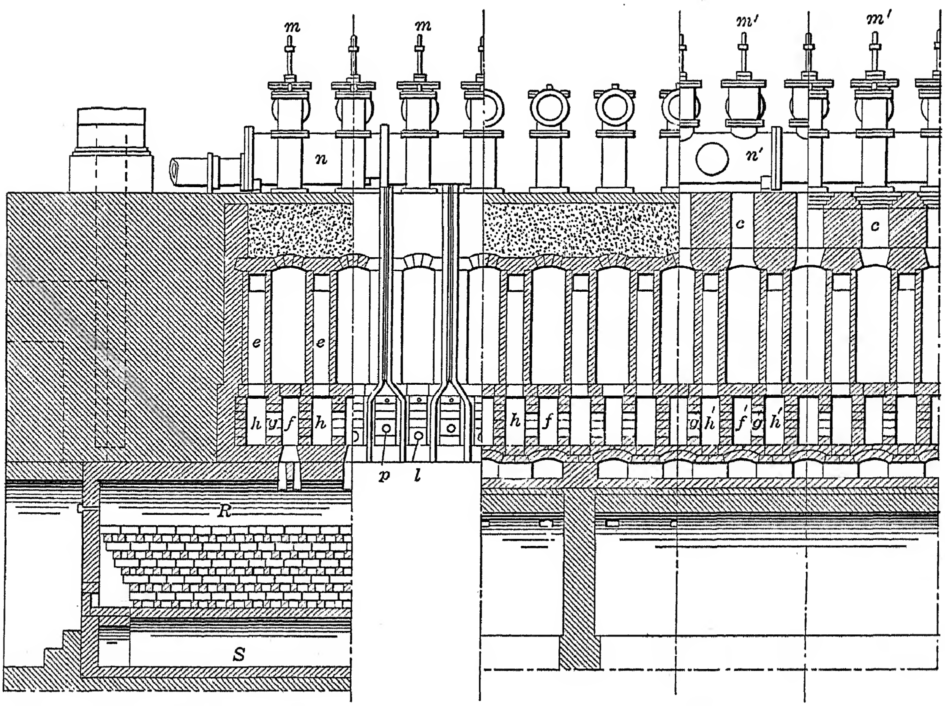
[3] *Stahl u Eisen*, 1884, IV, 396.

[4] *Zt. Berg. Hütten. Sal. Wesen i. Pr.*, 1896, XLIV, 411.

[5] *Zt. Berg. Hütten. Sal. Wesen i. Pr.*, 1891, XXXIX, 206; *Stahl u. Eisen*, 1892, XII, 818, *Min. Ind.*, 1895, IV, 223.



Longitudinal Sections



Section Through Regenerator. Elevation Transverse Sections.

FIGS. 159 and 160.—Otto-Hoffmann by-product oven.

doors[1] to be raised and lowered by a windlass. The coal which was formerly brought to the charging holes in three larries pushed by hand on separate tracks running along the top of the block, is at present handled by a single electrically operated larry with chutes suspended from a steel trestle (Fig. 165). On either side of a retort are 32 vertical heating flues *e* and *e′* which are connected at the bottom by horizontal flues *h* and *h′* through which the heating gas enters from pipes *p* and *p′*; hot air is admitted in the air-flues *f* and *f′* through openings *u* and *u′*, and passes through ports *g* and *g′* into the gas-flues *h* and *h′*. The silica-brick bottom flues *f* and *f′*, with peep-holes *l*, are therefore heated only by the superheated air, and the side-flues by the combustion of the gas. The bottom flue is divided by a partition wall *j* into equal halves. The heating gas arrives from the by-product plant alternately in the mains *q* and *q′* and enters the heating flues through pipes *p* and *p′* resp.; *r* and *r′* are regulating cocks. Supposing the gas to arrive in *q′* and to pass off through pipe *p′* into the horizontal flue *h′*. It

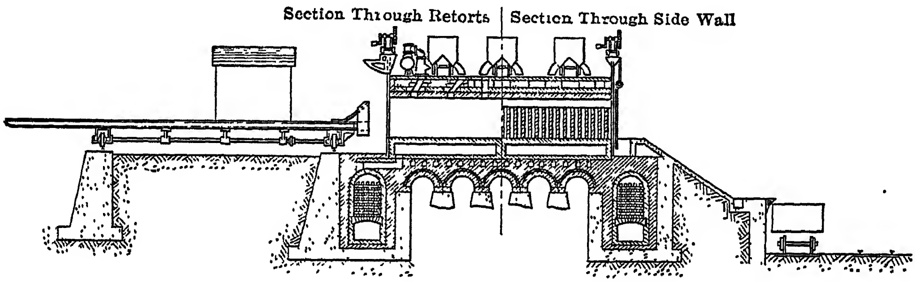


FIG. 161.—Otto-Hoffmann by-product oven with charging larries, pusher and quenching platform.

will meet the superheated air which has entered the bottom flue *f′* through openings *u′* and passed out through *g′*; the products of combustion will ascend in the vertical heating flues *e′*, pass through the horizontal flue *i*, descend in the vertical flues *e*, collect in the horizontal flue *h*, pass through ports *g* into flue *f* and thence through flues *u* into the regenerators *R*, the ports *t* and the flues *S* into the stack. After one-half hour gas- and air-currents are reversed and travel in the opposite direction.

The manner of discharging a retort is shown in Fig. 161. To the left (pushing-side or back) of the ovens is the coke ram consisting of a pushing face attached to a rack worked by a pinion. The frame carrying a boiler and a double reversing engine is carried by a truck. It takes 1 to 2 min. to push the ram through an oven. The coke falling on to the platform is quenched and loaded into cars. In some plants the quenching platform is replaced by a quenching car. The Moore coke quenching machine[2] aims to cool the coke with a minimum amount of water, to preserve the silvery luster, and to load

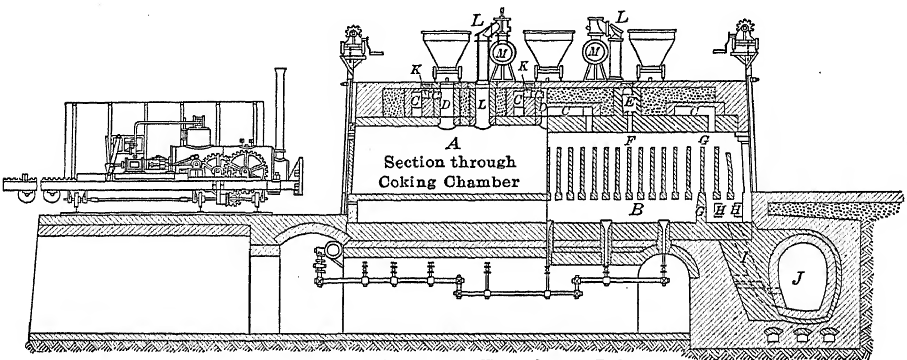
[1] Beach, Flued Coke-oven Doors, *Coll. Guard.*, 1906, XCII, 981.
Thau, *Glückauf*, 1907, XLIII, 998.

[2] Atwater, *Tr. A. I. M. E.*, 1903, XXXIII, 773.

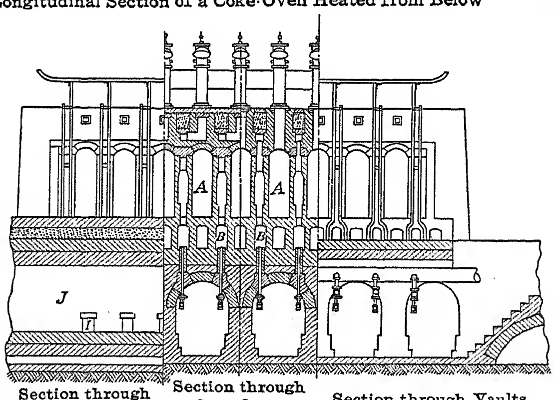
directly into railroad cars. In many modern plants an electric motor has replaced the steam engine.

The temperatures in an oven usually are: bottom flue 1,200 to 1,400° C.; side flues 1,100 to 1,200° C., regenerators at the start 1,000° C. and after reversing 720° C., stack 420° C.

STARTING A NEW PLANT.[1]—The ovens are warmed in the same manner as a Coppée plant (p. 260). In the charging openings *c* are small flues *k* which are closed in regular work. In starting, they are opened; the volatile matter passes



Longitudinal Section of a Coke-Oven Heated from Below



Cross Section through Coke-Ovens, Heated from Below

FIGS. 162 and 163.—Otto by-product oven.

off through them into the horizontal flue *i* where air is admitted through special ports. The products of combustion descend through the vertical flues *e* and *e'* into the regenerators *R* and *R'* and thence through a by-pass into the stack. When the retorts and regenerators have been sufficiently heated, the small flues *k* and the special air-ports are closed; the oven is then worked in the normal way. The volatile matter of each retort is drawn off through upright cast-iron pipes *m* and *m'* which discharge into their mains *n* and *n'*; this continues until

[1] Gillon, *Rev. Un. Min.*, 1912, XXXVIII, 106.
Thau, *Glückauf*, 1908, XLIV, 265.

the charge has been coked, when the valves between $m-m'$ and $n-n'$ are closed, the end doors of the retort raised and the coke discharged. The end doors are now lowered and luted; the oven is re-charged; the charging doors are closed, and the valves again opened. During the coking there is maintained in the retorts a slight pressure over the atmospheric.

THE OTTO OVEN.[1]—Figs. 162 and 163 show a radical departure in the manner of heating retorts with vertical flues. This change was made by Otto in his oven of 1896 by introducing the gas through large Bunsen burners beneath the flues. In Fig. 162, which is a longitudinal section through the coking and heating chambers *A* and *B*, the gas-main from the condensation plant is placed in a tunnel on the pushing-side of the oven. From it pipes branch off at right angles into vaulted passages underneath the heating flues and deliver the gas to eight Bunsen burners. Their flames spread in the lower combustion flue *B*, rise in the 32 heating flues to the collecting flue *F*, pass over the partition wall *G* and descend through *I* into the main *J*. As the flue underneath the coking chamber, Fig. 163, would not be heated at all, the partition walls between it and the adjoining heating flues *B* are perforated. In Fig. 162 *D* are charging openings, *L* gas-ascending pipes, *M* gas-collecting mains, *C* by-pass flues to be closed by blocks *K*, *E* a tile to be dropped upon *F* in starting up (similar to Coppée-Otto oven). A retort is 33 ft. long, 6 ft. 6 in. high to crown of arch, 20.75 in. wide in the middle (19.5 in. at back, 22 in. at front). The progress in coking has been investigated by Schmidt.[2]

THE OTTO-HILGENSTOCK OVEN,[3] shown in Fig. 164, is an improvement on the Otto oven, in which the chamber near *G* and the downward passage beyond *G* must become very much overheated, and the bottom of the retort underheated. In the Hilgenstock oven the products of combustion of four burners are combined and descend near the middle of the block through flue *H*, and, traveling through cross-passages into the flue *I* underneath the coking chamber *A*, furnish this with the necessary heat before they pass through *k* into the main *M*. Temperature and draft measurements made at Middlesbrough-on-Tyne gave for vertical flues 1,278 to 1,462° C. and 1.52 to 1.96 mm. water; for the lower combustion flue 1,215 to 1,312° C. and 1.79 to 2.10 mm.; for upper collecting flue 1,144 to 1,278° C. and 0.29 to 1.52 mm.; for gas main *C* 283 to 293° C.; for waste-gas flue 245° C. and 6.89 to 10.30 mm. water, and the air for combustion of gases was heated to 355–438° C. during its passage through the arches.

It will be noted that both the Otto and Otto-Hilgenstock ovens recover by-

[1] Simmersbach, *Stahl u. Eisen*, 1898, XVIII, 645.
Kroseberg, *Tr. Inst. Min. Eng.*, 1897–1898, XV, 402.
Lunge, *Min. Ind.*, 1896, V, 184.
Blauvelt, *Min. Ind.*, 1897, VI, 172.

[2] *J. für Gasbeleuchtung*, 1899, XLII, 24.

[3] Thiry, *J. I. and St. I.*, 1902, I, 32.
Schniewind, *Min. Ind.*, 1902, XI, 159.
Atwater, *Tr. A. I. M. E.*, 1903, XXXIII, 767.
Hurez, *Bull. Soc. Ind. Min.*, 1903, II, 790.

products, but have no regenerative chambers, the heat of the waste gases being utilized in waste-heat boilers. A modification[1] of this oven is in operation in the Saar district, Rhenish Prussia. In 1908, however, an Otto oven with regenerators was put into operation in Germany.[2] An oven similar in principle is the following:

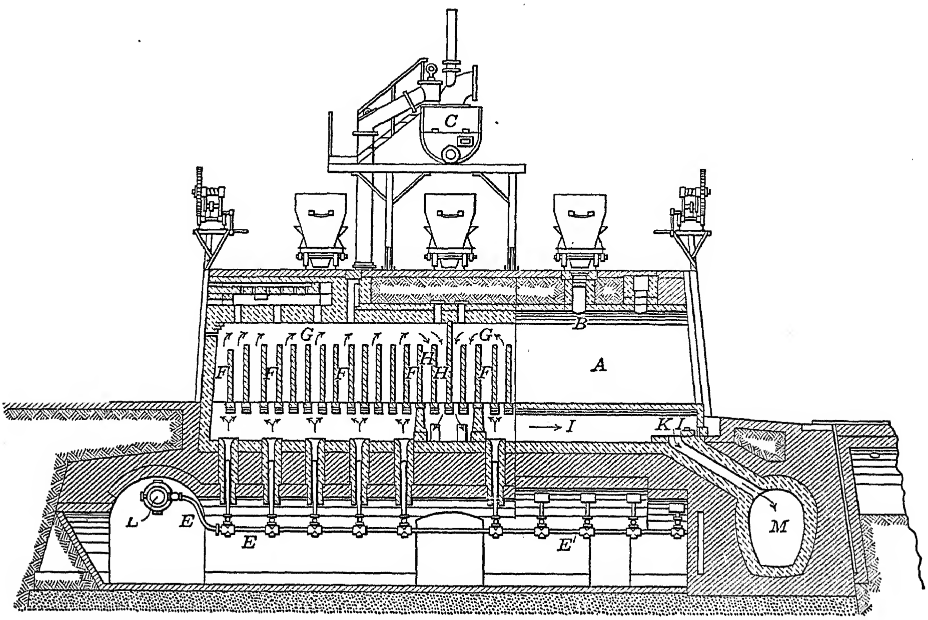
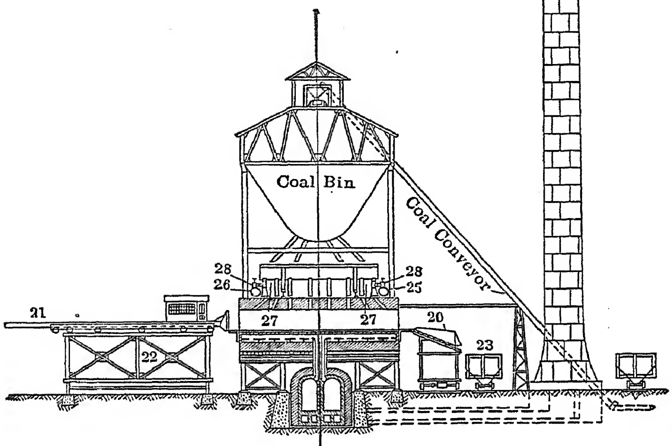


FIG. 164.—Otto-Hilgenstock by-product coke oven.

THE SCHNIEWIND OR UNITED-OTTO OVEN,[3] Figs. 165–167, combines the leading features of the Otto-Hoffmann oven and the new Otto system of heating, and shows improved methods of construction and of handling of materials. Figs. 165 and 166 show how the furnace block is supported by a steel-frame structure instead of resting upon brick arches, and how the regenerative chambers stand free without having to carry any part of the oven, two features characteristic of modern open-hearth furnaces. The section, Fig. 165, represents the handling of coal and coke. The coal is supposed to arrive by rail in cars on the right hand of the stack; it is raised by a conveyor to the stationary bin to be discharged through five spouts into an overhead traveling electric larrie having eight chutes for the coal to pass into corresponding charging openings of a retort. The volatile matter of a retort passes off at either end through uptakes 27, through valves 28 into mains 25 and 26 for rich and poor gas to the condensation plant. The coke is pushed by a ram, 21, resting on car 22, both of

[1] *Stahl u. Eisen*, 1905, XXV, 1388.
[2] *Zt. Berg. Hütten. Sal. Wesen i. Pr.*, 1908, LVI, 181.
[3] Schniewind, *Iron Age*, Nov. 28, 1901; *Min. Ind.*, 1901, X, 145.
Atwater, *Tr. A. I. M. E.*, 1903, XXXIII, 770.



Section A-A

FIG. 165.—Schniewind or United-Otto by-product oven.

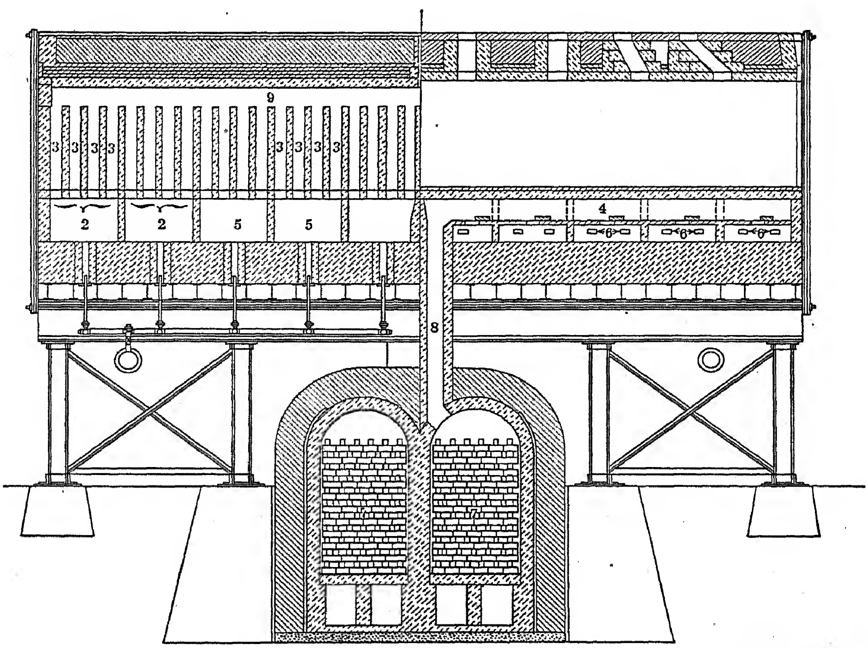


FIG. 166.—Schniewind or United-Otto by-product oven.

which are driven by electric motors. The coke is received in an inclined quenching car[1] 20 and discharged into the railroad car 23. The manner of heating the oven is shown in Figs. 166 and 167. The gas enters through five burners covering half the length of a retort, burns with superheated (980° C.) air in five combustion chambers 5, ascends through vertical flues 3, travels through the upper collecting flue 9, descends through the flues of the five chambers covering the other half of the length of the retort. It then passes from chamber 5 into chamber 4 situated underneath the retort and thence,

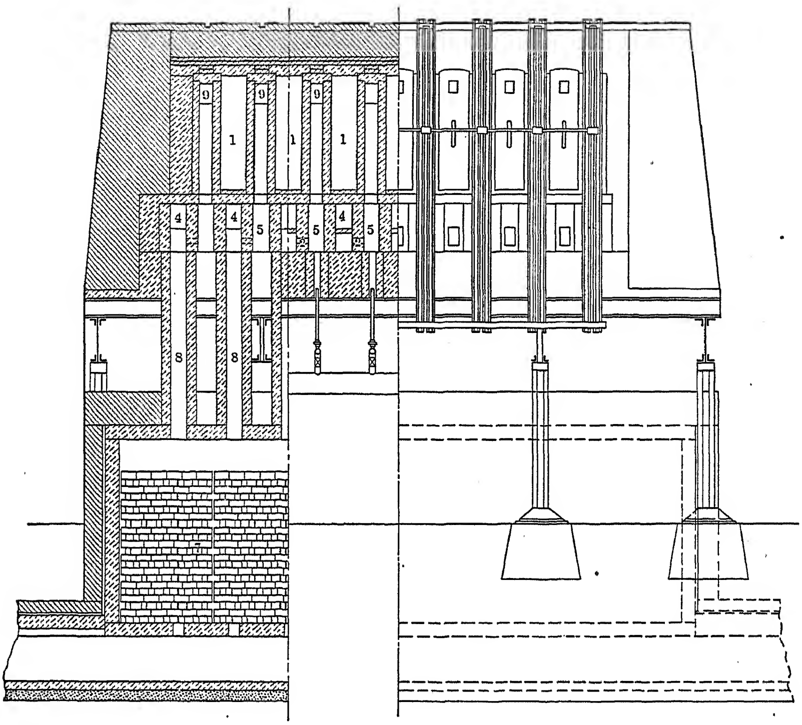


FIG. 167.—Schniewind or United-Otto by-product oven.

after heating the bottom of the retort, through the vertical flue 8 into the regenerator 7, and finally through a reversing valve into the stack. The air for the combustion chambers is admitted through openings corresponding to those marked 6 in the figure. The improvements in construction are said to have permitted lengthening the retorts which are now 48 ft. 6 in. long (*vs.* 33 ft.) 6 ft. 6 in. high, 17 in. wide, and hold eight tons of coal (*vs.* six tons).

Placing the regenerators along the center instead of along the ends of a block, as in the Otto-Hoffmann oven, has not always proved successful, because the coal near and at the ends of the retorts is often not sufficiently coked. Thus,

[1] Atlas Car & Mfg. Co., Coke Quenching Car, *Iron Age*, 1910, LXXXV, 860.
Reubold, *Stahl u. Eisen*, 1912, XXXII, 1784.

at Wyandotte, Mich., the regenerators along the center had to be replaced by others running along the ends. This led to the construction of the United-Otto by-product oven of 1906, shown in longitudinal section in Fig. 168.[1] The furnace is supported by three piers of masonry through steel I-beams. The regenerators are placed near the outer supporting walls. There are three burners for one-half of a retort; one enters at the end of the oven, the other two from beneath. A retort holds 9.5 net tons of coal and cokes the charge in 20 hr. (earlier ovens held five to six tons and required 30 hr.).

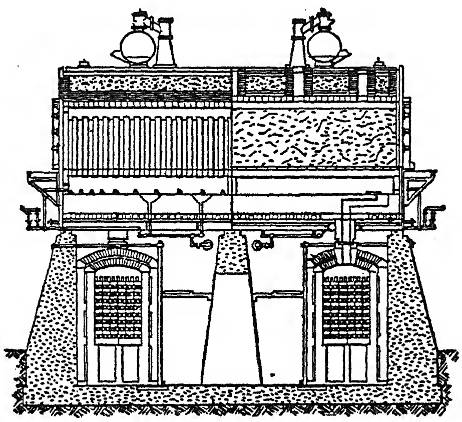


FIG. 168.—United-Otto by-product oven of 1906.

THE KOPPERS OVEN.[2]—The general form and arrangement of this oven resembles the United-Otto oven. The distinguishing features are the pre-heating of air, and the arrangements for gas and air to prevent local over-heating at points of issue and to carry the heat up high in the flues. Fig. 169 gives a longitudinal section through the coking chamber and heating flues. The retort as built to-day[3] is 40 ft. long, 11 ft. high and 19 in. wide in the middle, and takes a charge of 15 tons of coal which is coked in 15 to 20 hr., depending upon the character of the coal. The cross-section, Fig. 170, shows underneath each coking chamber a regenerator from which heated air passes into the 31 heating flues through 14 ports on the discharging and 17 on the pushing side. The division wall between the upward and downward paths is placed a little toward the discharge side. Cold air enters at K, passes

[1] *Min. Ind.*, 1906, XV, 182.

[2] *Tr. Inst. Min. Eng.*, 1903, XXIV, 627; 1907, XXXIII, 398; 1908, XXXVI, 326; *Rev. Mét.*, 1906, III, 241; *Glückauf*, 1906, XLII, 1301; *Min. Ind.*, 1907, XVI, 257; *Iron Age*, 1907, LXXX, 1671; 1910, LXXXV, 978, 1354; 1911, LXXXVII, 982 (Algoma); LXXXVIII, 1406; 1912, XC, 85; *Coll. Guard.*, 1909, XCVIII, 13; *Mines and Minerals*, 1910, XXXI, 185; *Iron Trade Rev.*, 1910, XLVII, 271; 1911, XLVIII, 89; XLIX, 1092; *Met. Chem. Eng.*, 1910, VIII, 479; 1912, X, 521; *Coal Age*, 1911, I, 337; *Can. Mg. J.*, 1911, XXXII, 333 (Algoma); *Oest. Zt. Berg. Hüttenw.*, 1911, XLIX, 255; *Zt. Berg. Hütten. Sal. Wesen i. Pr.*, 1911, XLIX, 478; *Proc. Eng. Soc. West. Pa.*, 1912, XXVIII, 311; *Eng. Min. J.*, 1908, LXXXVI, 378; *Iron Trade Rev.*, 1913, LII, 17.

[3] Hartmann, *Proc. Eng. Soc. West. Pa.*, 1912, XXVIII, 311.

damper J, rises through checkers, becomes heated, meets heating gas which arrives through distributing mains R, connections S, pipes T, and enters through nozzles A. By thus supplying each small heating flue with its own gas and air the retort is heated uniformly. The tops of the several vertical heating flues are provided with slide-dampers F manipulated through openings

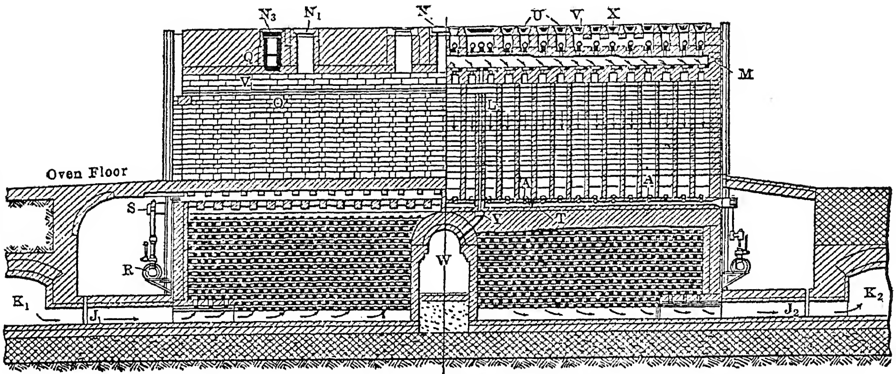


FIG. 169.—Koppers regenerative by-product oven.

U.V. . . . closed by plugs. Thus the upward passage of the gases along one-half of the retort-wall into the main flue M, and the downward travel in the heating flues along the other half, and then through ports c, regenerator, damper J_2, flue K_2 to stack, are easily regulated.

Fig. 169 shows that toward the center of the retort the flue M is made higher than at the ends; practical experience has shown this to be unnecessary. With two-thirds of the space underneath the oven taken up by regenerators, these furnish such ample heating surface that it is not necessary to pass all the products of combustion through the checkerwork; hence part of the products is withdrawn through flue L into main W leading to boilers or other furnaces for the utilization of waste heat.

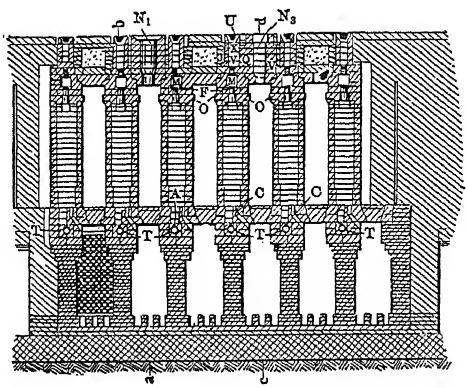


FIG. 170.—Koppers regenerative by-product oven.

A Koppers oven is started in the same manner as is an Otto-Hoffmann (p. 264). For this purpose there is provided an auxiliary uptake N_3, the opening Q and the distributing flue X. Removal of plugs V and V_1, allows the gases set free in the retort to burn in the heating flues and to travel thence through the regenerators to the stack.

The thermal balance of a by-product coking oven is represented graphically in *Stahl und Eisen*, 1909, XXIX, p. 41.

In the United States the Coppée, Otto-Hoffmann, Schniewind and Koppers ovens are the leading, if not the sole, representatives of the class of ovens heated by vertical flues. In Europe there is a greater variety as shown by the literature cited below.[1]

132. Retort-ovens with Horizontal Chambers and Horizontal Flues. THE SMET-BÜTTGENBACH OVEN.—The oldest oven with horizontal heating flues

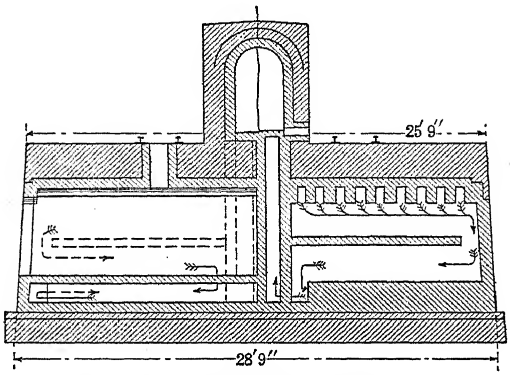


FIG. 171.—Smet-Büttgenbach oven.

is that of Haldy[2] built before 1864.[3] The Smet oven[4] is a modification of the Haldy oven, and improved by Büttgenbach it has become one of the leading horizontal-flue ovens in which by-products are not saved. Figs. 171 and 172, representing longitudinal and cross-sections, show that the heating flue on the side of a retort is divided in two by a vertical partition wall, that each section has a horizontal division wall, and that beneath the bottom of the retort are two horizontal flues. The volatile matter of say the front half of a retort is drawn off near the top through nine openings, enters the horizontal flue, meets air

[1] Brunk, *Stahl u. Eisen*, 1894, XIV, 677; 1900, XX, 685; *Eng. Min. J.*, 1895, LIX, 250; 1900, LXIX, 711; *Zt. Berg. Hütten. Sal. Wesen i. Pr.*, 1896, XLIV, 413; *J. I. and St. I.*, 1903, II, 299

Bauer, *Stahl u. Eisen*, 1899, XIX, 361; *Eng. Min. J.*, 1899, LXVIII, 310; *J. I. and St. I.*, 1903, II, 300; *Stahl u. Eisen*, 1906, XXVI, 1499; *Iron Trade Rev.*, 1908, XLIII, 189; *Iron Age*, 1908, LXXXI, 670.

Ruppert, *Zt. Berg. Hütten. Sal. Wesen i. Pr.*, 1896, XLIV, 413.

Collin, *Bull. Soc. Ind. Min.*, 1903, II, 807; *Iron Age*, 1910, LXXXVI, 272; *Zt. Berg. Hütten. Sal. Wesen i. Pr.*, 1911, LIX, 486.

Coppée (By-prod.), *Bull. Soc. Ind. Min.*, 1903, II, 797.

Poetter, *Ann. Min. Belg.*, 1903, VIII, 404; *Eng. Rev.*, 1904, XI, 159.

Kros, *Coal Age*, 1912, II, 84.

Müller, *Mines and Minerals*, 1912, XXXIII, 12; *Coll. Guard.*, 1912, CIV, 481.

Mackey-Seymour, *Coll. Eng.*, 1910, XCIX, 366.

General Review: Fr. Herbst, *Internat. Congress, Min. Met.*, Düsseldorf, 1900, volume: "Praktisches Hüttenwesen," p. 119; *Stahl u. Eisen*, 1910, XXX, 1483, 1582, 1593; *Met. Chem. Eng.*, 1910, VIII, 475.

[2] Huessener, *Stahl u. Eisen*, 1883, III, 399.

[3] *Berg. Hüttenm. Z.*, 1865, XXIV, 103.

[4] Rossigneux, *Bull. Soc. Ind. Min.*, 1891, V, 439.

entering through an opening in the door (not shown) and burns. The products of combustion, as indicated by the arrows, heat in two passes, the front half of the side wall and the bottom, and enter the main flue. The volatile matter coming from the back half of a retort does the same work for its section traveling in an opposite direction. A retort is 27 ft. 10.5 in. long, 27 to 30 in. wide and 5 ft. 5 in. high, takes a charge of 5 tons of coal, and cokes it in 42 to 48 hours.

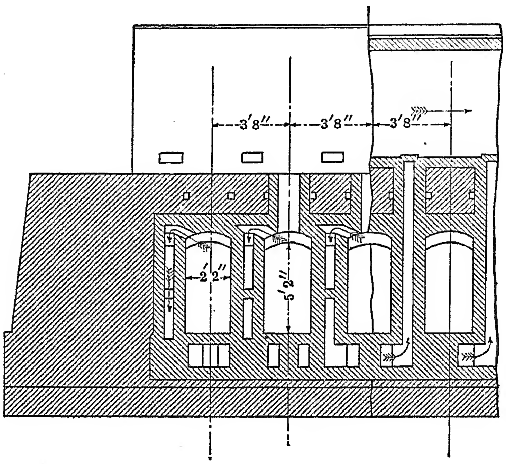


FIG. 172.—Smet-Büttgenbach oven.

THE SIMON-CARVÈS OVEN.[1]—This oven, an improvement on that of Knab[2] first built in 1856,[3] was erected in 1866 at Bessèges, France, and has been in use mainly in that country. Its latest form, as given by Babu,[4] is shown in Fig. 173. The retort is 30 ft. long, 4 ft. 7 in. high and 26 in. wide, receives a charge of five tons of coal through two hoppers and cokes it in 24 hours. The volatile matter is drawn off through the pipe *r*, the heating gas is returned from the condensation plant through a main with branch pipes *v*, air sometimes preheated in underground recuperators or bottom flues is admitted at *m* above a grate *w* on which a small coke fire is kept going to insure steady burning of the gas. From *w* the products of combustion pass out through one bottom flue *t*, return by another, ascend in the vertical passage *k*, descend in zig-zag through hori-

[1] Jougouet, *Bull. Soc. Ind. Min* , 1880, IX, 283.
Huessener, *Stahl u. Eisen*, 1883, III, 404, 560.
Simon, *J. I. and St. I.*, 1885, I, 108.
Lürman, *Stahl u. Eisen*, 1885, V, 306.
Tuska, *Iron Age*, Dec. 11, 1890.
Rossigneux, *Bull. Soc. Ind. Min.*, 1891, V, 458.
Simon-Carvès Ld., *Coll. Guard.*, 1897, LXXIV, 511; 1909, XCVIII, 1182.
Editor, *Eng. Min. J.*, 1897, LXIV, 579.
Ellison, *Tr. Inst. Min. Eng.*, 1901, XXI, 79.
[2] *Bull. Soc. Ind. Min.*, 1891, V, 456.
[3] *Dingl. Pol. J.*, 1859, CLIV, 97.
[4] "Métallurgie Générale," 1906, vol. II, p. 132.

zontal flues *z*, *h*, *h'* and travel through an underground flue *S* to the stack or to waste-heat boilers.

The ovens of Seibel[1] and Huessener[2] resemble the Carvès oven; that of Festner[1] is a modification of the Huessener. More recent European forms are those of Collin[3] and Bernard.[4]

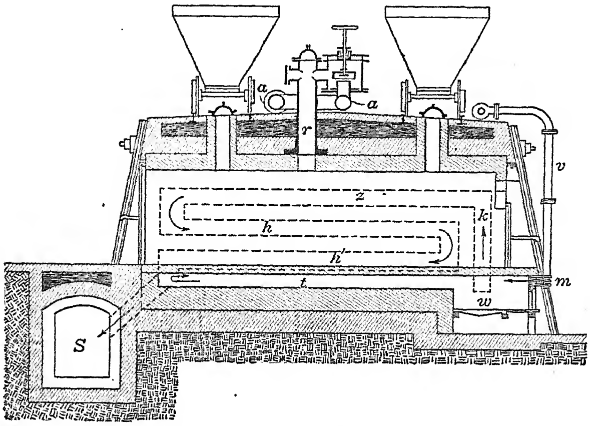


FIG. 173.—Simon-Carves by-product oven.

THE SEMET-SOLVAY OVEN.[5] Figs. 174 to 176.—This oven originated in Belgium in 1882. In the United States the first ovens were erected in 1891–93 at Syracuse, N. Y., and in 1906 there were in operation 1,230 ovens, and building 204. The main characteristics of the oven are that the heating-flues *J* on either side of a retort are thin hollow refractory tiles, Fig. 176, and that the roof *I* is carried by brick walls *G* placed between the tiles. The standard retorts of 1893 are 30 ft. long, 5 ft. 6 in. high and 16 to 17 in. wide; they have three charging holes *A* and one gas uptake *B*. The charge, 4.6 tons of coal, is coked in 18 to 20 hours, *i.e.*, very quickly. The gas returning from the condensation plant is admitted through the pipes *D* in the upper and middle heating-flues where it meets the necessary air which entering at *F* and traveling in the direc-

[1] *Bull. Soc. Ind. Min.*, 1891, V, 472.

[2] *Stahl u. Eisen.*, 1883, III, 409; *Bull. Soc. Ind. Min.*, 1891, V, 475; *J. I. and St. I*, 1904, I, 188; *Tr. Inst. Min. Eng.*, 1906, XXXI, 451; *Stahl u. Eisen*, 1892, XII, 827.

[3] *Mines and Minerals*, 1898, XVIII, 414; *Coll. Guard.*, 1898, LXXV, 17; *Eng. Min. J.*, 1898, LXV, 191; *Glück Auf*, 1901, XXXVII, 932; *Bull. Soc. Ind. Min.*, 1903, II, 813.

[4] *Op. cit.*, 1903, II, 826.

[5] *Eng. Min. J.*, 1890, L, 165; 1895, LX, 5131 (Editor); *Bull. Soc. Ind. Min.*, 1891, V, 505 (Rossigneux); 1903, II, 818 (Hurez); *Coll. Gaurd.*, 1892, LXIII, 149 (Demanet); *Stahl u. Eisen*, 1892, XII, 186, 828 (Lürmann); *Min. Ind.*, 1895, IV, 231 (Blauvelt); *Ala. Ind. Sc. Soc.*, 1897, VII[2], 33 (Blauvelt); *Tr. A. I. M. E.*, 1898, XXVIII, 578; 1912, XLIV (Blauvelt); *Proc. West. Soc. Eng.*, 1905, X, 477 (Blauvelt); *Proc. Am. Soc. Mech. Eng.*, 1908, XXX, 261 (Blauvelt); *Min. Ind.*, 1906, XV, 187; PLANTS: Ensley, Ala, *Tr. A. I. M. E.*, 1898, XXVIII, 579; Dunbar, Pa., *Mines and Minerals*, 1899–1900, XX, 297; Wheeling, W. Va., *Tr. Inst. Min. Eng.*, 1899–1900, XIX, 37; Steelton, Pa., *Iron Trade Rev.*, 1910, XLVII, 1155.

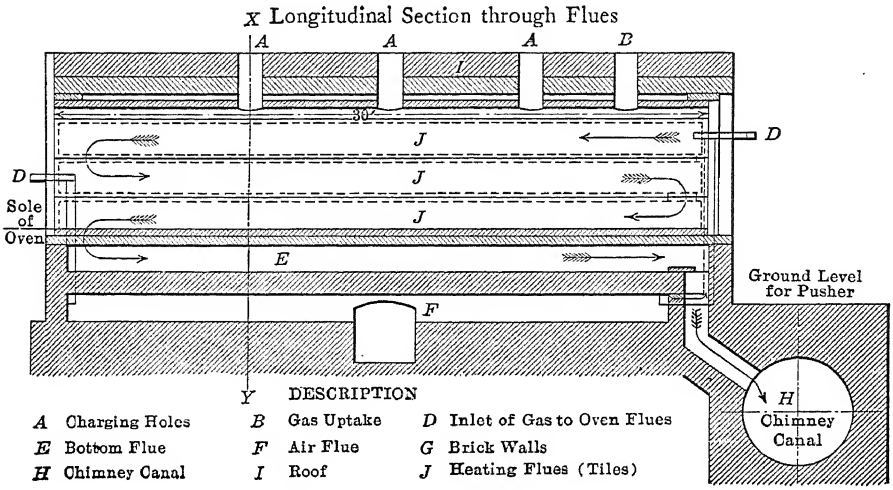


FIG. 174.—Semet-Solvay oven.

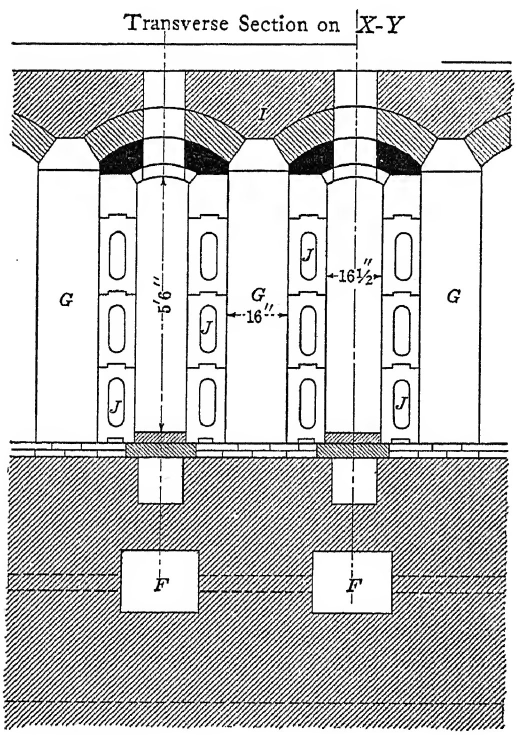


FIG. 175.—Semet-Solvay oven.

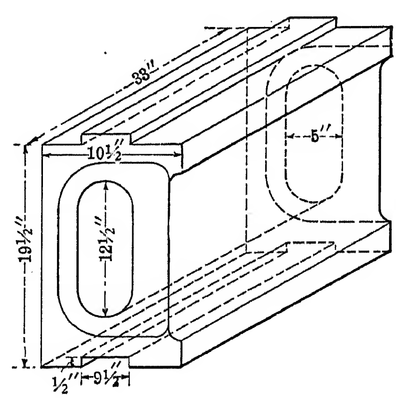


FIG. 176.—Tile of Semet-Solvay oven.

tion of the arrows has been superheated to 200–280° C. The products of combustion descend through *J* in a zig-zag course as shown and pass through the bottom heating-flue *E* to the chimney canal. The following temperatures have been observed: upper heating-flue 95 to 1,100° C.; middle flue 1,100 to 1,400° C.; lower flue 1,000 to 1,350° C.; main in front of oven 600 to 800° C.

The rise of temperature in the center of the charge is shown by the following records taken in Lancashire, England.[1] The retort was charged at 1 p. m.; rise in temperature until 10 p. m. not over 100° C.; at midnight to 200°; at 1 a. m. to 350°; at 2 p. m. to 530; at 3 to 600; at 4 to 700; at 5 to 800; at 6 to 900; at 7 to 920; at 11 to 1,000° C., at which time the coke was discharged.

On account of the recuperative effect of the 18-in. brick walls *G* it is not necessary to charge the even-numbered retorts alternately with the odd-numbered as is the case with other retort-ovens built of clay- or silica-brick. A record for May, 1897, of a block of 25 ovens at Syracuse, N. Y., was: coal coked 6,004,316 lb., large coke produced 67.2 per cent., breeze 2.76 per cent.; by-products per net ton coal: $(NH_4)_2SO_4$ 17.61 lb., tar 84.1 lb.

TABLE 120.—ANALYSES OF COAL AND COKE, SEMET-SOLVAY PLANT, SYRACUSE, N. Y.

	May 1 to 15		May 16 to 31	
	Coal	Coke	Coal	Coke
V.M.	29.56	1.00	25.47	1.00
F.C.	63.17	88.68	67.87	90.04
Ash	7.27	10 32	6.66	8.96
Total	100 00	100.00	100.00	100.00
S	1.45	1.40	1.38	1.17

The leading drawback of the original Semet-Solvay oven in the United States is that it has so far been found impossible to make a combination of clays for moulding flue-tiles, as these crack a short time after they have been in use. All tiles have been imported from Belgium, and to reduce the cost, the flues are generally built up at present of fire-brick or silica-brick.

In recent years the size of retort has been increased, the height from 5 ft. 6 in. to 6 ft. 2 3/5 in. (standard) and even to 9 ft., the length from 30 ft. to 35 ft., the width of 16.5 in. has remained unchanged; the capacity has correspondingly grown from 4.5 to 7 (standard) and even to 9 tons; the number of heating-flues has risen from 3 to 4 (standard) and even 5; the number of ovens in a block varies from 25 to 80; the coking time was 26 hr. in 1893; it was reduced to 18 hr. in 1906. This reduction is to be attributed to the use of higher heats and

[1] *Eng. and Min. J.*, 1907, LXXXIV, 27.

to the installation of better machinery for charging and discharging the retorts.

Fig. 177 represents a cross-section of the present form of Semet-Solvay oven. There are five horizontal heating-flues built of small brick having a special form of joint. The upper four flues are provided with gas and air inlets; the air for combustion is heated to a temperature ranging from 200 to 500° C.

THE ROTHBERG OVEN[1] of 1903 is shown in Figs. 178 and 179. It is a modification of the Semet-Solvay oven. The heating-flues *F* are divided in two by a vertical wall *A*; they are built up of standard brick and are not separated from one another by a solid wall so that one flue serves two adjacent retorts. The gas from the condensation plant is admitted at *P* and *P'*, the air for combustion, drawn in at *S*, is heated to 400° C. by passing through the recuperator flue *M*; air is further admitted through the peep-holes *B*; *GG* are dampers. The products of combustion, after zig-zagging downward in the side-flues *F*, pass through the bottom-flue *K* and then through *N* into *R*. Beneath the bottom-flues *K* are air-cooling flues to protect the brickwork. The retorts are 33 to 40 ft. long, 6 ft. 6 in. to 7 ft. 6 in. high, 16 to 22 in. wide, have a capacity of 6 to 9 tons loose or 7 to 11 tons of compressed coal; the coking time is 24 to 30 hr. The Cleveland Furnace Co., Cleveland, O., has two blocks of 40 ovens each, 36 ft. × 6 ft. 6 in. ×16 in., which are charged with 7.5 tons of 3/16 in. coal that has been moistened with 12 to 15 per cent. water and compressed. Tamping the coal has increased the net capacity of the oven by 15 per cent. The coal (V.M. 34.50, F.C. 54.50, ash 11.00, S 1.00) from Washington County, Pa., gives untamped a spongy brittle coke; tamped it furnishes a good iron blast-furnace coke (V.M. 1.50, F.C. 84.50, ash 14.00, S 1.00, cell-space 41.13 per cent.) with a coking-time of 24 hr. The yield in coke is 71 per cent., in breeze 1.50; the gas consumption per ton of coal is 6,700 cu. ft. leaving 3,260 cu. ft. surplus gas; there are produced 10.56 gal. tar and 19.36 lb. $(NH_4)_2SO_4(=4.84\ NH_3)$.

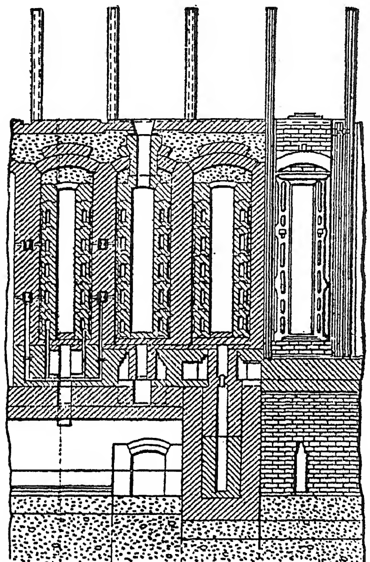


FIG. 177.—Semet-Solvay oven, latest form.

Other ovens with horizontal heating-flues are the Simplex,[2] Hüssener,[3] Mackey-Seymour,[4] Bauer,[5] etc.

[1] Rothberg, *Mines and Minerals*, 1904, XXIV, 362; Editor, *Iron Age*, 1905, LXXVI, 855.

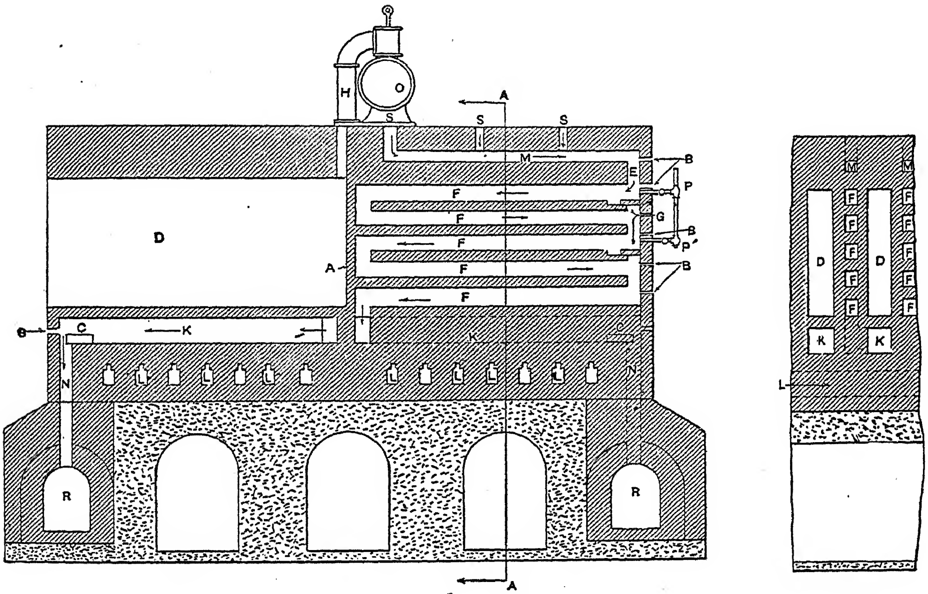
[2] Jackson, *Tr. Inst. Min. Eng.*, 1907, XXXVIII, 386.

[3] *Coll. Guard.*, 1909, XCVIII, 1283.

[4] *Coll. Eng.*, 1910, XCIX, 366.

[5] Herbst, Fr., *Internat. Congress Min. Met.*, Düsseldorf, 1910, volume: "Praktisches Hüttenwesen," p. 127.

133. Continuous Coke-ovens.—The Lürmann Oven,[1] until recently, was the sole representative. All coking ovens so far discussed work intermittently. In the Lürmann oven, with its horizontal pyramidal coking chamber, the coal is continuously fed at one end and the coke discharged at the other into a cooling chamber. With intermittent ovens the amount of volatile matter evolved changes considerably; it is large at first and diminishes steadily until the end of the process; in Lürmann's continuous ovens it is uniform, hence the heating of the retort is also uniform even when all the volatile matter is burnt in the flues. In an intermittent oven not recovering by-products, the temperature of a newly charged oven will rise quickly, reach a maximum, and decrease gradually to the low point when it will be further reduced by the cooling effect of a



Figs. 178 and 179.—Rothberg oven.

new charge. This is avoided by the continuous furnace. The Lürmann oven was in operation at Osnabrück, Hanover, Germany, where a mixture of 60 per cent. anthracitic and 40 per cent. good coking coal was converted into iron blast-furnace coke. Two other continuous coking ovens have been recently constructed, the Sheldon[2] and the Woodall-Duckham.[3]

134. Condensation of By-products.[4]—The amounts of gas, gas-liquor and

[1] Huessener, *Stahl u. Eisen*, 1883, III, 401; Rossigneux, *Bull. Soc. Ind. Min.*, 1891, V, 516.
[2] *Iron Age*, 1908, LXXXI, 197.
[3] *Coal Age*, 1911, I, 314; *Feuerungstechnik*, 1913, I, 211.
[4] Lunge, G., Coal Tar and Ammonia, Gurney, London, 4th Ed., 1910.
Otto, *Stahl u. Eisen*, 1884, IV, 396.
Leistikow, *op. cit.*, 1892, XII, 818.
Simmersbach, *Zt. Berg. Hütten. Sal. Wesen i. Pr.*, 1896, XLIV, 402.

tar produced were given on p. 228. A general scheme showing the treatment of volatile matter in a by-product plant is given in Fig. 180. It presupposes that the gas rich in illuminants given off during the first part of the coking period is drawn off and treated separately from that given off later on.

Fig. 181 is a diagrammatic representation of the general arrangement of a condensation plant of the United Coke and Gas Co. of New York for using the

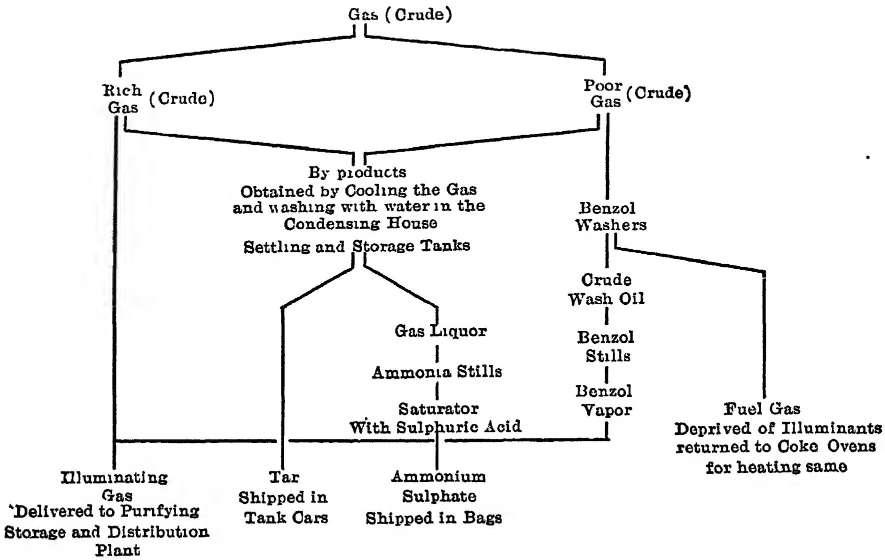


FIG. 180.—Tree showing treatment of volatile matter in by-product plant.

surplus gas for illuminating purposes. If fractional distillation is not practised, the mode of operating and with it the plant becomes very much simplified, as the treatment of the poor gas disappears and with it the complicated recovery of benzol. The volatile matter is drawn[1] from the ovens by rotary exhausters placed back of the scrubbers and then forced through the rest of the plant. It leaves an oven at a temperature of 600 to 700° C. through up-takes which are cast-iron pipes having valves to permit shutting off an oven. The up-take pipes of a block of ovens end in one or two inclined collecting mains running along the tops of the ovens, the number depending upon the use of surplus gas for heating or illuminating purposes. The temperature in the mains is about

Schniewind, *Iron Age*, Nov. 28, 1901.
Atwater, *Tr. A. I. M. E.*, 1903, XXXIII, 766;
Catalogue, United Otto System, New York, 1906.
Say, *Bull. Soc. Ind. Min.*, 1909, x, 75, 157, 269, 365, 463, 551; 1910, XIII, 135.
O. Rau, *Internat. Congress Min. Met.*, Düsseldorf, 1910, p. 280, vol. Prakt. Hüttenw.
General Review, *Stahl u. Eisen*, 1910, XXX, 1235, 1282.

[1] Gradenwitz, *Eng. and Min. J.*, 1910, XC, 825.

150° C. Some heavy hydro-carbons are condensed, and particles of coal dust[1] are collected. Tar and water are condensed later and collected in a cement-lined cistern (No. 1). The gases pass through air-coolers (Fig. 181) (oblong zig-zag steel-plate tubes) which reduce the temperature to about 75° C. and create a vacuum of 1.25 in. water; then through water-coolers (tubular boilers) which bring the temperature down to about 25° C., and finally through a tar-scrubber (square steel-plate box with perforated baffle-plates) into the exhauster (rotary blower) which they enter with a vacuum of 3.75 in. The gas has been freed from nearly all of its tar and water vapor, both of which are collected in cistern No. 1, the gas-liquor of which contains about 0.5 per cent. NH_3. The gas leaves the exhauster under a pressure of 21 in. water. The compression causes a rise in temperature of 6 to 8° C.; this is corrected by the after-cooler (called tar-extractor in Fig. 181, as the last traces of tar are removed) which

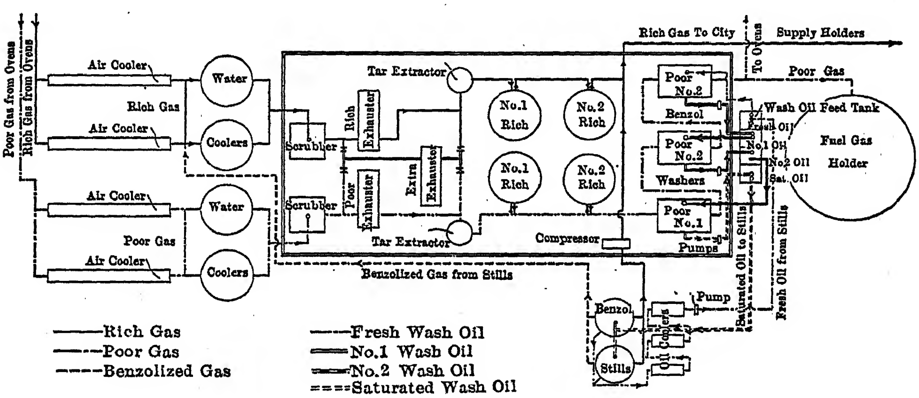


FIG. 181.—Arrangement of condensing house for coke oven gas with benzol enrichment.

reduces the temperature to 20° C. From the after-cooler the gas passes ammonia washers or scrubbers.[2] Stationary or rotary bell-washers have been replaced in some plants by tower-washers, *i.e.*, steel-plate towers (Zschokke scrubbers, Figs. 822 and 823) filled with wooden lattice work over which weak ammonia liquor from cistern No. 1, or pure water, trickles downward, meets the ascending gas current, absorbs the NH_3 and forms a strong gas liquor to be collected in cistern No. 2, or a weak liquor to be collected in cistern No. 1. Bell-washers, although more complicated and expensive than tower-washers, furnish a more concentrated NH_3 liquor, *e.g.*, 2.5 *vs.* 1 per cent. NH_3.

While a temperature lower than 16 to 20° C. would be more favorable for the collection of NH_3, there is too great a danger of clogging the passages with naphthalene. The poor gas which has been freed from tar and ammonia still contains about 20 grams benzol (C_6H_6) per cbm. (0.0012 lb. per cu. ft.). As this

[1] Platz, *Stahl u. Eisen*, 1889, IX, 755; Thau, *Glück auf*, 1910, XLVI, 1839.

[2] Berger, *Stahl u. Eisen*, 1910, XXX, 443.

is the most powerful illuminating hydrocarbon, it is extracted[1] from the gas by means of tar oil, separated again from the oil by distillation and conducted into the rich-gas circuit.

In Fig. 181 the gas is forced through three tar-oil scrubbers (Poor Nos. 1, 2 and 3), goes through the fuel-gas holder, which serves not so much to store gas as to equalize the pressure, and then back to the coking-ovens. The arrows show that fresh oil enters benzol washer 3; slightly charged it goes to tank "No. 1 oil" from which it is fed into benzol washer No. 2 to collect more benzol, and when enriched to pass through tank "No. 2 oil" into washer No. 1 where it becomes saturated and flows into tank "sat. oil." The saturated oil passes through two benzol stills from which the benzolized gas enters the rich-gas circuit, while the oil freed from benzol flows through oil-coolers to the tank "fresh oil" ready to take up again benzol.

The 0.75 to 1.00 per cent. NH_3[2] of the gas liquor is present as volatile (caustic, carbonate, sulphide) and fixed (sulphate, chloride, cyanide, rhodanate) ammonia; the latter is decomposed with lime and the two are recovered together by distillation[3] being collected as $(NH_4)_4SO_4$ in H_2SO_4 of 41° B.

In recent years the usual "indirect recovery" of NH_3 of Otto just described, which requires a number of operations (air- and water-cooling, water-washing, separation of tar and NH_3-water, distillation with $Ca(OH)_2$) has been replaced in several instances by what is termed a "direct recovery" in which the oven-gas is freed from NH_3 by a single operation.[4] In the Hilgenstock method[5] the gas-current is mixed by means of an injector with finely divided tar which acts as a collector for tar-vapor or mist and causes the tar to separate, when the tar-free gas is passed through H_2SO_4 which combines with the NH_3 forming $(NH_2)_2SO_4$, insoluble in the acid. Another principle is that of Feld [6]

The tar[7] contains a large number of solid and liquid hydrocarbons, phenols and asphalt-forming constituents. The two proximate analyses[8] of tar given in

[1] Donath, *Oest. Zt. Berg. Hüttenw.*, 1893, XLI, 637, 649.
Lunge, *Min. Ind.*, 1896, V, 185.
Heinzerling, *op. cit.*, p. 199.
A., *Stahl u. Eisen*, 1897, XVII, 531.
Bagley, *Iron Age*, 1911, LXXXVIII, 756.
[2] *Min. Ind.*, 1895, IV, 239.
[3] *Op. cit.*, 1896, V, 191.
Koppers plant, *Iron Age*, 1908, LXXXII, 784; *Iron Trade Rev.*, 1908, XLIII, 393.
[4] Ohnesorge, *Stahl u. Eisen*, 1910, XXX, 113.
Meyn, *Oest. Zt. Berg. Hüttenw.*, 1911, LIX, 15.
Dobbelstein, *Glück Auf*, 1911, XLVII, 194.
Burg, *Can. Min. J.*, 1911, XXXII, 449.
[5] Hilgenstock, *Stahl u. Eisen*, 1909, XXIX, 1644; *Iron Age*, 1909, LXXXIV, 1764.
[6] *Met. Chem. Eng.*, 1912, X, 395.
[7] Hubbard, P., "Coke-oven Tars of the U. S.," Dep. Agricult., Office Public Roads, *Circ.* 97, Feb. 12, 1912.
Proximate Analysis, Trobridge, J., *Soc. Chem. Ind.*, 1909, XXVII, 230.
A. Spieker, "Kokerei und Teerproducte," Knapp, Halle, 1908, p. 100.
[8] *Min. Ind.*, 1896, V, 188.

Table 121 are from a gas-works and an Otto-Hoffmann coking plant working the same coal; 26 examples are noted by Hubbard.

TABLE 121.—PROXIMATE ANALYSES OF GAS- AND COKE-OVEN TARS

Kind	Water, per cent.	Light oil up to 200°, per cent.	Benzol for aniline, per cent.	Solvent naphtha, per cent.	Creosote oil, per cent.	Crude naphthalene, per cent.	Anthracene oil, per cent.	Pure anthracene, per cent.	Pitch, per cent.	Carbon in pitch, per cent.	Yield of tar on coal, per cent.
Gas-tar........	2.90	4 00	0 92	0 20	8 60	7.40	17.40	0.60	58 40	15–25	......
Coke-oven tar..	2 20	3 40	1.10	0 32	14 50	6.70	27.30	0.70	44.30	5–8	3.00

The ultimate composition is within wide limits.[1] Lunge[2] gives as an average C 78.0, H 6.5, N 1.0, S 0.5, O 14.

135. Comparison of Methods of Coking. YIELD AND COST.—Below are assembled a few leading facts by Lucas[3] about coking in a beehive and a by-product coking-oven. Older data have been published by Blauvelt in 1895[4] and 1905.[5]

COST OF COKING IN BEEHIVE AND BY-PRODUCT COKING OVENS

Beehive: Ordinary type, 12.5 ft. in diameter Cost from $700 to $1,200 per oven. Product 4 net tons of coke in 48 hr. = 2 net tons in 24 hr. Yield of coke from coal, 60 per cent. By-products and surplus gas = none.

By-product Oven: Oven charge, 9 tons. Coking-time, 24 hr. Ovens may be larger or smaller than this, but 9 tons would probably be about the average charge for the modern type of oven. Coke produced on 70 per cent. yield = 6.3 tons of coke per oven in 24 hr.

By-products: Ammonium sulphate, 22 lb. per net ton of coal = 31 lb. per net ton of coke. Value, 2.25 cents per lb. above cost of manufacture = 70 cents per ton of coke made. *Tar.* 8.5 gal. per ton of coal = 10.7 gal. per ton of coke, at 2 cents per gal. = 21 cents per ton of coke. *Surplus gas,* 5,000 cu. ft. per ton of coal = 7,143 cu. ft. per ton of coke, at 10 cents per 1,000 cu. ft. = 71 cents per ton of coke. *Total Value of By-products as above:* Ammonium sulphate $0.070, tar $0.21, gas $0.71 = $1.62 per ton of coke.

To the above should be added the difference between 60 per cent. yield in beehive ovens and 70 per cent. in by-product ovens on the same coal. With coal at $1.50 per ton; the cost of coal per ton of coke produced in beehive oven = $2.50, that of coal per ton of coke produced in by-product oven = $2.14. Balance in favor of by-product oven = $0.36. The total saving in coal and by-products equals $1.62 plus $0.36 = $1.98 per ton of coke made, = $12.47 per oven in 24 hr. = $4,551.55 per oven per year. The saving in by-products alone, without saving in coal, $3,723 per oven per year. For a plant of 100 ovens, saving = $455,155 per year. Cost of 100-oven plant complete is approximately $1,000,000. A 100-oven plant of above capacity will produce 630 tons of coke per day = 229,950 tons per year, working on 24 hr.-coking time. If benzol is recovered it will further add to the income from by-products.

[1] *Glück Auf,* 1905, XLI, 1406.
[2] "*Chemisch-technische Untersuchungsmethoden,*" Springer, Berlin, 1905, II, 727.
[3] *Tr. A. I M. E.,* 1912, XLIV, 170, 184.
[4] *Min. Ind.,* 1895, IV, 242.
[5] Fulton, *op. cit.,* p. 398.

BEEHIVE *vs.* RETORT OVEN.[1]—The beehive is suited only for certain coals. The plant is cheap, but the yield in coke is low and the production small. The cost of coking is also low although all operations are carried on by manual labor. Retort-ovens are suited for a great variety of coals. The plant is more expensive than that of the beehive, but the yield is higher. The total cost of coking is likely to be higher than in the beehive if all the volatile matter is burnt in the flues, and very much so if by-products are recovered. The net cost with non-by-product ovens remains the same as the total cost; the net cost with by-product ovens is very much lower than with the beehive. The higher yield in coke of the non-by-product retort ovens more than balances the slightly higher total cost of coking when compared with the beehive. Retort coke is more uniform in size and harder than beehive coke; it has a smaller percentage of cell-spaces and is therefore about 10 per cent. heavier, bulk for bulk; the resistance to oxidation by CO_2 is greater; the tensile strength is about the same. It was once held that retort-ovens could not furnish an iron blast-furnace coke suited for the rapid driving of American practice. This belief was strengthened by the experience of Bell[2] in England; later experiments[3] decided him to prefer Huessener by-product coke to that from the beehive. Tests[4] carried out at Syracuse and Buffalo, N. Y., with Semet-Solvay coke showed that pound for pound retort-coke produced the same amount of pig iron as beehive coke; later tests by Uehling[5] with Otto-Hoffmann coke showed that less of this was required to smelt a ton of iron than when beehive coke was used, and that with an equal volume of blast the pressure was higher. The second fact agrees with other experiences that for the same production it is necessary to use a higher pressure. On the European continent, beehives have fallen into disuse; in Great Britain they are still in operation, although they are giving way to retort-ovens; in the United States where large sums of money are invested in beehives, it will take some time before they are replaced by retort-ovens, especially also as a market for tar has to be created; in 1910,[6] 17.12 per cent. of the total coke was made in by-product ovens.

VERTICAL *vs.* HORIZONTAL RETORTS.—With a few exceptions ovens with vertical retorts have had to give way to the horizontal type. Their advantages are: small floor space, quick coking on account of the large heating surface, high and even temperature of a retort, high coking column, ease of charging and discharging. The disadvantages are: small production, non-recovery of by-products, danger of overheating central retorts, frequent repairs and necessity of shutting down a block for inner repairs of any kind. However, the ovens

[1] Ernst, *Iron Age*, 1908, LXXXI, 220.
Hartmann, *Proc. Eng. Soc. West. Pa.*, 1912, XXVIII, 314.
Blauvelt, *Tr. A. I. M. E.*, 1912, XLII.
Andrews, *op. cit.*, 1912, XLIV, 154.

[2] *J. I. and St. I.*, 1885, I, 57; *Stahl u. Eisen*, 1885, V, 298.

[3] *J. I. and St. I.*, 1904, I, 188.

[4] Fulton, "Coke," 1905, p. 277.

[5] *Iron Age*, June 11, 1903.

[6] *Min. Res. U. S.*, 1910, part II, p. 262.

of Schorr and Bauer (p. 256) recovering by-products show that under suitable conditions the vertical retort may still hold its own.

VERTICAL *vs.* HORIZONTAL HEATING FLUES IN HORIZONTAL RETORTS.[1]—The relative advantages of vertical over horizontal heating-flues or *vice versa* cannot be so very decided as otherwise one class would have had to yield to the other, and we find both in extensive use.

ADVANTAGES OF VERTICAL FLUES.[2]—The brick retort is strong, the friction in the short flues is small, and the gases make at most two passes; the danger of uneven heating in the older by-product ovens has been in part corrected by the modern Otto method of underfiring, but the retort is liable to be hotter near the bottom than at the top, and this will act unfavorably for the recovery of by-products.

DISADVANTAGES OF HORIZONTAL FLUES.—The flues have to be numerous to give strength to the retort (obviated by Semet-Solvay tiles); flues are long, causing much friction (obviated in part by Rothberg's partition wall); gases make more than two passes which causes friction; uneven heating at center by admission of gas at ends only; greatest heat at top of retort which is unfavorable for recovery of by-products.

REGENERATIVE AND NON-REGENERATIVE SYSTEMS.—The object of having regenerators is to utilize the heat of the gases that pass from the heating-flues of the retorts for the operation of coking and not for extraneous purposes, such as raising steam, calcining, etc. Working with superheated air causes a saving in heating-gas and thus furnishes a larger amount of excess gas. As regenerators add about one-third to the cost of the plant, they are built only where it is of importance to have a large amount of excess gas. This, *e.g.*, is the case with plants in which the excess gas is used for illuminating purposes. (Boston, Baltimore, Pittsburgh, Hamilton, O., Camden, N. J. . . .). That regenerators are not imperative is shown by a Semet-Solvay plant furnishing gas of 18 candle-power to Detroit, Mich.

An argument in favor of regenerators is that utilization of the excess gas in gas engines[3] gives greater efficiency than raising steam with the waste heat of non-regenerative furnaces.

BURNING AND SAVING OF BY-PRODUCTS.[4]—The main points to be considered are character of coal, yield of coke, cost and location of plant, disposal of waste liquors, and market for by-products. Coking-coals rich in volatile matter which furnish much tar, ammonia and benzol, and require slow coking are better adapted for a by-product plant than lean coals demanding coking quickly at a high temperature. In a non-by-product oven the volatile matter, as soon as set free, is drawn off into the heating-flue and is given little chance for deposi-

[1] Schniewind, *Iron Age*, 1895, LV, 446.
Atwater, *Coll. Guard.*, 1896, LXXI, 695.
Blauvelt, *Min. Ind.*, 1897, VI, 172; *Tr. A. I. M. E.*, 1912, XLIV, 154.

[2] Fieschli, *Eng. and Min. J.*, 1908, LXXXVI, 378.

[3] Baum, "Die Verwertung des Koksofengases, insbesondere seine Verwendung zum Gasmotorenbetrieb," Springer, Berlin, 1904.

[4] Blauvelt, *Tr. A. I. M. E.*, 1912, XLIV, 154.

tion of carbon; further since the pressure in the retort is slightly below atmospheric, air is likely to enter through unavoidable cracks. For both reasons the yield in coke is from 3 to 5 per cent. smaller than in a by-product plant. The cost of the latter is, however, three to four times as great as a non-by-product plant. A by-product plant requires large amounts of water and has to dispose of corresponding quantities of offensive waste liquors[1] charged with cyanides, purifiers and other refuse; it further liberates offensive odors (H_2S). In the United States there is a steady demand for $(NH_4)_2SO_4$, but the tar industry is still in its infancy; nevertheless by-product plants are steadily growing in number.

136. Liquid Fuels in General.—Petroleum is the leading natural liquid metallurgical fuel. The artificial oils obtained in the distillation of shales and of tars, and in coking, and collected from iron blast-furnace gases run with coal, are of secondary importance.

137. Petroleum.[2]—Petroleum has been used in Europe and Asia for centuries. Evidence exists that petroleum was known to prehistoric inhabitants of the United States. The industrial production of petroleum began in the United States in 1859 with the striking of oil in the Drake well at Titusville, Pa. The United States was the only important producer until the deposits of Baku near the Caspian sea began to be operated about 1872.

The world's production in 1910 is given in Table 122.

TABLE 122.—WORLD'S PRODUCTION OF PETROLEUM IN 1910[3]
(Bbl. at 42 gal.)

Country	Million bbl.	Per cent. total	Country	Million bbl.	Per cent total
United States	209	63.99	Japan	2	0.59
Russia	70	21.48	Peru	1	0.40
Galicia	13	3.87	Germany	1	0.32
Dutch East India	11	3.37	Canada, Italy, others	1	0.12
Roumania	10	2.97			
India	6	1.87	Total	327	100
Mexico	3	1.02			

The production of the United States in 1910 (see *op. cit.*, p. 329) is given in Table 123.

[1] Wilson, *Trans. Inst. Min. Eng.*, 1910, XXIX, 71.

[2] Peckham, S. F., "Report on Production, Technology and Uses of Petroleum," *Tenth Census U. S.*, 1880, Washington, 1885.

Crew, B. J., "A Practical Treatise on Petroleum," Baird & Co., Philadelphia, 1887.

Redwood, B., "Petroleum," Griffin & Co., London, 1906.

Richardson, "The Petroleum of North America," *J. Frankl. Inst.*, 1906, CLXII, 57, 81.

Tassart, L. C., "Exploitation du pétrole," Dunod, Paris, 1908.

Engler, C., and Höfer, H., "Das Erdöl," Hirzel, Leipsic, 1909, 5 Vol., in process of publication.

[3] *Min. Res. U. S.*, 1910, II, 454.

TABLE 123.—PRODUCTION OF PETROLEUM IN THE UNITED STATES IN 1910
(Bbl. at 42 gal.)

State	Rank	Million bbl.	Percentage	State	Rank	Million bbl.	Percentage
California	1	73	34.84	New York	11	1	0.50
Oklahoma	2	53	24.83	Kentucky	12	...	0.22
Illinois	3	33	15.82	Colorado	13	....	0.12
West Virginia	4	12	5.61	Wyoming	14		
Ohio	5	10	4.73	Utah	15	1	0.05
Texas	6	9	4.25	Michigan	16		
Pennsylvania	7	9	4.20	Missouri	17		
Louisiana	8	7	3.26				
Indiana	9	2	1.03	Total	.....	209	100.00
Kansas	10	1	0.54				

The United States Geological Survey[1] distinguishes the following oil-fields:

The *Appalachian Field* includes western New York, Pennsylvania, eastern Ohio, West Virginia, Kentucky, Tennessee. The oil is free from S and asphalt, rich in paraffine wax, yields largest percentage of gasoline and illuminating oils. The *Lima-Indiana Field* includes northwestern Ohio and a strip of middle Indiana. The oil contains S, requires special treatment and yields paraffine wax. The *Illinois Field* includes southeastern Illinois. The oil contains less S than the preceding; much of it requires no special treatment; some contains asphalt as well as paraffine. The *Mid-continent Field* includes Kansas, Oklahoma and northern Texas. The oil may contain both asphalt and paraffine. The *Gulf Field* includes Texas and Louisiana. The oil contains much S, yields valuable lubricating oils and gasoline. The *California Field* includes California. The oils contain much asphalt, little or no paraffine.[2] *Other Fields:* Small quantities of oil are produced in Wyoming, Colorado, Indiana, Michigan, Montana, Washington, western Oregon, eastern Oregon, Idaho, Nevada, and southwestern Utah. Of these Wyoming gives most promise.

CRUDE PETROLEUM.—Crude petroleum is an odorous fluid ranging in color from pale-yellow to black, and in fluidity from highly mobile to viscid. Its specific gravity varies from 0.73 to 0.97 (61 to 14° Bé.). The specific gravity is usually given in degrees Beaumé for liquids lighter than water (range: 10 to 95° Bé.=spec. gr. 1.0000 (H_2O)−0.6222). American oils, on the whole, have a lower specific gravity (47 to 25° Bé.) than those of Russia (36 to 14° Bé.). Some Wyoming,[3] California[4] and Texas[5] oils resemble the Russian oils. Light-color

[1] *Min. Res., U. S.*, 1907, p. 352; 1908, p. 352.
[2] Requa, M. L., "Oil Resources of California, University Cal., 1912.
[3] University of Wyoming, School of Mines Petroleum Series Bulletin.
[4] Allen, I. C., and Jacobs, W. A., "Physical and Chemical Properties of Petroleum, San Joaquin, Cal., *Bull.* 19, Bureau Mines, Washington, 1911.
[5] Phillips, W. B., "Texas Petroleum," *Bull.* 5, University of Texas, 1900.

oils usually have a low specific gravity. The specific gravity of some American oils according to Oliphant[1] is in deg. Bé.: Pennsylvania 46 to 43; Ohio 43 to 32; Kansas 39 to 10; West Virginia 38 to 30; Beaumont, Texas, 25 to 31; Wyoming 23 to 1?, California 22 to 12. The higher the specific gravity or the lower the degree Bé., the more viscous the oil and the lower the calorific power.[2] Petroleum slowly volatilizes in part when exposed to air, as the lighter C_xH_y pass off; the specific gravity increases correspondingly, the oil becomes less mobile and finally viscous or solid. The solidifying point of crude oil ranges from +28 to −18° C., and is usually below 0° C.; the flash point (the lowest temperature at which inflammable vapors are given off) from 160 to −18° C., and is usually room temperature, 18° C.; the boiling-point 82 to 170° C. (Pennsylvania oils 74 to 82° C.); the burning point (the temperature at which by continued heating oil takes fire and burns) lies 10 to 15° C. above the flashing point. If the two lie close together, the oil is homogeneous; if not, it is a mixture of light and heavy fractions.

The ultimate analysis[3] gives C 79.5 to 88.7, H 9.6 to 14.8, S 0.07 to 2.00, O 0.9 to 3.2, N 0.008 to 1.10. The calorific power is about 10,000 calories.

TABLE 124.—ULTIMATE ANALYSES OF PETROLEUM

Kind and locality	Bé.	C.	H.	O.+N.	S.	Cal. power
Heavy oil, (a)Pennsylvania	28	84.9	13.7	1 4	......	10,672
Light oil, (a)Pennsylvania	39	82.0	14.8	3.2	.....	9,963
Heavy oil, (a)West Virginia	30	83.5	13.3	3.2	......	10,180
Light oil, (a)West Virginia	36	84 3	14.1	1.6	.....	10,223
Heavy oil, (a)Ohio	28	84.2	13.1	2.7	......	10,399
Rothwell, (a)Canada	33	84 3	13.4	2 3	......	11,399
———, California	....	86.9	11.8	1.1	.. ...	11,728
———, (b)California	15	81.5	10 0	6.9	0.55	10,360
Beaumont, (c)Texas	22	84.6	10.9	2 9	1.63	10,578
Heavy oil, (a)Baku	17	86 6	12.3	1.1	......	10,800
Light oil, (a)Baku	28	86.3	13.6	0.1	......	12,650
Refuse, (a)Baku	00	87.1	11.7	1.2	......	10,700

(a)Poole, H., "Calorific Power of Fuels," Wiley, New York, 1900, pp 251, 252.
(b)Peckham, *op. cit.*, p. 53.
(c)Melville, "Report U. S. Naval Fuel Board," Washington, 1904, pp. 68, 69.

RATIONAL ANALYSIS OF PETROLEUM.—The rational analysis shows that petroleum consists essentially of two series of C_xH_y: The methane (paraffine) series, C_nH_{2n+2} and the ethylene (olefine) series, C_2H_{2n}. Most American oils belong to the methane series; the oils from Baku belong to the ethylene series.

[1] Ries, H., "Economic Geology of the U. S., " Macmillan, New York, 1905, p. 41.
[2] Shermann-Kropf, *Braunkohle*, 1908, VII, 660.
Penn, *Min. Sc. Press*, 1909, XCIX, 874.
[3] Mabery, *Am. Chem. J.*, 1897, XIX, 419; 1905, XXXIII, 251.

TABLE 125.—FRACTIONAL DISTILLATIONS OF PETROLUEM.

Fraction		Appalachian field		Lima-Indiana field		Kansas	Colorado	Wyoming (Popoagir)	Texas			Louisana (Jennings)	Alaska	California			Baku (Russia)	Can-ada
° C.	Product								Cor-sicana	Beaumont				Heavy	Light	Aver-age		
										No. 1	No. 2							
Up to 150	Gasoline....	1.5	......	......	......	......	....	......	.....	.....	.....	.....	....	trace—	16	10	....	...
	Naphtha.........	13.0	20 1	10.9	9.75	18 0	10 0	2-5	10 8	6.01	1.79	.....	....	9 20	—20		5-10 6	6 25
150-300	Illuminating oil....	54.0	61.4	48 8	37 13	30.0	36.0	30-40	54 5	41.09	27.44	41	13 2	6-24 60	47-48	40	32-53.5	50 39
300+	Residuum	....	6.3	23.1	51.63	27 0	54.0	.... .	34.7			59	... ...	7.00-51.50	13-10	10	36-60	15 69
	Heavy and lubricating oils.	17.5	7.1	17.2	......	25.0	....	35-50	...	45.99	64.84	...	58.8	....	22	40	... .	8 47
	Paraffine..........	2.0	(3 1)	....	......	.....	.....	3-5	...			...	22 4	....	. .	....	....	19 66
	Coke.............	10 0 including loss	......	....	......	.....	.....	7-10	......	3 24	4.02	.....	5 6 including loss	....	.....	....	....	.. .
	Loss.............	....	5.1	in residuum	1.49	.....	.....	10-12	......	.....	.....	.....	.. ..	....	.....	....	..... .	......
Reference		Peckham, *op. cit.*, p. 165.	Oliphant, *Twelfth Census, U. S.*, "Mines and Quarries," p. 756.	Oliphant, *op. cit.*, p. 756.	Richardson, *J. Frankl. Inst.*, 1906, CLXII, 82.	Oliphant, *op. cit.*, p. 756.	*Ibid.*	*Bull.* No. 2, University of Wyo., 1897, p. 32.	*Bull.* No. 5, University of Tex., 1900, p. 42.	*Min. Ind.*, 1904, XII, 350.	*Ibid.*	Richardson, *J. Frankl. Inst.*, 1906, CLXII, 122.	*Op. cit.*, p. 124.	Oliphant, *Twelfth Census, U. S.*, "Mines and Quarries," p. 757.		*Eleventh Census, U. S.*, "Mineral Industries," p. 429.	Redwood, *op. cit.*, I, 211.	*Report Bureau of Mines*, Ontario, 1901, p. 17.

The products of fractional distillation[1] may be roughly divided into three groups: Light oils, which distil at 150° C. or less; illuminating oils, passing over at 150 to 300° C.; residuum, remaining behind at 300° C. Light oils upon further fractioning yield petroleum ether (40 to 70° C., 85 to 82° Bé.); gasoline (70 to 80° C., 88 to 80° Bé.); and petroleum spirit (a series of three naphthas, called C-, B- and A-naphthas, 80 to 150° C., 80 to 60° Bé.); they are used as solvents for resin, oil, grease; used in gas engines and benzine lamps, and as dryers for paints, etc. The illuminating oil (56 to 32° Bé.) is purified and forms the commercial burning oil kerosene.

The residuum (marzut of Baku) $>$38° Bé, is used as liquid fuel in reverberatory and crucible furnaces, or is further fractioned yielding heavy oils (58 to 33° Bé., used for lubricating); paraffine oil, 33 to 16° Bé. (paraffine used in manufacture of candles), and coke (manufacture of carbon electrodes).[2]

TABLE 126.—ULTIMATE ANALYSIS AND CALORIFIC POWER OF PETROLEUM COKE

	Orsowa, Hungary	Celle-Wietze, Germany
Specific gravity	1.3698	1.3550
C	91.72	87.44
H	3 64	3.37
S	0.88	1.40
O.+N.	3 05	3 29
H_2O	0 53	1.40
Ash	0.18	3.10
Total	100 00	100 00
F. C.+ash	93 92	92.96
Cal. power (det. in bomb), calories	8,496.	8,042.

The microstructure of petroleum coke has been studied by Roush.[3]

138. Use of Oil as Fuel.[4]—This has many advantages over solid fuel.

[1] Aisinman, S., "Die Destructive Destillation in der Erdölindustrie," Enke, Stuttgart, 1900.

[2] Prunier-Varenne, *Bull. Soc. Chim.*, 1880, XXXIII, 567; *Compt. Rend.*, 1880, XC, 1006.

[3] *J. Ind. Eng. Chem.*, 1912, III, 368; *Metallurgie*, 1912, IX, 166.

[4] Tweddle, "Crude Petroleum and its Products as Fuel," *Eng. Min. J.*, 1899, LXVIII, 459, 517.

Phillips, *Bull.*, Univ. of Tex., No 5, 1900, p. 80.

Williston, "Liquid Fuel for Power Purposes," *Eng. Mag.*, 1903, XXV, 237, 562, 721.

Melville, "Report U. S. Naval Liquid Fuel Board," Washington, 1904 (*Summary, Min. Ind.*, 1902, XI, 501).

Aubry, *Bull.* No. 32, State Mineralogist of Cal., Sacramento, 1904, pp. 58–160.

North, S. H., "Oil Fuel," Griffin, London, 1905.

Booth, W. H., "Liquid Fuel and its Apparatus," Dutton, New York, 1912.

Allen, I. C., Specifications for the Purchase of Fuel Oil by the Government, *Technic. Paper*, No. 3, Bureau Mines, Washington, 1911; also *Cass. Mag.*, 1911, XL, 383.

For the same fuel value the weight and volume of oil is 60 to 70 per cent. that of good bituminous coal; there is next to no ash and less labor; starting, regulating and stopping of fire are easily managed; there is no dirt. The disadvantages, such as danger[1] of explosion, loss by evaporation, odor, etc., are small. The crude oil of Texas, Louisiana and California is used extensively in the Southwest and West as a boiler and metallurgical fuel. Residuum serves commonly as an industrial fuel. This is especially the case with oil from the Lima-Indiana and Colorado fields. The greatest development of burning liquid fuel is that of eastern Europe and southern Asia where the residuum (mazut) or natural heavy oil (astaki) of Baku is shipped by river in steamers as far north as St. Petersburg and Moscow to metallurgical and general industrial plants. In the east of the United States very little residuum is burnt as fuel on account of its high price.

Residuum or heavy oil being viscous must be heated to 40 or 50° C. before it can be satisfactorily atomized. Compressed air used for this purpose is generally preheated; steam of high pressure is more advantageous than steam of low pressure; a combination of steam for heating and air for atomizing gives the most satisfactory results. Details are taken up in §153 in which apparatus for burning liquid fuels is described. The working of oil-bearing shales, of iron blast-furnace tar, etc., belong to industrial organic chemistry.[2]

139. Gaseous Fuel in General.[3]—Exudations of gas from the earth were known in ancient times (Holy fires of Baku). Natural gas was used for illuminating purposes in 1667 at Wigan, England, in 1821 at Fredonia, N. Y.; brine was evaporated with it in the seventeenth century in China, and in 1841 in the Kanawha Valley, W. Va. The beginning of the industrial use of artificial gaseous fuel was made in 1832 by Faber du Faur[4] at the iron works of Wasseralfingen, Germany. He withdrew the gas, usually burning at the top of the iron blast-furnace, from below the throat and used it for puddling and for heating the blast. Today gaseous fuel is not uncommon in industrial heating, and its application for this purpose is increasing. The reasons for this are that gaseous fuel offers many advantages over solid and even over liquid fuel. The leading ones are: (1) That perfect combustion can be obtained with a small excess of air over that required by theory on account of the intimate mixing of gas and air. (2) That a high temperature can be obtained on account of this nearer approach to perfect combustion, and the smaller volume of products of combustion which draw off heat. (3) That air and, as a rule, gas can be preheated with the waste

[1] *Cass. Mag.*, 1899, XVII, 21.

[2] Lunge, G., "Coal Tar and Ammonia," Gurney, London, 1900.

Redwood, B., "A Treatise on Petroleum," Griffin & Co., London, 1913.

Ells, R. W., "Joint Report on the Bituminous or Oil-shales of New Brunswick and Nova Scotia, also, on the Oil-shale Industry of Scotland," *Canada Department of Mines*, Nos. 55 and 1107, Ottawa, 1910.

Baskerville, "Distillation of Oil-shale," *Eng. Min. J.*, 1909, LXXXVIII, 149, 196.

[3] Ledebur, A., "Gasfeuerungen," Felix, Leipsic, 1891.

Steinmann, F., "Compendium der Gasfeuerung," Felix, Leipsic, 1900.

[4] Delesse, *Ann. Min.*, 1842, I, 433; Baur, *Stahl u. Eisen*, 1904, XXIV, 562.

TABLE 127.—COMPOSITIONS OF GASEOUS FUELS

Kind	C_6H_6	C_2H_4	CH_4	H	CO	CO_2	N	O	H_2S	Heat value		Reference
										Cal. per cbm.	Per cent.	
Natural gas, Ohio and Indiana..	.....	0.4	93.0	2.0	0.6	0.3	3.0	0.5	0.2	8,120	100	*Eighth Ann. Rep. U. S. Geol. Survey*, 1896–97, p. 592.
Oil gas..........................	......	16.5	48.0	32.0	......	.....	3.0	0.5	..	7,337	90	*J. Soc. Chem. Ind.*, 1905, XXIV, 299.
Coal (bench, illuminating) gas..	......	5.0	40.0	46.0	6.0	0.5	2.0	0.5	....	5,333	70	*Tr. A. I. M. E.*, 1889–90, XVIII, 881.
Coke-oven gas...............	0.5	1.5	36.0	54.0	6.5	1.0	.....	....	0.5	5,103	63	*Iron Age*, 1895, LV, 588.
Water gas....................	......	......	2.0	45.5	45.5	4.5	2.0	0.5	....	2,738	40	*Tr. A. I. M. E.*, 1889–90, XVIII, 881.
Producer gas, bituminous coal	....	0.4	2.5	12.0	27.0	2.5	56.2	0.3	....	1,402	17	*Tr. A. I. M. E.*, 1889–90, XVIII, 881.
Mond gas....................	.....	.....	2.0	29.0	11.0	16.0	42.0	.....	...	1,266	15	*Coll. Guard.*, 1901, LXXXI, 351.
Producer gas, anthracite coal..	......	......	1.2	12.0	27.0	2.5	57.0	0.3	.....	1,233	15	*Tr. A. I. M. E.*, 1889–90, XVIII, 881.
Iron blast furnace gas.........	......	......	2.0	2.0	24.0	12.0	60.0	.....	.....	959	12	Ledebur, *Eisenhuttenkunde*, 1903, 108.

heat of the furnace. (4) That the control of admission of gas and air permit maintaining a uniform character of fire with a uniform temperature. (5) The gas can be produced either at the furnace where it is to be burnt, or at a centrally situated place and then piped to the different furnaces. (6) That an inferior fuel may be used. (7) That gas may be used for generating power by means of gas engines with a high degree of efficiency.

In metallurgical plants gaseous fuel will be employed mainly in furnaces working continuously. Formerly it was used principally in high-temperature (smelting, reheating) furnaces; at present it is making its way in low-temperature (roasting) furnaces from which the products of combustion pass off at a temperature of less than 300° C.

The leading gases are natural gas, coal gas, producer gas, iron blast-furnace gas, water gas and oil gas. Table 127 giving average analyses of these gases is arranged according to heat values.

140. Natural Gas.[1]—In the United States natural gas was first put to economic use in 1821 at Fredonia, N. Y.; in 1875 it was piped the first time for any considerable distance (Butler Co., Pa.); the piping of natural gas from the Murraysville district into Pittsburgh, Pa., in 1883 marks the beginning of its extensive use there. In Ohio, natural gas was discovered at Findlay in 1885, and closely followed by its discovery in Indiana. Natural gas became of industrial importance in southeastern Kansas in 1892. Table 128 gives the total industrial consumption of natural gas in the United States in 1910.

The table shows that Pennsylvania still remains in the lead. West Virginia, Kansas, Ohio and Oklahoma form the leading producers at present. The average price per 1,000 cu. ft. is 8.63 cents.

TABLE 128.—INDUSTRIAL CONSUMPTION OF NATURAL GAS IN THE UNITED STATES IN 1910 (Million cu. ft.)

Item	Pennsylvania	Ohio	Kansas	West Virginia	New York	Oklahoma	Indiana	Kentucky	Illinois	Louisiana Texas Alabama	California	Arkansas Colorado Wyoming	South Dakota	Missouri	North Dakota	Michigan
Quantity mill. cu. ft.	125,471	47,535	58,137	63,431	1,947	2,2482	1,445	2,384	5,457	6,494	2,519	1,982	20	24	1	1
Cost, cents, per 1,000 cu. ft.	10.3	12.8	6.8	5.1	16.3	5.2	12.1	8.0	6.1	6.8	11.2	6.1	69.5	25.0	40	50

[1] *Eleventh Census U. S.*, 1890, "Mineral Industries," p. 505; *Twelfth Census*, 1900, "Mines and Quarries," p. 767; *Min. Resources U. S.*, 1892 and following; *Eighth and Twentieth Ann. Reports U. S. Geol. Survey.*

Natural gas in the ground is, as a rule, under considerable pressure: 700 lb. per sq. in. in Pennsylvania; 450 in Ohio and 500 in Indiana; these figures have decreased, as the gas in being consumed, to 50 in Pennsylvania, 20 in Ohio and 70 in Indiana. The gas flows by its own pressure; if the distance is too great or the pressure too low, it is pumped and forced onward by compressors. The gas of Butler Co., Pa., with an initial pressure of 119 lb. per sq. in. traversed 17 miles of 6-in. pipe in 20 minutes.[1] A pumping plant[2] in Indiana receives gas at 100 lb. pressure, compresses it to 300 lb. and delivers 1,500,000 cu. ft. per hour through a 10-in. pipe to Indianapolis, 30 miles away, at a pressure little above atmospheric.[3]

Natural gas contains 90–97 per cent. CH_4; the remainder is N, H, CO, CO_2, O and H_2S, as seen by the analyses given in Table 129.

TABLE 129.—ANALYSES OF NATURAL GAS

Constituent	Pennsylvania			Indiana		Ohio		Kansas		New York	
	Kane, Mc-Kean Co.[4]	Lyons Run, Murraysville[4]	Average[5]	Muncie[6]	Kokomo[6]	Fostoria[6]	Findlay[6]	Iola[7]	Assawatamie[7]	Fredonia[4]	West Bloomfield
H	....	....	5–20	2.35	1.42	1.89	1.64	....	....	....	....
CH_4	90.01	97.70	60–80	92.67	94.16	92.84	93.35	89.66	97.63	90.05	....
ill. C_xH_y	trace	trace	0–8	0.25	0.30	0.20	0.35	....	0.22	trace	2.94
CO	0.20	0.20	trace	0.45	0.55	0.55	0.41	1.23	1.33	0.41	10.11
CO_2	....	....	0.3–2	0.25	0.29	0.20	0.25	0.90	0.22	....	82.41
O	trace	....	....	0.35	0.30	0.35	0.39	0.45	trace	....	0.23
N	9.79	2.02	1–12	3.53	2.80	3.82	3.41	7.76	0.60	9.54	4.31
H_2S	....	....	....	0.15	0.18	0.15	0.20	....	....	....	....

Gas from Pennsylvania is frequently without odor; gas from Ohio and Indiana contains much H_2S. The analyses show that natural gas is a heating gas and that its illuminating power is weak. The calorific power of 1 cbm. is 8,000 to 10,000 Cal. Upon superheating or in imperfect combustion natural gas deposits hard, black, glossy, sonorous carbon, the "Natural-gas Coke" or "Gas Carbon" which being absolutely free from grit is manufactured into lamp-black, printer's ink, paint, also into electric-light carbons.

[1] *Mineral Resources U. S.*, 1892, p. 656.

[2] *Cass. Mag.*, 1897–98, XIII, 293.

[3] Liquefied Products from Natural Gas: Allen, I. C., and Burrell, G. A., *Technic. Paper* No. 10, Bureau Mines, Washington, 1912.

Leland, *Eng. Min. J.*, 1912, XCIV, 991.

[4] *Min. Res. U. S.*, 1892, 653.

[5] *Tr. A. I. M. E.*, 1886–87, XV, 11.

[6] *Eighth Ann. Rep. U. S. Geol. Surv.*, 1896–97, 582.

[7] *Min. Resources of Kansas*, 1897, p. 52.

141. Coal-gas and By-product Coke-oven Gas.—Coal gas, first produced by Murdock in 1798, results from the destructive distillation of bituminous coal. The coal is distilled in fire-clay retorts that are fired with coal, coke or producer gas. A retort is about 26×16 in. in cross-section, 8 to 10 ft. long, and has walls 2.5 to 3 in. thick; it is charged with 400 to 500 lb. coal; the distillation lasts four or more hours. The coke produced, 15 to 25 per cent. of the weight of the coal, is porous on account of the small height of the charge which, when in a semifused state, is swelled by the gases. An analysis of coal (bench-illuminating) gas was given in Table 127. There is little difference in composition and hence in heating value between bench- and coke-oven gas. Bench-gas as a rule runs a little lower in H and higher in CH_4 than coke-oven gas; its illuminating power is greater (18 to 20 *vs.* 10 to 12 candles) on account of the higher percentage of C_2H_4. One net ton of gas-coal yields about 10,000 cu. ft. bench-gas. Illuminating gas on account of the cost of production is too expensive to be used as an industrial fuel. This is not the case with the gas from by-product coke-ovens. The relation of cost of the two processes is shown in the Tables 130 and 131.[1] In Table 130 the calorific value of 100 lb. of Cape Breton coal is traced through the products obtained in an Otto-Hoffmann by-product coke-oven; in Table 131 these figures are compared with the data of average German gas-house practice.

TABLE 130.—COMPARATIVE COSTS OF BENCH AND BY-PRODUCT COKE-OVEN GAS

100 lb. dry Cape Breton coal yield—	B.t.u. (per lb.)	Total calorific power (B.t.u.)	Per cent. of calorific power of dry coal
71.13 lb. coke	12,645	899,456	72.3
3.38 lb. tar	12,210	51,410	4 1
229.6 cu. ft. surplus gas	686	157,504	12.7
234.2 cu. ft. heating gas	567	132,835	10.7
Ammonia liquor, sulphur in purifier, and loss.	....	2,496	0.2
Total=100 pounds dry coal	....	1,243,700	100.0

Table 131 shows that in the by-product coke-oven practice 10.7+0.2=10.9 per cent. of the heat value of the coal is consumed in destructive distillation, while gas house practice requires 10.1+17.0=27.1 per cent.

[1] Hofman, *Eng. and Min. J.*, 1898, LXVI, 460.

TABLE 131

The heat contained in 100 pounds dry coal is distributed as follows:	Per cent. of total heat value of coal	
	German coal in gas retorts	Cape Breton coal in Otto-Hoffmann ovens
In coke, salable	46.4	72.3
In coke used for heating retorts	10.1	....
In tar	5.5	4.1
In gas, salable	21.0	12.7
In gas for heating ovens	....	10.7
In ammonia liquor, sulphur in purifiers, and loss.	17 0	0.2
Total	100.0	100 0
Heat used and lost in distillation process	27 1	10 9
Heat contained in products	72 9	89 1
Total	100.0	100.0

142. Producer Gas.[1]—Producer gas is the result of imperfect combustion of C by means of air. It is made in a producer, generally a low, shaft-like furnace, holding a deep bed of incandescent fuel, through which air or air and steam is drawn or forced. The results of the experiments of Ernst,[2] plotted in Fig. 182, show that C heated in air begins to be oxidized at 400° C. and burns mainly to CO_2; the formation of CO_2 increases with the temperature up to 700° C., when that of CO grows up to 995° C.; above this temperature no more CO_2 is formed. A producer run on C at 1,000° C.[3] ought to furnish a mixture of CO and N. The composition of such a gas would be as follows:

[1] Deschamps, J., "Les Gazogènes," Dunod, Paris, 1902.

Sexton, A. H., "Producer Gas," Scientific Publishing Co., Manchester (England), 1905.

Wyer, S. S., "A Treatise on Producer Gas and Gas Producers," *Eng. and Min. J.*, New York, 1907.

Jüptner, H. von., "Beiträge zur Theorie des Generator (oder Luft-) und des Wasser-Gases," Enke, Stuttgart, 1904.

Dowson, J. E., and Larter, A. T., "Producer Gas," Longmans, Green & Co., London, 1912.

Fischer, F., "Kraftgas, seine Herstellung und Beurteilung," Spamer, Leipsic, 1911.

Campbell, *Tr. A. I. M. E.*, 1890–91, XIX, p. 136.

Lürmann, *Stahl u. Eisen*, 1903, XXIII, 433, 515, 1151, 1154.

Körting, *op. cit.*, 1907, XXVII, 685.

[2] Ernst, *J. prakt. Chemie*, 1893, XLVI, 31; *J. I. and St. I.*, 1893, I, p. 217; *Oest. Zt. für Berg. Hüttenw.*, 1894, XLII, p. 239; *Berg. Hüttenm. Z.*, 1893, LII, 286.

[3] Le Chatelier's measurements gave 720° C. as the temperature of the gas leaving the producer, and 400° C. when in the cooling tube: *Eng. and Min J.*, 1890, L, 429.

TABLE 132.—THEORETICAL COMPOSITION OF PRODUCER GAS

Per cent.	CO	N
By volume	34.57	65.41
By weight	34.59	65.43

In actual work, gas free from CO_2 is not obtained, as producers are not run throughout at 1,000° C., and as the fuel bed is not sufficiently high to prevent any free O from passing which burns some CO. In addition to CO and

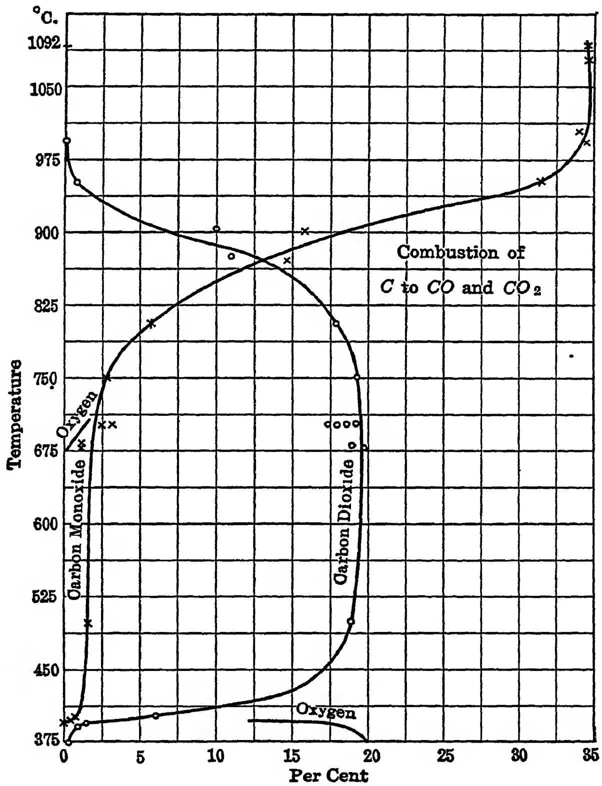


FIG. 182.—Combustion of C to CO and CO_2 as governed by temperature.

N the gas contains some H which comes from the fuel and from the decomposition of the moisture of the air by incandescent carbon: $H_2O+C=H_2+CO$.

Table 133 gives averages of three analyses by Ebelmen using charcoal[1] and coke,[2] they represent a very high grade of gas from carbonized fuel:

[1] *Ann. Min.*, 1843, III, p. 215.

[2] *Op. cit.*, 1844, V, p. 81.

TABLE 133.—HIGH-GRADE PRODUCER GAS FROM CARBONIZED FUEL, PER CENT. VOL.

Gas made from	CO	N	CO_2	H
Charcoal	33.63	63.37	0 45	2 55
Coke	33.31	64.64	0 80	1.25

It is not often, however, that carbonized fuels are used for making producer gas; usually natural fuels—wood, peat, brown coal, bituminous coal, and anthracite—serve for this purpose. In gasifying these, two processes go on simultaneously: distillation and imperfect combustion; and the gas will contain more or less volatile C_xH_y. In the analyses by Gantt,[1] Table 134, the anthracite used contained 5 per cent. V.H–C, and 10 per cent. ash; the bituminous coal, 55 per cent. F.C. and 32 V.H–C. It is presupposed that none of the V.H–C has been allowed to be condensed. If the gas cools, as is always the case when producer and furnace are separate, some of the C_xH_y of the bituminous-coal gas will be condensed and the gas will lose considerable part of its heat value. In fact, the two gases when cooled down will show approximately the same composition.

TABLE 134.—PRODUCER GAS FROM ANTHRACITE AND BITUMINOUS COAL (NO STEAM USED), PER CENT. VOL.

Item	CO	V. H–C	H	CO_2	N
Anthracite	27.79	1 37	....	3.72	67.14
Bituminous coal	23.67	11.88	....	4.01	60.44

Making producer gas is an exothermic process: $C+O=CO+29{,}160$ cal. In its formation 30 per cent. of the heat developed in perfect combustion ($C+O_2=CO_2+97{,}200$ cal.) is obtained. If the gas is cooled down, it loses these 30 per cent. and can develop only the remaining 70 per cent., or 68,040 cal., when burned to CO_2. The gas will be cooled down to some extent if producer and furnace are separate. There will be no loss of heat, except through losses by radiation, etc., from the producer, if this is placed close to the furnace and the gas can enter with its initial heat. As it is in many cases more economical to have the producers separate from the furnaces in which the gases are to be burned, part of the sensible heat of C burning to CO has been utilized by introducing H_2O vapor or CO_2 into the producer.

Water vapor[2] at 100° C. acting upon incandescent C is decomposed into H_2 and CO or CO_2 according to the temperature of the C. The endothermic reaction taking place: at and above 1,200° C. is exclusively: I. $H_2O+C=$

[1] *Cass. Mag.*, 1895–96, IX, 49.

[2] Voigt, *Metallurgie*, 1908, V, 383, 399.

$H_2+CO-28{,}374$ cal.; between 500° and 1,200° C. is mainly: II. $2H_2O+C=2H_2+CO_2-16{,}868$ cal. Gas formed according to equation I has the composition CO 50, H 50 per cent. vol., and according to equation II CO_2 33.3, H 66.7 per cent. vol. In producer gas, there will be found both CO and CO_2.

The difference between the two thermal values shows that the reaction of equation II is less endothermic than that of equation I, while the heat value of the resulting gas from equation I is slightly higher than that from equation II: 128,100 *vs.* 116,130 cal.

TABLE 135.—PRODUCER GAS FROM ANTHRACITE AND BITUMINOUS COAL (STEAM USED), PER CENT. VOL.

Item	CO	V. H–C	H	CO_2	N
Anthracite[1]	30.24	1.49	9.68	4 02	54 57
Bituminous coal[1]	24 00	12.60	8.60	5 30	49.50

The volumes of CO_2 in both gases are high; they should not exceed 4 per cent.; the lowest is 2 per cent., the highest 8 per cent. Beside the above constituents producer gas made from coal always contains some S.

In large producers the sensible heat of the gases and the loss by radiation and conduction amounts to perhaps 10 per cent. of the calorific power of the fuel, leaving 20 per cent. available for the decomposing effect of steam.[2] In small producers the loss is often so high as to forbid the use of steam.[3] While the greatest amount of steam permissible for gasifying C can be calculated, the proportion usually admitted is governed by the temperature of the producer which may not fall below the point at which C will burn largely to CO_2 instead of to CO. The usual proportions are 10 vol. steam to 90 vol. air.

Gautier[4] found that an excess of H_2O vapor acting upon CO at 1,200 to 1,250° C. gave $3CO+3H_2O=CO+H_2O+2H_2+2CO_2$ until the volume of H was double that of CO, and that the reaction then was reversed: $3CO_2+3H_2=CO+H_2O+2H_2+2CO_2$, until the volume of CO was again one-half that of H. A further advantage of steam, beside recovering some of the sensible heat of the C burning to CO and enriching the gas, is that it counteracts to some extent the clinkering of ashes, disintegrates clinkers that have been formed, and keeps the grates cool.

CO_2 acting upon incandescent C is reduced to CO, viz: $CO_2+C=2CO-38{,}800$ cal. As pure CO_2 is not used on a large scale, but part of the products of combustion of a furnace which contain at least 79 vol. N, the gas obtained will be correspondingly diluted. Supposing air to consist of 1 mol. O+4 mol. N,

[1] *Cass. Mag.*, 1895–96, IX, 49.

[2] Ennis, *Tr. Am. Inst. Chem. Eng.*, 1909, II, p. 115.

[3] Bone-Wheeler, *J. I. and St. I.*, 1907, I, 126.

[4] *Compt. rend.*, 1906, CXLII, 1382; 1910, CL, 1564; *Metallurgie*, 1911, VIII, 79; *Met. Chem. Eng.*, 1911, IX, 511.

which is approximately true, then the formation of producer gas can be expressed by $C+[1/2(O)_2+2N_2]=CO+2N_2$. This burnt in air gives: $CO+2N_2+[1/2(O_2)+2N_2]=CO_2+4N_2$. The product of perfect combustion conducted through incandescent C gives: $CO_2+4N_2+C=2CO+4N_2$, a gas of the same composition as the original producer gas. Thus only the heat brought to the producer by the products of combustion would represent a saving over the use of cold atmospheric air, and this heat can be better utilized by superheating the air. CO_2 is therefore little used in making producer gas,[1] whereas there is hardly a large producer run at present without the admixture of steam, at least when the resulting gas is conducted through pipes to the distant furnace where it is to be burned.

Producer gas is made from all sorts of fuels, but some are better suited than others. Wood, peat,[2] and brown coal[3] are sometimes used, but, being very rich in H_2O, require special apparatus for condensing the water vapor. They retain their form while they are being gasified, shrinking slowly until they finally crumble to ashes. Bituminous coal is the leading fuel. Splint coal, which falls to pieces, and coking coal, which fuses, swells, and cakes, are not desirable, as they both obstruct the passages of the gas. Gas, steam, and furnace coals, which cake only slightly and are rich in volatile matter, form the most desirable fuel. Nut coal is the most suitable, but run of mine, slack, or a mixture of the last two is generally used. The coal ought to run low in ash and water. Water vapor, if present in abundance in the gases, beginning at 600° C.,[4] will have a decomposing effect upon CO, viz., $H_2O+CO=2H+CO_2$. One ton Pittsburgh coal gives about 150,000 cu. ft. of producer gas. Anthracite, semi-anthracite, and brown coal have a tendency to decrepitate in the producer and thus choke the draught. The most suitable anthracite is No. 1 buckwheat (over 3/16 to 3/8 and under 3/8 to 5/8 in. holes); it ought to run low in ash, which must be difficult to fuse, and high in volatile matter. Anthracite in the form of culm or poorly prepared buckwheat is not suitable. One gross ton buckwheat anthracite gives

[1] Head-Pouff, *J. I. and St. I.*, 1889, II, 256.
Åckerman, *op. cit.*, 1890, I, 24.
Schoeffel, *Oest. Zt. Berg. Huttenw.*, 1891, XXXIX, 212, 225.
Hempel, *Verh. Ver. Bef. Gewerbefl.*, 1891, LXX, 77.
Schmidhammer, *Stahl u. Eisen*, 1907, XXVII, 558.

[2] Frank, *Electrochem. Met. Ind.*, 1907, V, 405.
Haanel, *Report*, Utilization of Peat in Gas Producer, Ottawa, 1912.
Bartel, F., *"Tortkraft," Springer, Berlin*, 1913.

[3] Anon., *Thonind.Z.*, 1907, XXXI, 1560, 1586.
H. L., *Braunkohle*, 1911, IX, 837, 855, 869; X, 97, 113, 129.
Randall, D. T., and Kreisinger, N., *Bull.* 2, Bureau Mines, Washington, 1910.

[4] See Gautier, p. 297.
Blass, *Stahl u. Eisen*, 1886, VI, 4.
Würtenberger, *op. cit.*, 1903, XXIII, 447.
Naumann-Pixtor, *Ber. deutsch. Chem. Ges.*, 1885, XVIII, pp. 1647, 2724, 2894.
Haber, *Zt. Electrochemie.*, 1904, X, 861
Jüptner, *op. cit.*, 862.
Boudouard, *Ann. Chim. Phys.*, 1901, XXIV, 53.
Coetz, *Tr. A. I. M. E.*, 1889–90, XVIII, 612.

about 170,000 cu. ft. gas. Charcoal and coke are not used in producers, being too expensive. They are, however, a very desirable fuel, and the gas being practically free from volatile matter, can be better piped than that made from natural fuels.

143. Gas Producers.[1]—It is difficult, if not impossible, to find a classification that will include all the forms of producers that have been constructed. In all producers with one exception, distillation and gasification take place in a single chamber; in the Gröbe-Lürmann,[2] Bulmahn,[3] and Westinghouse[4] producers the two processes take place in separate chambers or stages. Of the producers of the first class some are worked by natural draught, others[5] by forced or induced draught (suction gas producers), with or without steam; some have a horizontal or inclined grate, others have a solid bottom, which in some cases is stationary, in others movable; many modern producers have mechanical stirrers; some producers have a water seal, others are worked dry, etc. The producer may be placed close to or form part of the furnace which it supplies with gas, or a number of producers may be grouped together and the gas conducted through one or more mains to the different furnaces of a plant.

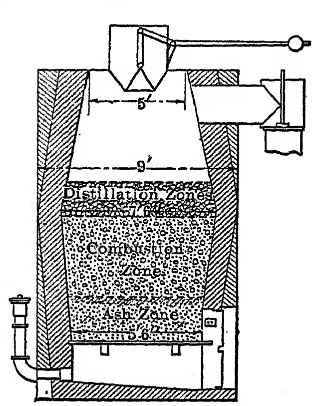


FIG. 183.—Philadelphia Engineering Works or Wellman gas producer.

In all single-chamber producers three zones (Fig. 183) can be distinguished: the ash zone at the bottom, followed by the combustion, and the distillation zone. The function of the ash zone is to heat air and steam. As the ashes are generally (excepting in the Kitson producer) not cleaned out continuously, they accumulate. This has to be kept in mind in regulating the depth of the fuel bed. The combustion zone occupies the greatest depth of the charge; the distilation zone comprises only the upper part of the fire.

Air, warmed by passing through the ash zone, meets upon entering the combustion zone coke at a relatively low temperature and converts the C mainly into CO_2, viz., $C+O_2+4N_2=CO_2+4N_2+97,200$ cal. The CO_2 formed, rising in the furnace, passes through the coke at a relatively high temperature and is reduced more or less to CO, viz., $CO_2+4N_2+C \rightleftarrows 2CO+4N_2-38,880$ cal. The reaction is reversible, as CO_2 conducted through a carbon tube at about 700° C.

[1] Rowan, *Proc.* (*Eng.*) *Inst. Civ. Eng.*, 1886, LXXXVI, pp. 2 to 113.

[2] Ledebur, "Gasfeuerungen," Felix, Leipsic, 1891, p. 60.

Fischer, "Chemische Technologie der Brennstoffe," Vieweg, Brunswick, 1901, II, p. 295;

[3] *Met. Chem. Eng.*, 1910, VIII, 159.

[4] *Iron Age*, 1909, LXXXIII, 1916.

[5] Mathot, R. E., "Gas Engines and Gas Producer Plants," Henley, New York, 1905.

Robson, P. W., "Power Gas Producers," Arnold, London, 1908.

Latta, N., "American Gas Producer Practice," Van Nostrand, New York, 1910.

Fernald, R. H., "The Status of the Gas Producer and of the Internal Combustion Engine," Bureau Mines, *Technical Paper* No. 9, Washington, 1912.

is partly reduced to CO, and 2CO treated in the same way is split in part into CO_2 and C. The fact that the equation read from left to right is endothermic proves that a high temperature is necessary to overcome the heat absorbed by the reaction; the researches of Ernst (Fig. 182) have shown that a high temperature is necessary for C to burn completely to CO; Naumann-Pistor[1] conducting CO_2 over charcoal obtained 12.6, 58.3, 92.2 vol. CO at 550, 900, 1,000° C. The reduction of CO_2, however, is never complete because an equilibrium is set up between CO_2 and CO. Boudouard[2] found that the reduction stopped at 650° C. when the gas mixture had been changed to 61 per cent. CO_2+39 per cent.

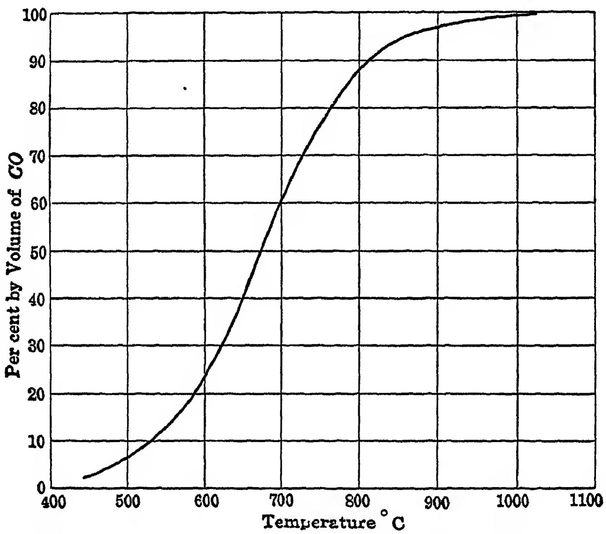


FIG. 184.—Equilibrium curve for CO_2 and CO with C at different temperatures.

CO, at 800° C. with 7 per cent. CO_2+93 per cent. CO, at 950° C. with 4 per cent. CO_2 and 96 per cent. CO. His equilibrium curve[3] for varying proportions of CO_2 and CO with C at different temperatures is given in Fig. 184.

The speed of reaction, which is greater at 800 than at 650° C., is further influenced by the porosity of the C or its fine state of division. Boudouard's results, plotted in Fig. 184*a*, show that both at 650 and 800° C. deposited C does not act as energetically as charcoal; that at 800° C., gas-house coke is less active than blast-furnace coke, and that the latter has less reducing power than deposited C or charcoal. The reversible reaction above read from left to right shows that 1 vol. CO_2 forms 2 vol. CO; therefore if the pressure in the producer is high, less CO will be formed than if it is low; or, forcing the producer weakens the gas. The height of the fuel bed,[4] finally, may not go below a certain mini-

[1] *Ber. deutsch. Chem. Ges.*, 1885, XVIII, 1647.

[2] *Ann. chim. et phys.*, 1901, XXIV, 28.

[3] See also Wheeler, *Eng. Min. J.*, 1912, XCIV, 65.
Clement, *Bull.* 30, University of Ill.; *Stahl u. Eisen*, 1909, XXIX, 1324.

[4] Körting, *Stahl u. Eisen*, 1907, XXVIII, 685, 800.

mum if CO_2 is to be satisfactorily reduced to CO, and unconsumed air, steam, or both prevented from entering the gas chamber. It varies with the fuel, and is the greater the lighter the fuel; thus wood will have 10 ft., peat 8 ± ft., lignite and bituminous coal 3 1/2 to 4 1/2 ft. (sometimes, however, 6 ft.), anthracite 2 1/2 to 3 ft. If with bituminous coal the fuel bed exceeds a depth of 6 ft., "the lower part[1] is not thoroughly broken up, no matter how thoroughly the upper part may be stirred; the zone of combustion becomes honeycombed with large cavities, affording passage for undecomposed steam and air. This condition is most marked along the walls, and the intense heat produced on the interior surfaces of these coke chimneys causes clinkers to fuse to the brickwork. Practice therefore demands that the thickness of the fire be limited, that steam be used to avoid extremes of temperatures, and that the mass be kept thoroughly broken by frequent poking."

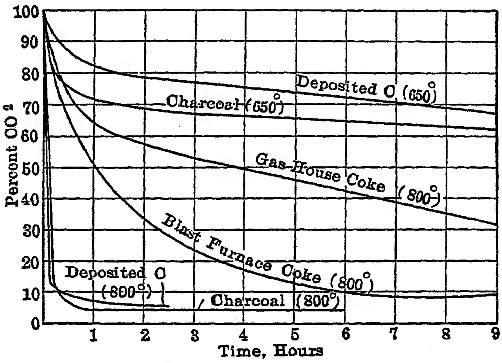


FIG. 184a.—Influence of form of carbon upon the reduction of CO_2.

Upon leaving the combustion zone, the producer gas enters the distillation zone[2] in which it expels the volatile matter of the raw fuel and carries it along. The distillation absorbs heat and reduces the temperature, a condition favorable for the formation of CO_2. The composition of volatile matter will vary with the temperature of distillation just as is the case in charring (§114) and coking (§129). Condensation of tarry matter, when the gas cools in its passage from the producer to the furnace, causes loss in calorific power, which, however, is not high[3] as long as the suspended C_xH_y are not completely removed. These make the flame luminous, and a luminous flame heating by impact as well as by radiation, is more powerful than a clear flame.

Producers in the United States are not often run exclusively by natural draft; as a rule a steam blower furnishes the blast, unless the ash is to be fused and tapped as slag,[4] when fans are employed. The Eynon-Evans Steam blower[5] shown in Fig. 185 is the injector in general use. Steam enters the smallest of the four concentric nozzles, draws in air and becomes thoroughly mixed

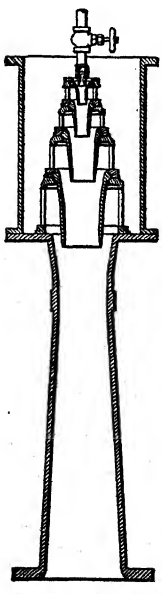


FIG. 185.

[1] Campbell, *Tr. A. I. M. E.*, 1893, XXII, 372.

[2] *Ibid.*, p. 376.

[3] Bell, *Engineering*, 1908, LXXXV, 141, 171.

[4] Smith, C. D., "The Slagging Type of Gas Producers," *Technical Paper* No. 20, Bureau Mines, Washington, 1912.

[5] Improved Blowers for Gas-producers, *Iron Age*, March 6, 1902.

with it. The gas from a steam-blown producer, a mixture of simple producer gas and water gas, is sometimes called Dowson Gas. The gas leaves the producer at a temperature of 300 to 1,000; average 700° C.

144. Examples of Gas Producers.[1] (1) THE SIEMENS PRODUCER (Fig. 186).—This was the first commercially successful gas producer. It is a rectangular fire-brick chamber with one side inclined at an angle of 45 to 60° and provided at its foot with a step grate ending in the inclined grate *B*. *A* is the charging hole; *E* are poking and peep holes; the gas passes off at *C*, its flow being regulated by dampers; four producers form a block; the gases from the four vertical flues *C* unite in the main flue *D*. The area of each producer is at the grate 6×3 ft., and at the top 9×7 ft.; the producer holds 10 to 12 tons coal and gasifies 180 lb. per hr. or 10 lb. per hr. per sq. ft. of grate area. It is worked with natural draft without the use of steam; the gas is essentially a hot gas, which cools considerably in the main before it reaches the furnace where it is to be burned. The efficiency is therefore low. The original Siemens furnace has become obsolete. Where used to-day, it is worked with a closed ash pit and a steam injector. Swindell[2] has modified the Siemens producer by closing the ash pit, introducing a steam injector, and providing it with a water seal and a shaking grate.

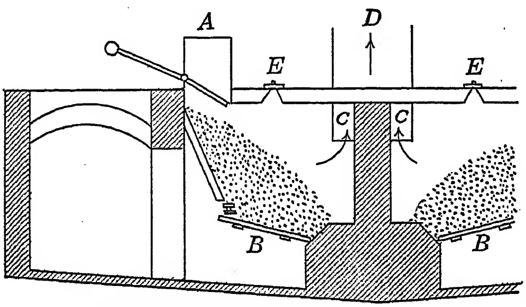


FIG. 186.—Siemens gas producer.

(2) THE PHILADELPHIA ENGINEERING WORKS, OR WELLMAN PRODUCER (Fig. 183).[3]—This is a circular, wrought-iron shell, 14 ft. high and 9 ft. diam., lined with fire-brick and red brick. The producer has a bosh, 7 ft. 6 in. diam., narrowing to 5 ft. at the top, to prevent air from creeping up at the walls. The charge rests on wrought-iron grate bars; the grate is 6 ft. 9 in. ×6 ft. 1 in. The front is closed by two pairs of doors, one above the other. When drawing ashes, a false grate is put in above the bearing bars by pushing bars through the fire above the ashes; they support the charge while the lower bars are being drawn and the ashes and clinkers dropped into the closed ash pit. While cleaning, the damper in the gas-outlet pipe is closed to prevent gas from backing down through the fire onto the workmen. The coal is charged at the center through bell and hopper; a steam blower below the grate furnishes blast and steam; the gas is drawn off at the side. The capacity is 420 lb. coal per hr., or 10 lb. per hr. per sq. ft. grate area.

[1] Wolff, *Stahl u. Eisen* 1905, XXV, 387.
Körting, *op. cit.*, 1907, XXVII, 685 and books quoted on pp. 289 and 293.
[2] *Cass. Mag.*, 1895–96, IX, 51.
[3] Campbell, *Tr. A. I. M. E.*, 1893, XXII, 372.

(3) The Duff Water-seal Producer (Fig. 187).[1]—This is a cylindrical gas producer with water-seal. It has a gabel-shaped central grate and an under-grate steam blast. The outer shell, of 3/16-in. steel, has a diameter of 10 ft. 6 in. and a height of 11 ft.; it is lined with red brick and fire-brick so as to give a square shaft 7×7 ft. The water-seal pan, 15 ft. 4 in. long ×7 ft. wide ×2 ft. 3 in. deep, is of 1/4-in. steel. The gas-outlet pipe (36 in. outer diam.) is lined with brick (27 in. inner diam.). The top has a bell and hopper feed and the usual poking holes. The capacity claimed is 1120 lb. Pittsburgh coal per hr. or 23 lb. per hr. per sq. ft. grate area. The advantage of a water-seal is that the ashes, cold when taken out, have returned their heat to the producer in the form of steam and thus caused a saving of a corresponding amount of live steam. The pan will always remain more or less filled with water and ashes. The ashes will reach into the producer for from 1 to 2 ft. above the water-line; they are removed periodically without interfering with the work of the producer.

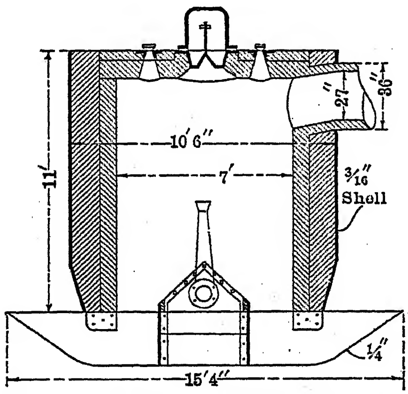


Fig. 187.—Duff water-seal gas producer.

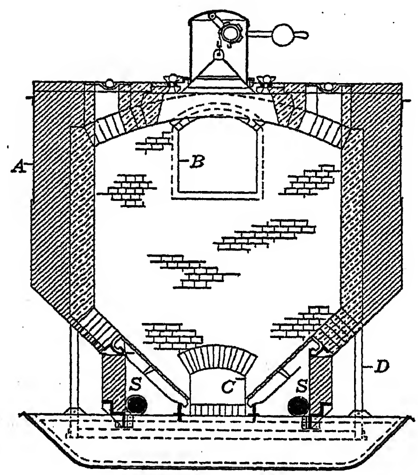


Fig. 188.—Swindell water-seal steam-blast gas producer.

(4) The Swindell Water-seal Steam-blast Producer (Fig. 188).[2]—This is a circular producer with bosh of 45° and bosh-line at one-third the height; the continuation of the bosh walls is formed by two sloping grates; *C* is the cleaning door at the base of the grate. The steam-injector pipes, *S*, extending the entire length of the grate, have rows of openings blowing toward the center; the water-seal pan has the width of the grate and a length equal to that of the diameter of the furnace; the gas-outlet pipe, *B*, is situated at the highest point and has a cleaning door; it can be shut off from the gas main by a sand-damper plate. In the roof there are two coal hoppers, six ball poke-holes and in the plates of

[1] *Cass. Mag.*, 1902, xxii, 504; *Field Mag.*, 1901, v, 550; *Stahl u. Eisen*, 1903, xxiii, 1191; *Foundry*, 1906, xxviii, 17.

[2] *Iron Age*, May 19, 1898, Jan. 29, 1903; 1905, lxxv, 647; 1906, lxxvii, 953.

each hopper six additional ball poke-holes, so that every part of the coal surface can be reached. A producer 12 ft. diam., and 12 ft. high, gasifies 1,500 lb. coal in 24 hr.

(5) THE SMYTHE CONTINUOUS AND SELF-CLEANING GAS PRODUCER (Fig. 189).[1]—This is a circular producer with water-seal bottom, large inclined grate, steam blast, poking and cleaning holes above the water level, and a bell and hopper feed. The capacity is 900 lb. coal per hr.

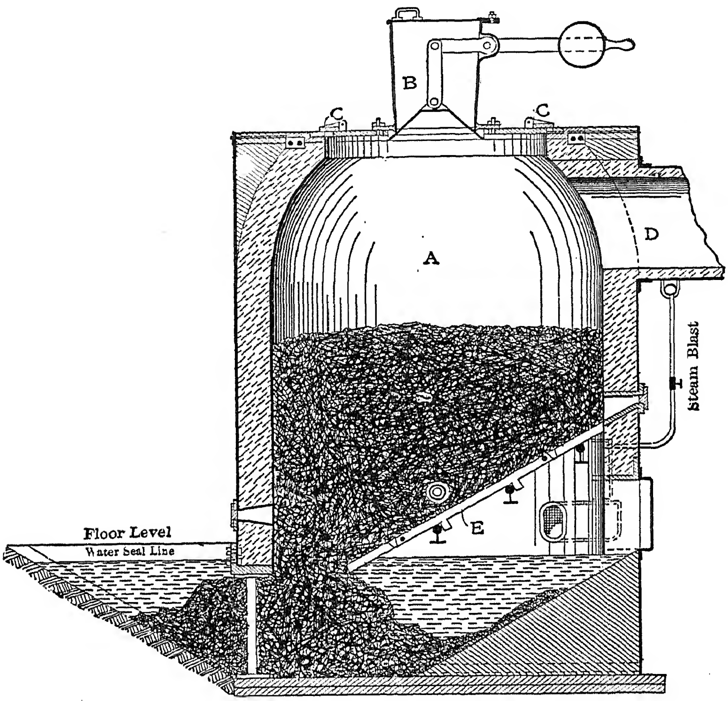


FIG. 189.—Smythe continuous and self-cleaning gas producer.

(6) THE MORGAN GAS PRODUCER WITH BILDT AUTOMATIC FEEDER[2] is represented in Fig. 190. *A* is the receiving hopper for coal, *B* the valve for controlling the admission of coal into receiver *C*, *D* revolving distributing disc rotated by bevel gears *E* and ratchet *F* operated by shaft *G*, *H* water-seal, *I* cast-iron mantle for support of producer, *J* steam-blower, *K* blast-pipe, *L* conical cap, *M* poking holes. A 10-ft. producer gasifies[3] under usual conditions 625 lb. coal per hour or 12.7 lb. per hour per sq. ft. grate; with forced draft, its capacity is increased to 800 lb. The automatic feeder by its uniform charging and

[1] *Cass. Mag.*, 1895–96, IX, 53; *Eng. Min. J.*, 1906, LXXXII, 835; *Iron Trade Rev.*, 1908 XLII, 763.

[2] *Iron Age*, April 30, 1896; *Tr. A. I. M. E.*, 1898, XXVIII, 166; George Feeding Apparatus at Lackawanna Steel Co., *Iron Age*, Dec. 29, 1904.

[3] Williams, *Iron Age*, 1905, LXXVI, 1532.

Quasebart, *Metallurgie*, 1908, V, 224, 251.

even distribution of coal insures greater uniformity in quality and quantity of gas than is usual with hand-feeding.

In the Hughes mechanical gas producer,[1] made by the same firm, the shaft turns slowly on its axis, while a vertical water-cooled cast-steel stirrer is made to move to and fro. The producer 11 ft. 6 in. diam. at top, 7 ft. at bottom of cone, and 12 ft. high, with four poking holes to break off clinkers from the sides, makes 6 r.p.m., requires 3 h.p., gasifies 24 tons of coal in 24 hr.; one man tends three furnaces.

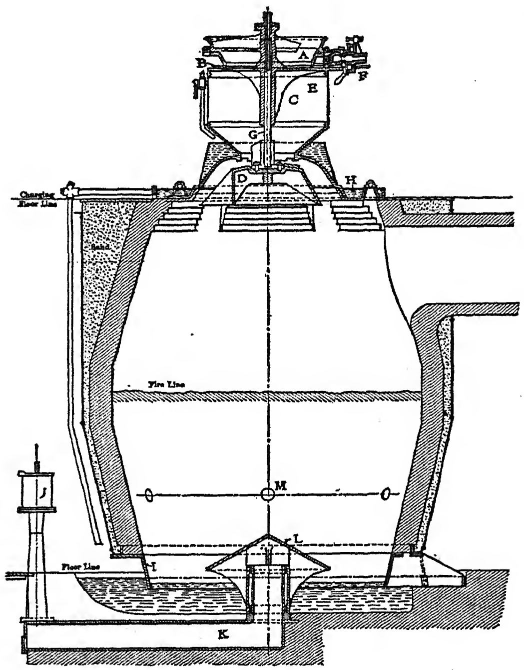


FIG. 190.—Morgan gas producer with Bildt automatic feeder.

(7) THE TAYLOR GAS PRODUCER.—The characteristics of this producer are: a circular solid bottom which, on being revolved, breaks up any clinkers and discharges them with the ashes into the closed ash pit; a deep bed of ashes; a central steam-injector blast carried through the ashes to the fuel. The two leading types marked *A* and *B* are shown in Figs. 191 and 192; both have a bosh. Type *A*, Fig. 191, an iron shell lined with fire-brick, is suited for anthracite and bituminous coal of which the ashes do not clinker readily. In type *B*, Fig. 192, the lower part of the shaft is a water jacket, the upper an iron shell lined with fire-brick; this makes the producer suited for coals the ashes of which clinker readily, as the clinkers will not adhere as easily to the jacket as to the brick wall. With

[1] *Eng. Min. J.*, 1906, LXXXII, 1061.

both types, the part beneath the bottom plate is often hopper shaped (Fig. 193) to receive the ashes to be dropped at intervals into an ash car. The shaft is slightly contracted toward the top. The bottom of the bosh has a smaller diameter than the revolving bottom and does not extend down to it; the space between the two is filled with ashes, which, with an angle of repose of about 55°, glide over the edge of the bottom, when this is revolved. In order to grind up the ashes, four sets of agitating bars or scrapers are arranged above the table. In case the ashes should be discharged faster on one side than the other, with the result that the coal would sink more quickly on the discharging side, the bars can be withdrawn, which will retard the discharge. The ash bed, which is put on the bottom before firing up, is about 3 1/2 ft. deep and is kept about 6 in. above

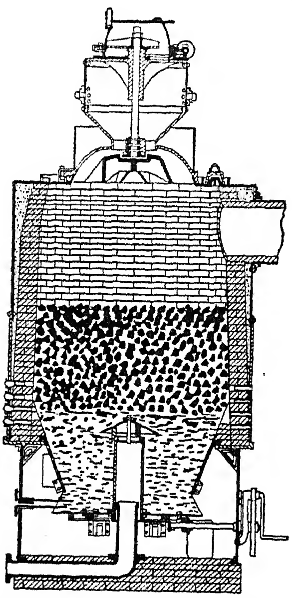


FIG. 191.—Taylor gas producer. Type A.

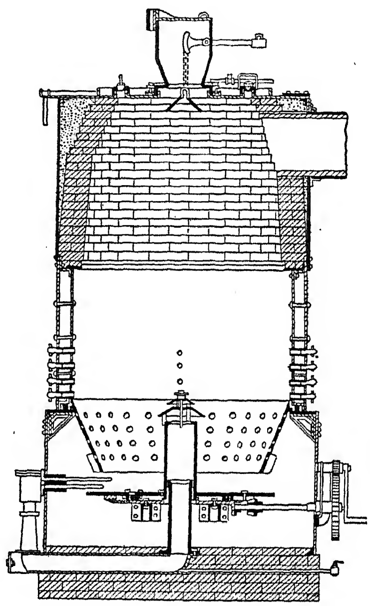


FIG. 192.—Taylor gas producer. Type B.

the cap of the central pipe. The bottom is revolved as soon as this height is exceeded, *i.e.*, every 6 to 24 hr. according to the rate of working. Ashes are taken out once in 24 hr. The blast is furnished by a steam jet. In the side walls are 4 rows of peep holes to watch the process and the position of the top of the ash bed. The producer is made in seven sizes, Nos. 2 to 8; No. 2 is 2 ft. diam., 10 ft. high, and has a grate area of 3.1 sq. ft.; No. 8 is 8 ft. diam., 15 ft. high, with a grate area of 50.3 sq. ft.; No. 8 will gasify 600 lb. anthracite coal (pea) per hour or 12 lb. per square foot of grate, 745 lb. West Virginia or Pennsylvania bituminous coal per hour or 12 lb. per hour per square foot of grate. With bituminous coal, the fuel bed has to be deeper than with anthracite.

(8) THE FRASER-TALBOT PRODUCER.[1]—This producer, Fig. 194, consists of a cylindrical sheet of boiler iron riveted to four I-beam columns *C* and lined with brickwork as shown. To the bottom of the shell is attached an inverted truncated cone of cast-iron *D* holding the ashes; its lower end is sealed by water held in the concrete ash pan *E*. The blast pipe in the center of the ash pan is of cast-iron. Its cone-shaped roof holds the bearing for the water-cooled vertical shaft, carrying two water-cooled stirring arms *I*. The shaft *H* has a slow rotat-

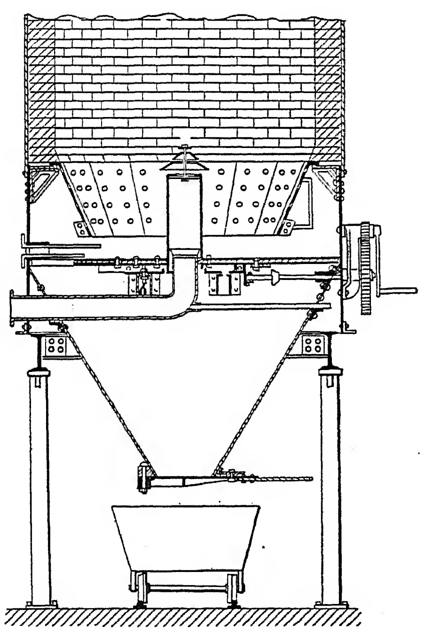


FIG. 193.—Ash-hopper of Taylor gas producer.

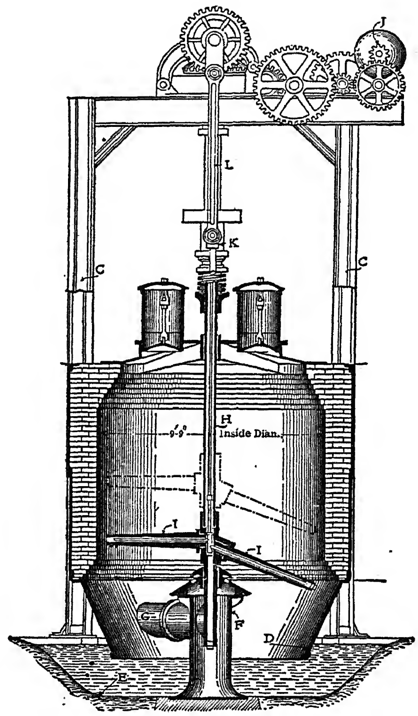


FIG. 194.—Fraser-Talbot gas producer.

ing and a vertical motion, both of which are obtained by gearing carried on a steel platform riveted to the tops of the I-beam columns *C*. The drawing shows two coal hoppers closed by cones. A 10-ft. producer gasifies 750 lb. coal per hour or 16.3 lb. per hr. per sq. ft. grate and requires an engine of 3 to 4 h.p.

(9) THE KITSON GAS PRODUCER (Fig. 195).[2]—It consists of a circular, brick-lined iron shell, with inclined, continuously revolving iron bottom (8) to agitate the coal and discharge the ashes into a water box, and a continuous feed, which delivers the coal below the normal fuel line (35). The inside dimensions of a 7-ft. producer are: diam. at bottom, 7 ft.; at top 4 ft.; height, 12 ft. The iron

[1] *Iron Age*, Feb. 14, 1901; *Stahl u. Eisen*, 1908, XXVIII, 1355.

[2] *Iron Age*, March 14, 1895; April 30, 1896, ill.; Sept. 30, 1897, results; improved form: *Eng. Min. J.*, 1898, LXV, 254.

shell rests on eight cast-iron (3) columns, 3 ft. high, which are bolted to a circular base plate (2) supported by six 10-in. I-beams (14), resting with their ends on two 18-in. foundation walls (1). The inclined (18°) hearth (8) carrying the fuel bed is cast in one piece with the central hollow shaft (12), and is carried by a train of steel balls rolling in a groove upon the base plate. The shaft, carrying a horizontal worm wheel (9), is driven by an endless screw (11). The hearth, which is elliptical and 2 ft. higher at one end than the other, has a grating or perforated plate (8) on the brick surface to facilitate the discharge of the ashes. The space in which the eight supporting columns stand is enclosed by upright cast-iron ash plates (3), which are radially adjustable to permit regulating the discharge of the ashes onto the base-plate, whence they are conveyed by scrapers attached to the underside of the hearth to an opening in the base plate, and dropped into a water-sealed receiver extending sufficiently outside the producer to permit removing the ashes. The air and steam blast (36) enter the producer through the hollow central shaft, and are distributed through a central tuyère and the perforated hearth plates or gratings. The coal, which was formerly charged from the top by a bell and hopper feed (60 lb. every 5 to 6 min.), is now introduced continuously below the fuel line by a screw feed (17 and 18), connecting by belting (19 and 20) with the shaft (11) of the endless screw driving the worm wheel of the shaft. The bottom makes about one revolution in 40 min. with anthracite, one in 20 to 30 min. with bituminous coal. About 2 h.p. is required to drive the bottom. The advantages claimed for the producer are that, by the slow, continuous revolving of the bottom, the fuel is always in motion, with the result that a uniform good quality of gas is obtained, and that no clinkers are made which require labor for barring.

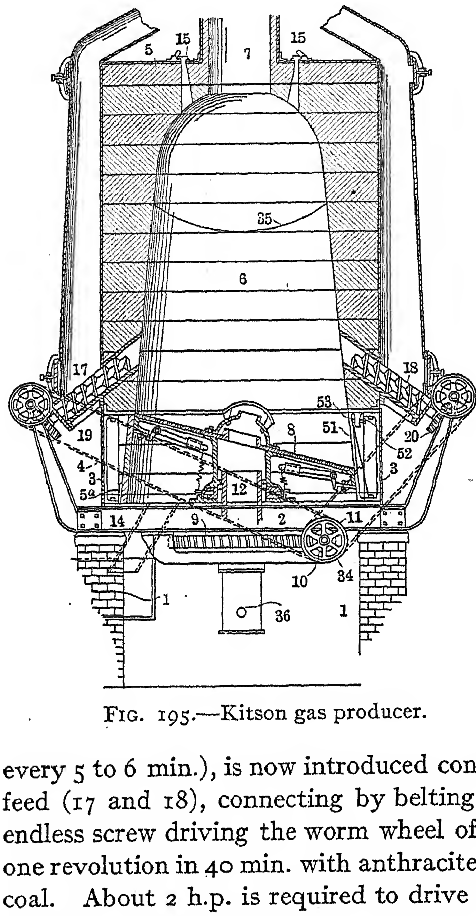


Fig. 195.—Kitson gas producer.

(10) Other Modern Producers.—Other modern producers are described in the literature cited below.[1]

[1] Dickmann, C., and Reynolds, A., "The Basic Open-hearth Steel Process," Constable, London, 1911, pp. 1–136. Forter-Miller, *Iron Age*, June 13, 1904. Amsler, *op. cit.*, 1905, LXXV, 60. Deschamps, *Bull. Soc. Ind. Min.*, 1903, II, 889. Lencauchez, *op. cit.*, 1905, V, 86, Bache, *op. cit.*, 1910, XII, 5, 185. Burger, *Iron Age*, 1905, LXXVI, 1669. Herrick, *op. cit.*, 1907, LXXIX, 964. Miller, *op. cit.*, 1907, LXXX, 14. Canaris, *Stahl u. Eisen*, 1908, XXVIII, 537.

(11) PRODUCERS FOR WOOD, SAWDUST, PEAT, ETC.[1]—Producers for light fuels must be spacious and high to hold a sufficient quantity of fuel and to prevent unconsumed air from passing through them; they must, further, be connected with condensers to free the gases from the large amounts of water vapor generated.

(12) MANAGEMENT.[2]—A new producer has to be dried and warmed, like any other furnace, by a slow fire. In starting, a fire is made, the gases being allowed to pass off through the charging opening; fuel is added gradually, and the thickness of the bed is slowly increased. When well under way, the charging opening is closed and the gas main opened. In order to insure complete expulsion of air, which might cause explosions to take place, the bed is at first kept low so as to obtain gases containing CO_2 and water vapor, which drive out the air. In normal work, the gas in the producer ought always to be above atmospheric pressure, even when the producer is worked by natural draught, so as to prevent air from being sucked in. The regular operations are charging coal, poking it down, and removing ashes and clinkers. In order to ascertain the rate of sinking of the coal, iron rods are periodically inserted through the roof holes. They serve also to poke down the coal in case spongy hollow spaces form, and to break up any clinkers. The ashes and clinkers are removed periodically, the manner and frequency varying greatly with the different producers.

While the leading use of producer gas has been and remains to serve as a heating fuel, recently it has been used directly for the production of energy in a gas engine.

145. Efficiency of Gas Producers.—The thermal efficiency of a coal gas producer, $\frac{\text{cal. power of gas}}{\text{cal. power of coal}} \times 100$, run with a steam-blower, the gas being burnt cold, is about 80 per cent., *i.e.*, the loss in heat by conversion amounts to 20 per

Hofmann, *op. cit.*, 1910, XXX, 993. *Iron Age*, 1910, LXXXVI, 281. Küpper, *Stahl u. Eisen*, 1911, XXXI, 636. *Iron Age*, 1911, LXXXVII, 1534. Hilger, *Stahl u. Eisen*, 1911, XXXI, 100. Forter-Trump, *Iron Age*, 1909, LXXXIII, 646. *Met. Chem. Eng.*, 1910, VIII, 371. Akerlund, *Iron Age*, 1910, LXXXVI, 64. Nordensson, *op. cit.*, 1911, LXXXVIII, 1310. Klepetko, *Tr. A. I. M. E.*, 1904, XXXIV, 296. Gloodz, *Glück Auf*, 1909, XLV, 1826; 1910, XLVI, 1495, 1525. Lucke, *Eng. Mag.*, 1912, XLII, 889. Gradenwitz, *Eng. Min. J.*, 1909, LXXXVIII, 1019. Herrick, *Iron Age*, 1907, LXXIX, 964. Pulverized Fuel: Latta, *Iron Trade Rev.*, 1911, XLVIII, 14. Mechanical Feeding: Smythe, *Iron Age*, 1908, LXXXII, 130. Hermanns, *Glück Auf*, 1911, XLVII, 948. Chapman, *Iron Trade Rev.*, 1912, LI, 878.

[1] Daniels, *Tr. A. I. M. E.*, 1881, IX, 312.
Loomis, *op. cit.*, 1890–91, XIX, 995.
Odelstjerna, *op. cit.*, 1894, XXIV, 293.
Lang, *op. cit.*, 1904, XXXIV, 748.
Loomis, *J. I. and St. I.*, 1890, II, 280.
Douglas, *op. cit.*, 1902, I, 287.
Davis, *Cass. Mag.*, 1909, XXXVI, 344.
Haanel, *Report Utilization of Peat*, Ottawa, 1912.
Snypp, *Eng. Mag.*, 1912, XLIV, 96.

[2] Fernald-Smith-Clement-Grine, "Industrial Problems in Gas-producer Tests," *Bull.* No. 393, U. S. Geol. Surv., Washington, 1909.

cent. of the calorific power of the coal, and the producer delivers the remaining 80 per cent. as available in the combustible gas.

In calculating the heat balance of a producer, this is debited with the heat values that go into it and credited with those that come from it; the difference between the two is the heat lost by radiation. A test carried on by Wendt,[1] in a water-seal gas producer run with a steam-blower forms the basis of the following calculation. In the test, lasting 71 hr., there were burned 14,760 kg. coal of the composition C 58.14, H 3.38, S 0.40, O 9.60, N 0.70, hygr. H_2O 10.05, ash 17.73. There were obtained from 100 kg. coal 14.36 kg. ashes with 1.64 kg. C; the cleaned and dried gas had the. composition CO_2 5.40, CO 27.01, CH_4 2.93, H 14.55, N 50.11; 1 cbm. clean gas carried in g. H_2O 87.0, tar 15.35, dust 0.95 (with C 13.5 per cent.). The other observed data are given in the calculation, and the results are assembled in the balance-sheet.

Before the heat-balance can be obtained, a few preliminary calculations have to be carried out to answer the following questions:

1. What is the amount of gas in cbm. produced from 100 kg. coal? 100 kg. contain 58.14 kg. C, not all of which was gasified, as the ashes, the tar and flue-dust retained some of it. The ashes retained 1.64 kg. C. One cbm. crude gas contained 15.35 g. tar which, averaging 80 per cent. C, gives $0.01535 \times 0.80 = 0.01028$ kg. C; it also contained 0.95 g. dust which, with 13.5 per cent. C, gives $0.0095 \times 0.135 = 0.000128$ kg. C. The analysis of the clean and dry gas shows per cbm.

$$\begin{array}{ll} \text{C in } CO_2, & 0.0540 \times 0.54 \\ \text{C in } CO, & 0.2701 \times 0.54 \\ \text{C in } CH_4, & 0.0293 \times 0.54 \\ \hline & 0.3534 \times 0.54 = 0.1908 \text{ kg. C, hence} \end{array}$$

$$1 \text{ cbm.}: 0.1908 \text{ kg. C} = x : [58.14 - 1.64 - (0.01228 + 0.000128)x]$$

$$x = 279.1 \text{ cbm. clean gas per 100 kg. coal.}$$

2. What is the amount of air in kg. introduced per 100 kg. coal? It has just been found that 100 kg. coal produced 278.1 cbm. gas. According to analysis, it contains 50.11 per cent. or 139.4 cbm. N which weigh $139.4 \times 1.26 = 175.6$ kg. Deducting the 0.7 kg. N of the coal, leaves 174.9 kg. from the blast. Assuming the air to contain 77.2 per cent. N by weight, the desired answer is $174.9 : 0.772 = 226.5$ kg. air.

3. What is the amount of steam in kg. introduced per 100 kg. coal? Tests made showed that the steam-injected air contained 140 g. H_2O per cbm. dry air. As no data are given for the moisture and temperature of the air before coming in contact with steam, let it first be assumed that the air was dry and at 10° C. The vol. of 226.5 kg. dry air at 10° C. $= \frac{\text{weight}}{\text{weight 1 cbm.}} \times \frac{273+10}{273} =$

[1] "Mittheilungen über Forschungsarbeiten aus dem Gebiete des Ingenieurwesens," 1906, Heft 1, p. 57; abstr. *Stahl u. Eisen*, 1906, XXVI, 1184; transl. of abstract with values changed into B.t.u. and deg. F., *Iron Age*, 1906, LXXVIII, 1539;

Ennis, *Tr. Am. Inst. Chem. Eng.*, 1909, II, 115.

$\frac{226.5}{1.293}\times\frac{273+10}{273}=181.6$ cbm.; and at 10° C. the weight of the steam $=181.6\times 0.140=25.4$ kg. Assuming secondly that the air was not dry, but contained 0.007 kg. H_2O per cbm., or 1.3 kg., then the amount of steam introduced was $25.4-1.3=24.1$ kg. The second figure is the one that will be adopted.

4. What is the amount of H_2O in kg. dissociated per 100 kg. coal? The analysis of clean and dry gas shows per cbm.:

Free H, 0.1455×0.09

H as CH_4, $\frac{2\times 0.0293\times 0.09}{0.2041\times 0.09}=0.018369$ kg. H

H in tar $0.065\times 0.01535=0.000998$ kg H. (see composition in part VIII)
Total H in gas (from 100 kg.) $0.019367 \times 278.1=5.3860$ kg.

Deducting from this the H in the coal, 3.38 kg., leaves 2.0060 kg. H which has been supplied by dissociating water. Then $2.006\times 9=18.054$ kg. of water decomposed.

The items in the accompanying balance sheet were obtained as follows:

I. Heat from the Calorific Power of the Fuel.—The calorific power of the raw coal was determined calorimetrically as 5,598 Cal.; this gives $100\times 5{,}598 = 559{,}800$ Cal. for 100 kg. coal.

II. Heat from Air-blast.—The temperature of the atmosphere was 10° C. and that of the air-blast was raised to 73° C. by the steam-injector. This increase will be taken up under the next head; here only the sensible heat of the air at 10° C. will be taken into account. It is the product of weight $\times$ spec. heat $\times$ temp. $=226.5\times 0.2345\times 10=531.1$ Cal.

III. Heat from Steam.—The average steam pressure of the boiler was 3.83 atmospheres absolute, $=41.6$ lb. per sq. in. gauge pressure, and the temperature of the superheated steam 354° C. The amount of heat required to convert 1 kg. liquid H_2O at 0° C. into vapor at 0° C. is 606.5 Cal., and the mean specific heat of this vapor will then be $0.34+0.00015\ t$. The total heat in the steam will then be $24.1[606.5+(0.34+0.00015\times 354)354]=17{,}970.4$ Cal.

IV. Heat in Calorific Power of Gas.—The calorific power of the gas was determined as 1,549 Cal. per cbm. As there were produced 278.1 cbm., the heat in this gas was $1{,}549\times 278.1=430{,}777$ Cal.

V. Heat Absorbed in the Decomposition of Steam by Carbon.—According to $H_2O+C=CO+H_2$, there is a deficit $(-69{,}000+29{,}160=-39{,}840)$ of 39,840 Cal. in decomposing 18 kg. liquid H_2O. If the start is made with steam at 100° C., each kg. contains 637 Cal. of sensible heat, and 18 kg. give

$18\times637=11,466$ Cal., which reduces the above deficit to $39,840-11,466=28,374$ Cal. The amount of heat absorbed in the decomposition of 1 kg. steam by C is therefore 1,576 Cal. The weight of steam dissociated per 100 kg. coal was found above to be 18.054 kg.; hence the heat absorbed per 100 kg. is $1,576\times18.054=28,453$ Cal.

VI. Heat Lost in Sensible Heat of Ashes.—This is found by adding the product of weight, spec. heat and temperature of dry ashes to the product of the weight of water (removed with the ashes from the water-seal) and its temperature. The weight of the ashes formed per 100 kg. coal was shown above to be 14.36 kg.; the spec. heat is about 0.25; the amount of water removed with the ashes was 8.59 kg. and its temperature 45° C. The desired values are $14.36\times0.25\times45=161.5$ and $8.59\times45=386.6$; hence the heat lost 548.1 Cal.

VII. Heat Lost in Unburnt Carbon of Ashes.—The 14.36 kg. ashes were shown above to contain 1.64 kg. C. The amount of heat lost by them is therefore $1.64\times8,100=13,284$ Cal.

VIII. Heat Lost in Tar and Soot.—One cbm. of gas was shown to contain 15.35 g. tar, of which two determinations gave 76.75 and 78.50 per cent. C; and dust 0.95 g. with 13.5 per cent. C. According to Lunge[1] an average analysis of tar free from H_2O is C 78.0, H 6.5, N 1.0, S 0.5, O 14.0. The calorific power of average water-free tar calculated according to the modified Dulong formula for liquid water is 7,950 cal. For every 100 kg. coal there were formed 278.1 cbm. gas, hence the heat lost in the tar, assuming Lunge's average analysis, is $278.1\times0.01535\times7,958=33,972.7$ Cal.

The heat lost in the soot is the product of cbm. gas per 100 kg. coal $\times$ kg. soot per cbm. gas $\times$ per cent. C in soot $\times$ calorific power of C, or $278.1\times0.00095\times0.135\times8,100=288.9$.

The loss in tar and soot is therefore $33,972.7+288.9=34261.6$ Cal.

IX. Sensible Heat in Uncleaned Gas.—The amount of heat present in the gas from 100 kg. coal corresponds to the product of calculated volume $\times$ measured temperature $\times$ mean spec. heat, or

$$278.1\times529\times(0.35686+0.000025\times529)=54,630 \text{ Cal.}$$

X. Heat in Undecomposed Steam.—This item is included in the preceding head which refers to uncleaned gas.

XI. Heat Consumed in Evaporating Water in Coal.—According to the chemical analysis 100 kg. coal contain 10.05 kg. H_2O which multiplied by the latent heat gives the desired amount; $10.05\times606.5=6095.3$ Cal. The heat of the gaseous water is included in heading VIII.

[1] "Chemisch-technische Untersuchungsmethoden," Springer, Berlin, 1905, II, 727.

XII. Heat Consumed in Evaporating Water from Water-seal.—The amount of water evaporated from the seal is the difference between the dissociated + condensed water and the water introduced by the blast and the steam. The water dissociated per 100 kg. of coal was 15.6 kg. The analysis showed that 1 cbm. uncleaned gas contained 87 g. =0.087 kg. H_2O. As 278.1 cbm. gas were formed per 100 kg. coal, the H_2O amounted to 24.2 kg. The total is therefore $15.6+24.2=39.8$ kg. The water introduced by the blast was 24.20 kg., and the coal contained 10.05 per cent. H_2O, hence the sum of these two items is $24.20+10.05=34.25$ kg. Deducting this from the above total leaves the water from the seal, or $39.8-34.25=5.65$ kg.; and this multiplied by the latent of H_2O gives $5.65\times606.5=3426.7$ Cal.

XIII. Heat Lost by Radiation.—This is found by difference and is shown in the table to amount to 4042.8 Cal.

Balance Sheet for 100 Kg. Coal

	Calories	Per cent.
Debit		
I. Heat from calorific power of fuel	559,800.0	97.3
II. Heat from air-blast	531.1	0 1
III. Heat from steam	15,183.4	2.6
	575,514.5	100.0
Credit		
IV. Heat in calorific power of gas	430,770.0	74.85
V. Heat absorbed in the decomposition of steam by carbon.	28,453.0	4 94
VI. Heat lost in sensible heat of ashes	548 1	0 10
VII. Heat lost in unburned carbon of ashes	13,284 0	2.31
VIII. Heat lost in tar and soot	34,261.6	5.95
IX. Sensible heat in uncleaned gas	54,630 0	9.50
X. Heat in undecomposed steam	....	....
XI. Heat consumed in evaporating water from coal.	6,095 3	1.04
XII. Heat consumed in evaporating water from water-seal.	3,426 7	0 60
XIII. Heat lost by radiation	4,042.8	0.71
	575,522.5	100 00

The thermal efficiency of the producer is $\frac{430,777}{559,800}\times100=77$ per cent., presupposing the gas to be burnt cold; if burnt hot, its sensible heat will have to be

added $\frac{432300+54630}{559800} \times 100 = 87$ per cent. In order to complete the statement, the amount of coal burnt under the boiler to raise the necessary steam will have to be considered.

146. Mond Gas.[1]—Mond gas differs from ordinary producer gas in that it is rich in CO_2 and H, and poor in CO and N, as seen by the following analyses:

TABLE 136.—ANALYSES OF MOND GAS

Locality	CO_2	O	CO	C_mH_{2n}	CH_4	H	N	Authority
Detroit, Mich	14 6	0 5	10 3	. ...	5.3	23.5	55.8	Pennock
England.........	16 5	. ...	11.0	....	2.0	27 5	43.0	Pennock

It is formed by running a producer with an excess of steam (2.5 lb. steam at 250° C. : 3 lb. air per lb. coal), two-thirds of which remains undecomposed and passes off with the gases. The resulting low temperature of the producer causes the formation of much CO_2 and the conversion of 70 per cent. of the N of the coal (with 1 to 1.5 per cent. N) into NH_3, one ton coal yielding 90 lb. $(NH_4)_2SO_4$. The gases from the producer (500° C.) travel through U-shaped recuperator pipes, around which passes a mixture of air and steam on its way to the jacket and grate of the producer; they are then drawn through a wrought-iron chamber filled with a fine water-spray by means of mechanical dashers. The watery atmosphere removes dust and some tarry matter and collects some NH_3, but the gas is not washed sufficiently for any NH_3-liquor to collect. The gases, cooled to about 90° C., now rise in an acid tower filled with checkerwork of acid-proof brick over which 4-per cent. H_2SO_4 trickles downward and combines with the NH_3. The acid is circulated until it contains about 36 per cent. $(NH_4)_2SO_4$, when part of it is drawn off, replaced by fresh acid, and worked up.

The gas freed from NH_3 rises through a cooling tower *A*, filled loosely with wood packing over which water of 40° C. trickles downward reducing the temperature of the gas to 55° C.; this is then delivered to the furnace where it is to be burnt. The cooling water warmed to 75° C. is pumped to the top of the heating and saturating tower *B*, filled in the same way as *A*, and allowed to trickle through it. Air from a blower is forced in near the bottom of tower *B*. In its upward passage it becomes warmed and charged with water vapor and then passes through the U-shaped pipes and air-jacket of the producer that it may become superheated before it comes in contact with the glowing coal, or coke.

[1] Mond, *J. Soc. Chem. Ind.*, 1889, VIII, 503.
Darby, *J. I. and St. I.*, 1896, I, 144.
Editor, *Eng. Min. J.*, 1896, LXI, 421.
Humphrey, *Engl. Inst. Civ. Eng.*, 1896–97, CXXIX, 190; *Eng. Min. J.*, 1897, LXIV, 640; *Iron Age*, Nov. 18, 1897; *Coll. Guard.*, 1897, LXXIV, 750, 795, 844.
Pennock, *J. Soc. Chem. Ind.*, 1905, XXIV, 599.
Heber, *Braunkohle*, 1910, VIII, 759.

The water collecting at the bottom of tower *B* has been cooled down to 40° C., and is pumped on the top of tower *A*.

Mond gas is of uniform quality and clean; its calorific power may be higher than that of ordinary producer gas on account of its high percentage of H. The efficiency of the producer is about the same as that obtaining with the ordinary manner of making producer gas, for the reason that the heat evolved in the formation of CO_2 is mostly absorbed by the decomposition of H_2O and the generation of H. In order to warrant the increased cost of the Mond plant over that of the ordinary producer plant, it is necessary to have a capacity of gasifying at least 30 tons of coal per day.

147. Iron Blast-furnace Gas.—The iron blast-furnace may be considered as a large gas-producer. Its gas shows the following range of composition: N 55 to 65 per cent. vol., CO 20 to 32, CO_2 6 to 18, H 1 to 6, CH_4 0 to 6; the average is N 60, CO 24, CO_2 12, H 2, CH_4 2 per cent. The gas contains more CO_2 and less H than normal producer gas, and is free from S; it is charged with more or less moisture and dust. It used to be burned exclusively in heating stoves and boilers. At present part of it is being more and more used for the production of energy in the gas-engine after it has been thoroughly cleaned (§340). It has been suggested that ordinary producer gas be enriched by running the producer with a mixture of air and iron blast-furnace gas on account of the low percentage of H of the latter (part of the blast-furnace coke having been burnt by the O of the ore) and the endothermic reduction of CO_2.

148. Water Gas.[1]—The first water-gas plant was erected by T. S. C. Lowe in 1874 at Phoenixville, Pa., who was awarded a medal by the Franklin Institute in 1886 for his work as a pioneer. Water gas is obtained by the action of steam upon incandescent C, viz., $H_2O+C=H_2+CO-39{,}840$ cal. Its theoretical composition by vol. is therefore: 50 H + 50 CO (or by weight 6.67 H + 93.33 CO) with a calorific value of $H_2+CO+O_2=H_2O+CO_2+126{,}100$ cal. As the reaction of making water gas is strongly endothermic, the necessary heat must be supplied in some other way, and this is done by working the generator intermittently as an ordinary producer ($C+O=CO+29{,}160$ cal.).

In the manufacture there are therefore two distinct periods alternating in from 5- to 10-min. intervals: the period of "heating-up" (1,200° C.) by introducing air and making producer gas, and the "period of cooling" (steaming) by introducing steam and making water gas. The steaming period may not reduce the temperature to much below 1,000° C., as then the reaction $2H_2O+C=2H_2+CO_2$ would prevail over $H_2O+C=H_2+CO$ (p. 296).

[1] Strache, W., "Das Wassergas, Seine Herstellung und Vermendbarkeit," Deutike, Vienna, 1896.

Geitel, M., "Das Wassergas und Seine Verwendung in der Technik," 3d ed., Siemens Berlin, 1900.

Blass, *Stahl u. Eisen*, 1886, VI, 3.

Lunge, *Min. Ind.*, 1900, IX, 149.

Dicke, *Stahl u. Eisen*, 1907, XXVII, 1181, 1223.

Thibeau, *Rev. Univ. Min.*, 1908, XXI, 58, 100.

Chantraine, *ibid.*, p. 76, 115.

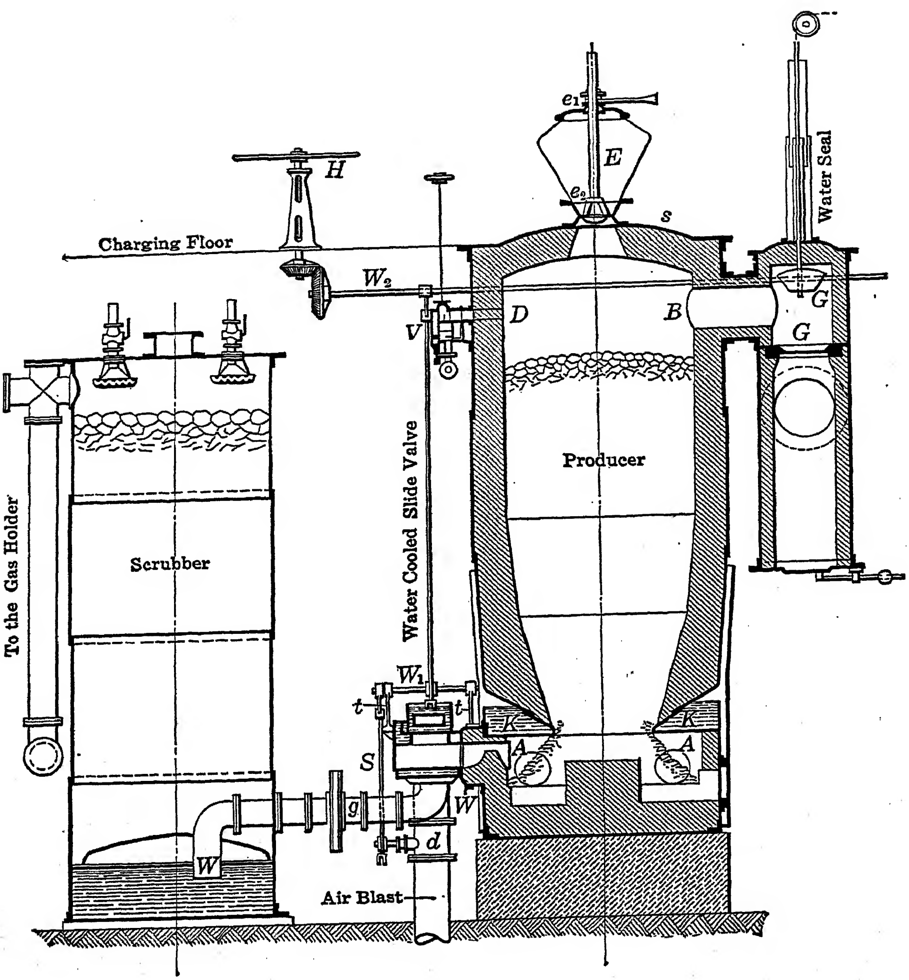


FIG. 196.

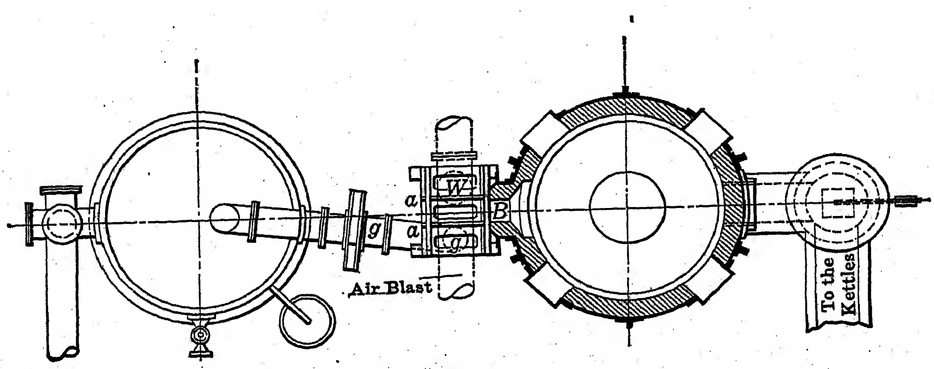


FIG. 197.

FIGS. 196 and 197.—Water gas producer, German type.

The fuel used is anthracite and carbonized fuel: coke (charcoal). Industrial water gas shows the following range of composition: H 44 to 53 per cent. vol., CO 40 to 45, CH_4 0 to 4, CO_2 1.5 to 6, N 1 to 8; an average has been given in Table 127.

The gas is used mainly as an illuminant after enrichment with heavy hydrocarbons to give it the required candle power. As a heating gas it serves mainly in small fires for welding, brazing, forging, tempering, crucible-fusions, etc., where a fuel of high calorific value is desired. In large fires, such as open-hearth furnaces,[1] boilers,[2] and in the production of energy in gas-engines, it has been or is being replaced by mixed gas generated in producers run with steam blowers, as water gas is expensive on account of the steam required, the heat lost in heating up, and the intermittent character of the process of manufacture.

The plant,[3] Figs. 196–197, in its simplest form consists of a generator and a scrubber. The generator is a cylindrical furnace of boiler iron (about 20 ft. high, 8 ft. outer and 5 ft. inner diam.) lined with fire-brick. Its top is closed by the charging hopper *E* with cover e_1 and valve e_2. The bottom has a cooling ring *k* which prevents the adhering of clinkers and thus facilitates their removal. In "blowing-up," the low-pressure air-blast (15.5 in. water) passes by the damper *d*, enters the annular chamber *A*, traverses the ash-bed and the column of anthracite or coke (about 8 ft. high), and is converted into producer gas which passes through flue *B* and valve *G* into the delivery pipe. After thus blowing hot for 10 to 11 min., the changes necessary for steaming are made by simply turning the hand-wheel *H*, which is connected by gearing, rods and levers with the different parts to shut off the air with the throttle valve *d* and to turn on steam at *D* with slide valve *V*, to shut off the producer-gas valve *G*, and to reverse the water-cooled slide-valve *S* having explosion ports *a*. This allows the water gas now generated by the downward passage of the steam through the incandescent carbon to pass off through *g* and *W* into the scrubber, be cleansed by ascending through the trickling wash-water, and then leave through the discharge pipe into the gas holder. After steaming 4 min., the hand-wheel *H* is turned in the opposite direction, and the furnace operated again with air. While charging fuel, both air and steam valves must be kept closed. The pressures in the generator and scrubber must be carefully regulated. Thus, in "blowing-up" the pressure at the bottom of the generator is 15.7 in. water, at the top 7.0 in., and in the scrubber 4.7 in.; in steaming, it is at the top of the generator 27.5 in., at the bottom 15.7 in., and in the scrubber 9.8 in. Fischer[4] examining a water-gas plant at Essen, Germany, found that one net ton of coke gasified in 1.6 hr., gave 36,200 cu. ft of water gas; the gas analysis gave the following values:

[1] Lilienberg, *Tr. A. I. M. E.*, 1884–85, XIII, 708.

[2] Jacobus, *op. cit.*, 1888–89, XIX, 300.

[3] *Min. Ind.*, 1900, IX, 153.

[4] "Die chemische Technologie der Brennstoffe," Vieweg, Brunswick, 1901, vol. II, p. 226.

	Blow-up gas			Water gas			
	After 1 min.	After 6 min.	After 10 min.	After 1 min.	After 2 min.	After 3 min.	Average
CO_2	7.04	4.03	1.60	1.8	3.0	5.6	3.3
CO	23.68	28.44	32.21	45.2	44.5	40.9	44.0
CH_4	0.44	0.39	0.18	1.1	0.4	0.2	0.4
H_2	2.95	2.20	2.11	44.8	48.9	51.4	48.6
N_2	65.89	64.94	63.90	7.1	3.1	1.9	3.7

Of the calorific value of 7,000 cal. of the coke, 3,000 (43 per cent.) were recovered in water gas, 2,970 (42 per cent.) went into producer gas, 800 into cooling water and 290 were added by the use of steam, leaving 520 lost by radiation. As far as the water gas is concerned, the heat generated in blowing up is lost unless the producer gas formed is burnt and the heat of the products of

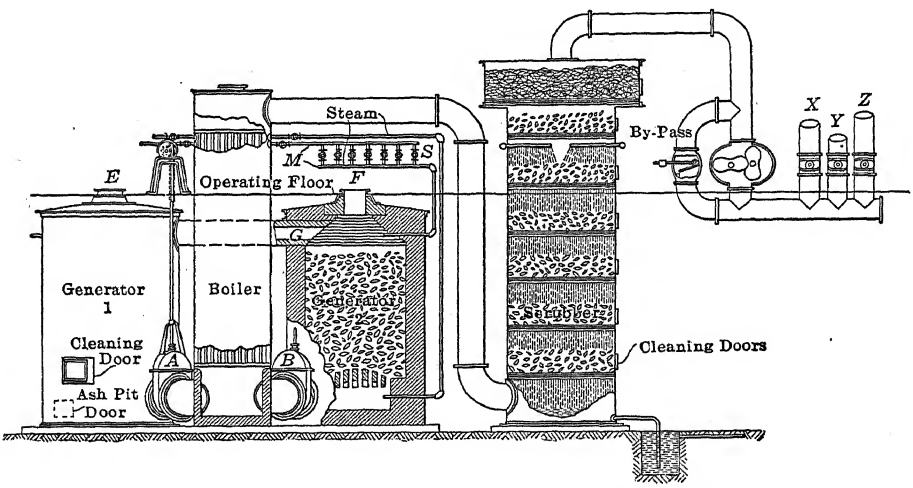


FIG. 198.—Loomis-Pettibone gas-producer at Nacozari, Mexico.

combustion are returned to the generator, by conducting them, *e.g.*, through recuperators or regenerators through which the steam is conducted before it enters the generator.[1] With large plants, the producer gas of several furnaces is led into a holder and burnt under boilers.

Another form of generator is the down-draft type of Loomis-Pettibone[2] which is suited for natural fuels, rich in volatile matter, such as wood and bituminous coal.

[1] *J. I and St. I.*, 1890, II, 281 (Loomis); 1902, I, 287 (Douglas).
[2] *Tr. A. I. M. E.*, 1890–91, XIX, 995 (Loomis); 1904, XXXIV, 748 (Langton).

Figs. 198 and 199 represent the plant of the Moctezuma Copper Co., Nacozari, Mexico.[1] A unit plant consists of two generators (inside diam. 6 ft. 9 in., fuel bed 8 ft. deep) connected with one another by a horizontal pipe *G* below the charging doors *E* and *F*; a boiler (economizer) joined to the generators below the arched fire-brick grates by means of pipes with gate-valves *A* and *B*; and a scrubber attached to the positive-blower plant which provides the necessary draft and delivers the gases to two holders or into the open through pipes *X* (producer gas), *Y* (water gas), and *Z* (waste gas).

In starting the producers, a layer of fuel about 5 ft. deep is charged and ignited at the top. The blower is set going, the fuel brought to incandescence, and replenished through openings *E* and *F* until the normal depth of 8 ft. has been attained. All the air entering at the top comes in contact with fresh fuel; the products of distillation and imperfect combustion are drawn downward through incandescent fuel and completely gasified. They leave the bottoms of the generators through valves *A* and *B*, rise through the tubular boiler and gen-

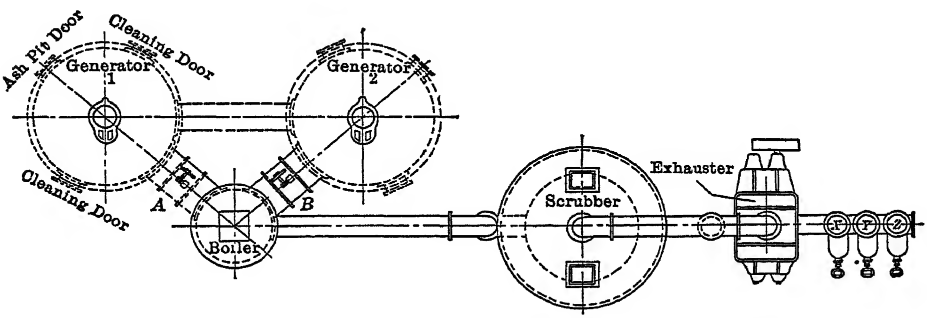


FIG. 199.—Loomis-Pettibone gas-producer at Nacozari, Mexico.

erating steam leave it at the ordinary temperature of chimney gases; they are then further cooled to about atmospheric temperature and cleansed by traveling through the water-sprayed coke-filled trays of the scrubber, and finally pass off into the open through pipe *Z*. After about 10 min., the gas will be rich enough to be diverted through pipe *X* into the holder for producer gas. In making water gas, doors *E* and *F* and valves *B*, *X* and *Z* are closed; steam is turned on under grate of generator No. 2. The water gas formed passes through the top-connecting pipe into generator No. 1, leaves it through valve *A*, passes through boiler, scrubber, exhaust and pipe *Y* into the water gas holder. In the next steaming, the direction of the gases is reversed; valve *A* is closed and *B* opened, and steam admitted below the grate of generator No. 1. Making of producer and water gas alternate about every 5 min., the time depending upon the quality of gas it is desired to obtain.

At Nacozari the usual mode of operating has been modified, by introducing on top of the charge just enough steam through valves *M* and pipe *S* during the "heating up" period to have the temperature in the generator rise very

[1] *Tr. A. I. M. E.*, 1904, xxxiv, 755, 756.

gradually, then making a short water gas run with bottom steam to slightly cool the fire, and resuming to make mixed (Dowson) producer gas with top-steam. The water gas collected in a holder is fed gradually into the main holder for mixed gas which supplies the engines.

A radical departure from the ordinary method of making water gas is that of the Dellwick-Fleischer System[1] which aims to burn the C during the heating up period to CO_2 instead of to CO, and accomplishes it by having a low fuel bed (3 to 4 ft.) and a high-pressure blast. In thus saving fuel (8,000 *vs.* 4,000 cu. ft. water gas per ton of coke) and time (20 *vs.* 80 per cent. of the time required for blowing hot), the new process becomes much more efficient than the older system. The following calculation by Dellwick[2] brings this out clearly. The thermal data used by the author are slightly different from those given by Dellwick.

Old Method and Dellwick Method Compared

	Old method	Dellwick method
One pound C requires for combustion........	To CO: $\frac{16}{12}$ lb. O	To CO_2: $\frac{32}{12}$ lb. O
This O is accompanied by..................	4.32 lb. N	8.64 lb. N
The products of combustion carry away at 700° C.	1,136 cal.	2,092 cal.
The heat of combustion of 1 lb. C is	2,400 cal.	8,080 cal.
Balance available for heating fuel	$2,400-1,136=1,264$ cal.	$8,080-2,092=5,988$ cal.
To fill the balance of 28,760 cal. required for making 30 lb. water gas there must be burnt.	$\frac{28,760}{1,264}=22.75$ lb. C	$\frac{28,760}{5,988}=4.83$ lb. C
Apart from loss by radiation, etc., there are required for making 30 lb. (753 cu. ft.) of water gas.	$12+22.75=34.75$ lb. C	$12+4.83=16.83$ lb. C
Or: per 1 lb. C is produced.................	21.7 cu. ft. water gas	44.7 cu. ft. water gas
As water gas of theoretical composition contains 167 cal. per cu. ft., there are utilized in water gas from 1 lb. C.	3,627 cal.	7,465 cal.
This is equal to............................	48 per cent. of the total heat of C	92.5 per cent of the total heat of C

In the old method, the quantity of CO formed is amply sufficient to raise the necessary steam; in the Dellwick method the escaping heat suffices only to warm the boiler water. The fuel consumed (12 to 15 per cent.) for raising steam reduces the actual efficiency from 92.5 to about 80 per cent.

[1] Dellwick, *Iron Age*, April 8 and July 21, 1898; *J. I. and St. I.*, 1900, I, 119.
Lunge, *Min. Ind.*, 1900, IX, 156.
Sexton, *Eng. Mag.*, 1903–04, XXVI, 204.
Sjöstedt, *J. Can. Min. Inst.*, 1905, VIII, 290.

[2] *J. I. and St. I.*, 1900, I, 123.

Fig. 200 gives a diagrammatic representation[1] of the plant. The generator *B*, about 8 ft. diam. and 10 to 12 ft. high, has a bed of coke 3 to 4 ft. deep. In heating up, blast is admitted through pipe *E*, the products of combustion pass off through valve *A* either into the open through chimney *D* or first through some recuperator to be traversed by steam. After heating up for 2 to 3 min., the coke has reached a temperature of about 1,500° C. and is ready for the change to steaming which is made by turning the ratchet-wheel *K*. This cuts off the blast at *G*, closes the top-valve *A*, opens the gas-valve *J*, and admits steam through *L* or *L'* alternately from top and bottom. Practice has shown that alternation gives a more uniform product. Steaming lasts 8 to 10 min. (a test-flame shows

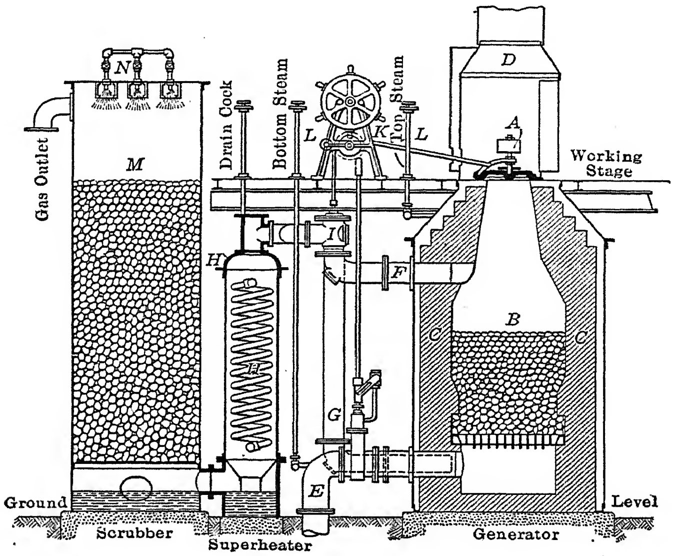


FIG. 200.—Diagrammatic representation of Dellwik-Fleischer water-gas plant.

the character of the gas), when a reversal of the ratchet-wheel *K* will set the valves again for heating up. The coke consumed is replenished through *A*, and ashes and clinkers are removed through doors (not shown) above and below the grate bars. The water gas, drawn off above or below the charge, passes through valve *J*, either directly into the scrubber *M* or first through the steam-superheater *H*, into the gas-holder. Its average composition is given as H 51, CO 42, CH_4 0.5, CO_2 4.0, N 2.5.

At the water-gas plant of the Lake Superior Power Co.,[2] Sault Ste. Marie, Ont., the cost of water gas, running two generators continuously, was 15 cents per 1,000 cu. ft. with 70,000 cu. ft. gas per ton of coke: 11.5 tons coke (@ $6) $69; steam $20; four men (@ $2.50) $10; superintendence, power, light depreciation and repairs $21; total $120; or 800,000 cu. ft. @ 15 cents = $120. Running intermittently for a couple of hours at a time to furnish the gas for two

[1] *Eng. Mag.*, 1903–04, XXVI, 210.

[2] Sjöstedt, *J. Can Min. Inst.*, 1905, VIII, 290.

holders of 20,000 cu. ft. capacity, the gas to be used in the chemical laboratory for welding, etc., the cost was 50 cents per 1,000 cu. ft.

149. Oil Gas.[1]—This gas is formed by the rapid destructive distillation (cracking) of petroleum (tar, shale oil) in retorts at elevated temperatures with or without the presence of superheated steam. The composition of the gas varies a good deal with the temperature of the retort.

TABLE 137.—COMPOSITION OF OIL GAS

Temperature	CO_2	O	C_mH_n	CH_4	H	CO	N	Authority
Correct.......	1.27	1.89	36.02	41.30	18.89	0.63	0	Lewes, *J. Gasbeleucht-*
Too high.....	1.33	1.33	5.32	23 22	68.47	1.83	0	*ung*, 1891, XXXIV, 668.

At a correct temperature the gas is rich in illuminants, C_mH_n, and poor in H; if the temperature is too high, the cracking is carried too far, the C_xH_y are decomposed into H and tar, and the gas loses its illuminating power. According to Noyes-Blinks-Mory[2] the undiluted oil gas of Terre Haute, Ind., produced at a temperature of 700 to 900° C., had the following composition: CO_2 2.3, O 0.02, C_mH_n 28.1, CH_4 44.8, H 20.5, CO 0.6, N 3.5 per cent. vol. Steam was used to inject the oil into the generator, but was not decomposed. The gas requires a large amount of air for combustion; in burning, it has a tendency to smoke.

In the Archer gas fuel process,[3] crude oil brought to a temperature of about 200° C. is pumped through a 0.5-in. coil to be further superheated, and then brought in contact in a closed vessel (the vaporizer) with steam of similar temperature. The office of the steam is to prevent the breaking up of hydrocarbon vapors with deposition of C. The gasification of the oil is not complete, as the gas always retains some oil vapor. One gallon of oil is said to yield from 80 to 150 cu. ft. of gas having a calorific value of 135 Cal. per cu. ft. The gas has been used in heating-furnaces (Cleveland Rolling Mills) and the Pernolet open-hearth steel furnace (Cambria Steel Co.), but has fallen into disuse on account of the low efficiency of the process, as only about 40 per cent. of the heating power of the oil is found in the gas.

In the Pintsch process furnishing the Pintsch gas used for lighting railroad cars,[4] the oil is gasified by dropping it slowly into the upper of a pair of ⋂-shaped horizontal retorts, placed one above the other and heated from a fireplace beneath the lower. Vapor and gas from the upper retort, which vaporizes

[1] Producers: Willard, *Iron Trade Rev.*, 1909, XLIV, 327; International *Amet Gas Power Co.*, *Iron Age*, 1911, LXXXVIII, 100; Gwodz, *Braunkohle*, 1912, XI, 357.

[2] *J. Am. Chem. Soc.*, 1894, XVI, 689.

[3] *Eng. Min. J.*, 1890, XLIX, 251; *Jernk. Annal.*, 1897, plate XII, Fig. 1.

[4] Drawings of Manhattan Elevated Railroad Plant, *Am. Engineer. and R. R. J.*, 1899, LXXIII, 1.

and cracks the oil, pass into the lower retort kept at 1,000° C. which transforms the remaining vapors into permanent gas.

150. Apparatus for Burning Fuels, General.—Of the four classes of metallurgical operations, termed dry, wet, electric and mechanical, the first practically always requires an elevated temperature produced in furnaces by the combustion of fuel. The fuel is either in contact with the ore, in which case the furnace is its own burning apparatus, or it is burnt in a separate fireplace, in which case the flame either acts directly upon the ore or indirectly by heating a closed vessel containing the ore. The constructions of the separate fireplaces will differ for solid, liquid and gaseous fuels, but whatever the forms, they ought to permit as much as possible:

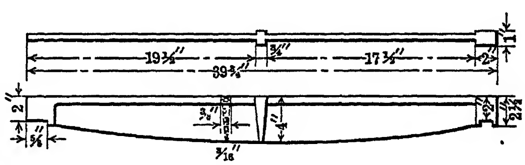
(1) Perfect combustion with the smallest excess of air over that which is theoretically necessary.

(2) Raising of the temperature well above the normal when occasion requires and at the same time giving the flame space for free development.

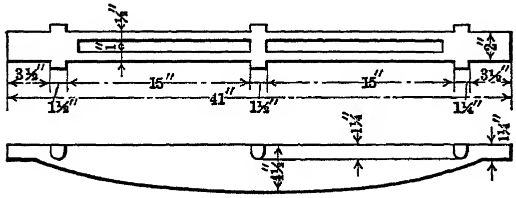
(3) Small loss of heat by radiation.

All separate fireplaces have to be supplied with the air necessary for combustion (natural and forced draft) and with the means of removing the products of combustion.

151. Apparatus for Burning Solid Fuels.—Most fire chambers for solid fuels have a grate, a set of iron bars placed in such a manner as to leave air-spaces between them. The grate, Figs. 201 to 204, supporting the fuel, separates the fireplace from the ash-pit. The air enters beneath the grate, passes through the open spaces, strikes the clinkers and ashes, then travels through a bed of partly burnt fuel to the raw fuel. On its way it becomes heated to the kindling temperature of the fuel (225° C. for peat, 300 for wood, 400-500 for coal, 600 for anthracite and coke) and ignites it. The necessary air is supplied either by natural (chimney) draft or by forced draft (undergrate blast); the supply with the latter is more regular than with the former, as it is independent of the condition of the fuel-bed. In low-temperature (roasting) furnaces natural draft[1] is more common and in high-temperature (smelting) furnaces forced draft.[2] With both methods the draft has to be the stronger



FIGS. 201 and 202.—Single grate-bar.



FIGS. 203 and 204.—Double grate-bar.

[1] Cobb, Natural and Induced Draft, *J. Soc. Chem. Ind.*, 1905, XXIV, 528; *Eng. Min. J.*, 1905, LXXX, 676.

[2] Booth, Natural and Forced Draft, *Cass. Mag.*, 1901–02, XXI, 130.

the higher the fuel-bed and the greater the tendency of the coal to coke or to decrepitate; lump coal will require less draft than when medium size or fine.

The ashes of the coal as well as some of the clinkers will be collected in the ash-pit. This is open with natural draft, closed with forced draft. The combustion varies somewhat with the character of the fuel, the kind of grate and the manner of firing. The flame from the fireplace, Fig. 205, passes over a fire-bridge onto the hearth of the furnace where it may have to supply only heat, or at the same time to produce an oxidizing or a partially reducing effect; the fuel-bed should be lower with an oxidizing than with a reducing flame.[1]

THE ORDINARY GRATE, Figs. 201 and 202, consists of a set of parallel iron bars about 3 ft. from the floor, placed horizontally or slightly inclined (pitch 0.75 in. : 1 ft.) and supported at the ends by cast-iron bearers. Grates for low-temperature furnaces are usually of cast-iron and for high-temperature furnaces of wrought iron. The air-space between two cast-iron bars is fixed by the width of the heads which is greater than that of the bars. In order to prevent buckling, grates have projections at the center. For the same reason grates are often cast in pairs, Figs. 203 and 204. In Figs. 201 and 202 the heads differ from one another; they represent the front half of a double grate, Figs. 203 and 204; the head is intended to enclose a cross-bar, allowing the head to glide to and fro as demanded by expansion and contraction; the distance between the heads is maintained by a forked casting.

The length of a grate rarely exceeds 3 or 4 ft.; with longer fireplaces there will be two or more sets of grate-bars resting upon transverse bearers, expansion spaces of about 0.25 in. being left between the sets. The width (web) is about 2 in. at the ends and 3 to 5 in. at the middle. The cross-section is tapered downward to assist the ascending of the air and the descending of the ashes. The thickness at the top with natural draft or lump coal is 5/8 to 1 in.; it is often maintained constant for a short distance downward and then tapers quickly to 5/16 to 5/8 in. at the bottom; with forced draft or fine coal, the thickness at the top is 3/8 to 9/16 in. The air-spaces for natural draft or lump coal are 3/8 to 1/2 in. wide, for forced draft or fine coal 1/16 to 1/4 in. The area of the air-spaces is 30 to 50 per cent. of the total grate area. Coal that has ashes which clinker readily requires wide air-spaces.

Wrought-iron bars are usually square, the sides being 1 to 1.5 in. long. They are ordinarily worked with a deep layer of fuel and a thick bed of clinkers (clinker-grate). With low-temperature furnaces wrought-iron grates are sometimes made of flat bars bolted together at the ends and in the middle, with washers (3/16 in.) intervening to furnish the air-spaces and to prevent warping.

Fireplaces usually are oblong; the bars are placed parallel to the sides; occasionally with long fireplaces they are laid parallel to the ends and cleaned from the side, the cleaning slot being closed by a hinged door.

Grates up to 4 ft. long are fired from one end, and when longer they are fired from both ends. The essential requirements of good firing[2] are that the neces-

[1] Tilden, Burning Wood, *Eng. Min. J.*, 1900, LXXXVII, 499.

[2] Cary, *Iron Age*, Oct. 2, 9, 16, 23, 30, Nov. 13 and 27, 1902.

sary amount of air shall pass at an even speed through the grate spaces; that air and gases shall be thoroughly mixed; that the temperature in the fireplace shall be uniform and sufficiently high to ignite the gases, and that the fuel shall be given sufficient time to burn. An ordinary grate may be run with a deep (16 in.) or a shallow (6 in.) fuel bed, depending upon the character of the fuel and the kind of flame that is to be produced.

COMMON METHOD OF FURNACE FIRING.—If a fire has burnt long enough to require replenishing, the grate should be covered with a layer of $6 \pm$ in. of glowing coke. If, according to the common method, coal is spread over the coke, the coal will choke the air-spaces and the evaporation of H_2O and partial distillation of V.H-C. will reduce the temperature to below 560° C., the ignition temperature of the gases; the combined effect is a large volume of smoke and soot due to imperfect combustion. When the distillation is finished, the coke will burn normally as long as its depth is commensurate with the air-supply. If it is too deep, the C will burn to CO as in a gas producer; if too shallow, the combustion will be satisfactory, but too much unconsumed air passes through the grate-spaces, cools the fire and increases unnecessarily the volume of waste gases which, as it is, carry away a considerable portion of the heat generated. In the common method of firing, it is therefore important to feed little coal at a time and this at regular intervals; to spread it evenly over the surface (the fine over the coarse and *vice versa*); to loosen up caking coal with the slice-bar; to break up and remove clinkers; and lastly with natural draft, to keep the fire-door open as little as possible. The inrush of cold air checks the draft and cools the fire. The evil of imperfect combustion may be in part corrected by having air-ports in the fire-bridge or the roof or both. Often fireplaces are purposely made and run with a heavy fuel-bed, resembling a small gas-producer, when the volatile matter is burnt by air superheated in flues passing through the walls of the fireplace.

COKING METHOD OF FURNACE FIRING.—The coking method of firing, Fig. 205, frequently used with high-temperature furnaces, corrects many of the evils of the common method. The fireplace, instead of being closed by a door, has a sheet-iron hopper filled with coal. The rear half of the grate will be covered with half-burnt coal or coke through which an excess of air passes, while the volatile matter of the coal on the front half is being distilled off and, coming in contact with the superheated air, burnt perfectly; at the same time the coal in the hopper is being warmed. After a time, the front coal freed from part of its volatile matter is pushed back and leveled over the rear part of the grate, and the front filled again with coal. As the operation takes some time, it is advisable to check the draft while it is being carried out. The method is not well adapted to coal with a high percentage of readily-fusible ash, unless a slot at the side, usually closed by a hinged door, is provided to remove the clinkers.

Whatever may be the method of hand firing, the changes in the fuel-bed and the corresponding variations in the volume of air make it impossible to obtain perfect combustion. The best results are those in which the chimney gases are

free from CO, H and C_xH_y, contain 4 to 8 per cent. O and over 13 per cent. CO_2.[1]

Feeding fresh coal underneath the glowing coke, down-draft grates,[2] shaking and dumping grates, mechanical stokers,[3] etc., common with boilers and causing according to recent experiments[4] a saving of 7 per cent. of fuel over hand firing, are little used in metallurgical fires, as with these the temperatures often have to be varied from low at the beginning of an operation to high and then again to low toward the end. Rocking grates to assist in breaking up clinkers are found here and there.

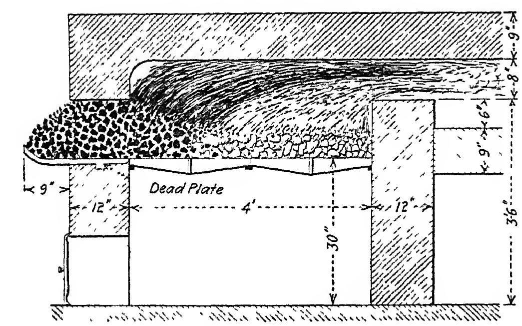


FIG. 205.—Coking method of firing.

The amount of coal that can be burnt per sq. ft. of total grate area depends upon the character of the fuel, the area of air-spaces, and the strength of the draft. A square foot of grate will burn per hour. with a natural draft of 2 in. (water) depression 12 lb. of coal charged to a depth of 4 to 5 in.; with forced draft of 5.5 in. in (water) pressure and a bed 5 to 6 in. thick it will burn to 30 lb., and with higher pressures 40 lb. Table 138 gives some data of Grüner[5] in regard to the amounts of coal burnt per sq. ft. of grate area per hr. in fire-places of typical reverberatory furnaces.

TABLE 138.—COAL BURNT PER SQUARE FOOT OF GRATE AREA IN REVERBERATORY FURNACES

Hand reverberatory roasting furnace	3 to 8 lb.
Agglomerating or lead reverberatory smelting furnace	12 to 16 lb.
Copper reverberatory smelting furnace	16 to 30 lb.
Puddling furnace	20 to 30 lb.
Heating (welding) furnace	30 to 40 lb.
Locomotive boilers with induced draft	80 to 100 lb.

The subjoined table No. 139 by Beckert[6] gives the total grate area and the ratio to area of air-spaces ordinarily found in furnaces for burning 100 lb. of solid fuel with natural draft.

[1] CO_2 Recorders: Haslam, *S. Wales Inst. Eng.*, 1910, XXVI, 1225; Uehling, *Met. Chem. Eng.*, 1911, IX, 329, 656; 1912, X, 497; Sarco, *Iron Age*, 1910, LXXXVI, 1440.

[2] Hawley Down-draft Grate, *Industrial World*, 1907, XLI, 1.

[3] *Cass. Mag.*, 1897–98, XIII, 311; *Eng. Mag.*, 1902, XXIII, 528, 717; *Proc. Eng. Soc. West. Pa.*, 1903–04, XIX, 169.

[4] Report, *Proc. Thirty-seventh Ann. Conv. R'y. Master Mechanics Assoc.*, 1904, p. 367; *Min. Mag.*, 1904, X, 145; *Eng. Min. J.*, 1905, LXXX, 913.

Randall, D. T., and Weeks, H. W., *Bull.* 373, U. S. Geol. Surv., Washington, 1909.

[5] Traité de Métallurgie Générale, I, 257.

[6] "Leitfaden zur Eisenhüttenkunde," Springer, Berlin, 1898, p. 104.

TABLE 139.—RATIOS OF AREAS OF TOTAL GRATE TO AIR SPACE

	Total grate area, sq. ft.	Ratio, grate area: area of air spaces
Coke	6.5 to 9.8	3 : 1 to 2 : 1
Bituminous coal	3.6 to 6.8	3.3 : 1 to 2 : 1
Brown coal	1.2 to 4.9	5 : 1 to 3 : 1
Peat or wood	3.6 to 6.1	7 : 1 to 5 : 1

THE STEP-GRATE.—This grate, Fig. 206, consists of a number of horizontal cast-iron plates placed one above the other in the form of a step at an angle of 40 to 50° with the horizontal. Below the bottom step is an ordinary horizontal grate. The plates rest on castings or special forms of hard fire-brick built into the side-walls of the fireplace. Step-grates are used with bituminous coal having no coking power (sandy coals) and brown coal, both of which fall to

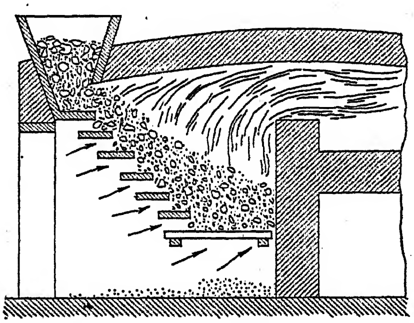


FIG. 206.—Step-grate.

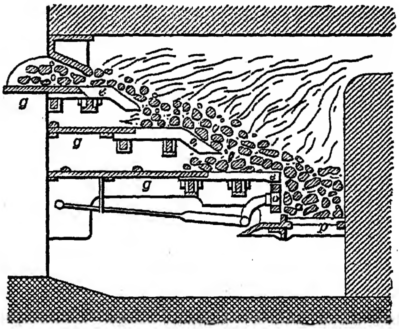


FIG. 207.—Langen underfed step-grate.

pieces when heated. They are usually run with natural draft; with very fine coal forced draft has been found to be necessary. Roasting furnaces requiring an even heat often have step-grates. In operating the grate, the fuel is charged through a hopper at the top which being kept filled prevents cold air from entering the fireplace. The volatile matter is driven off at the upper grates and burnt more or less by the excess air passing through the lower grates covered with half-burnt coal. The coal is gradually worked down the grate by the fireman pushing from the back, and completely burnt on the horizontal grate. Mechanical step-grates,[1] such as the Meissner, Roney, Brightman, etc., commonly used with boilers, are occasionally found with metallurgical furnaces.

THE LANGEN UNDERFED STEP-GRATE.[2]—The Langen grate, Fig. 207, differs from the ordinary form in that the grate-bars *e* are angles forming the continuation of cast-iron plates *g*; *p* is a horizontal grate and *o* a cleaning door.

[1] *Cass. Mag.*, 1897–98, XIII, 311; 1909, XXXVII, 517; *Iron Age*, 1908, LXXXII, 172, 639, 771.

[2] *Dingler*, *Pol. J.*, 1860, CLVIII, 241; *Ann. Min.*, 1862, II, 411; other forms: Georgius, *Braunkohle*, 1910, VIII, 805.

The coal is fed on to the plates *g*, becomes warmed and is gradually pushed forward into the fire. The liberated volatile matter passing off through half-burnt coal is heated to above its kindling temperature and burnt by the excess air passing through the air-spaces. The half-burnt coal gradually glides or falls down the inclined plane on to the horizontal grate to be completely burnt, and its place taken by coal freed more or less from volatile matter. Stoking is begun at the bottom. The grate has not made as much headway as one would be led to expect from the principles embodied in its construction.

WETHERILL GRATE.—The Wetherill grate, Fig. 208, is made up of a number of perforated cast-iron plates; it was devised for burning fine anthracite culm with a forced draft. The plates are of varying lengths and widths (*e.g.*, 6 ft. long ×6 in. wide ×1 in. thick), and have tapering conical holes, 1 in. at the top and 1.25 in. at the bottom. The manner in which the grate works is shown in the illustration. The grate is used in the manufacture of zinc white from ore, in the Bartlett process for smelting zinc-lead sulphides. The European modification is called the Kudlicz grate.[1]

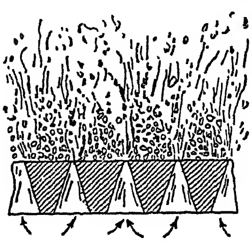


FIG. 208.—Wetherill grate.

BURNING PULVERIZED FUEL.—The apparatus used for burning pulverized fuel or fuel dust was discussed in §107.

152. Thermal Efficiency of a Fire.—The thermal efficiency of a fire, independent of the work the heat may have to perform, is found by bringing into relation the calorific power of the fuel with the losses in heat due to C remaining in the ashes, to imperfect combustion (CO, CH_4, H, soot), and to the quantity of heat carried off by the gases through the chimney.

The following calculation[2] is based upon data given by Gill.[3] They are: composition of coal: H_2O 1.5, S 1.2, C 83.0, H 2.5, ash 11.4, O+N 0.4; its calorific power, 7,220 cal.; temperature of air 25° C.; saturation with moisture, 50 per cent.; composition of chimney gases, CO_2 11.5, O 7.4, CO 0.9, SO_2 n.d., N (by diff.) 80.2, their temperature 275° C.; ashes 14.4 per cent. with 20.83 per cent. C.

To find the volume of gases produced per kg. coal: the 144 g. ashes from 1 kg. coal contain $144 \times 0.2083 = 30$ g. C, hence only 800 g. C were burnt instead of 830. The analysis shows that 1 cbm. gas contains 0.115 cbm. CO_2 and 0.009 cbm. CO, each of which contains cold 0.54 kg. C per cbm., or 1 cbm. contains $0.124 \times 0.54 = 0.06696$ kg. C; hence 0.8 kg. C form 11.95 cbm. gas. Of this, 11.5 per cent. (1.375 cbm.) is CO_2, 7.4 per cent. (0.888 cbm.) O, 0.9 per cent. (0.108 cbm.) CO and 80.2 per cent. (9.584 cbm.) N. The water vapor in the gas per kg. of coal comes from 0.015 kg. moisture in coal (: 0.81) = 0.018 cbm., from

[1] *Oest. Zt. Berg. Hüttenw.*, 1900, XLVIII, 411; *Coll. Guard.*, 1900, LXXX, 334.

[2] See also Richards, "The Thermal Efficiency of a Rotary Cement Kiln," *J. Am. Chem. Soc.*, 1904, XXVI, 80.

[3] "Gas and Fuel Analysis for Engineers," Wiley, New York, 1913, 28.

TABLE 140.—TENSION AND WEIGHT OF WATER VAPOR IN 1 CBM SATURATED AT 5 TO 30° C.

Temp. ° C.	Tension mm.	Weight g.	Temp. ° C.	Tension mm.	Weight g.	Temp. ° C.	Tension mm.	Weight g.
5	6.5	6.8	14	11.9	12.0	23	20.9	20.4
6	7.0	7.3	15	12.7	12.8	24	22.2	21.5
7	7.5	7.7	16	13.5	13.6	25	23.6	22.9
8	8.0	8.1	17	14.4	14.5	26	25.0	24.2
9	8.5	8.8	18	15.4	15.1	27	26.5	25.6
10	9.1	9.4	19	16.3	16.2	28	28.1	27.0
11	9.8	10.0	20	17.4	17.2	29	29.8	28.6
12	10.4	10.6	21	18.5	18.2	30	31.5	29.2
13	11.1	11.3	22	19.7	19.3			

0.025 kg. $H \times 9 = 0.225$ kg. H_2O (: 0.81) = 0.278 cbm. and from air 0.465 cbm. or 0.761 cbm. in all. Table 140 shows that one cbm. air at 25° C. saturated contains 22.9 g. water vapor, hence 12 cbm. 50 per cent. saturated contain $12 \times 22.9 \times 0.50 = 0.377$ kg. (: 0.81) = 0.465 cbm. The heat carried off by the furnace gases is the sum of the products of each gas × specific heat × rise of temperature (275 − 25 = 250° C.).

Gas	Cbm.	Mean specific heat 25—275° C.	Rise in temp. ° C.	Cal.
CO_2	1.38	0.37 + 0.00022t	250	153
O	0.888	0.303 + 0.000027t	250	69
CO	0.108	0.303 + 0.000027t	250	8
N	9.624	0.303 + 0.000027t	250	745
H_2O vapor	0.761	0.34 + 0.00015t	250	95
				1,070

The percentage of heat carried off by the furnace-gases is 14.8.

Lunge[1] has devised the following formula for calculating the loss of heat which does not require the ultimate analysis of the coal:

$$1.854(t'-t)c + 1.854\left(\frac{100-k}{k}\right)(t'-t)c'$$

In which $t'-t$ represents the rise of temperature, c the mean specific heat of 1 cbm. CO_2, k the percentage of CO_2, and c' the mean specific heat of 1 cbm. N. Applying it to the example calculated gives a loss of heat =

$$1.854 \times 250 \times (0.37 + 0.00027t') + 1.854 \times \frac{100-11.5}{11.5} \times 250 \times (0.303 + 0.000027t') =$$

1,312 Cal. or $1{,}312 \times 100 : 8{,}100 = 16.2$ per cent.

[1] *Z. angew. Chem.*, 1889, p. 240.

The results obtained are 2 to 2 1/2 per cent. higher than those calculated from the ultimate analysis of the coal.

Noyes[1] gives the following formula:

$$\left(0.0111+\frac{100-\text{per cent. } CO_2}{\text{per cent. } CO_2}\times 0.00605\right)\times(t'-t)=14.4 \text{ per cent. in the above}$$

example.

153. **Apparatus for Burning Liquid Fuel.**[2]—The conditions[3] to be fulfilled for the economic combustion of liquid fuel are, that the oil shall be finely divided (vaporized or atomized) and intimately mixed at the proper place with the necessary air, and that the combustion chamber shall be at an elevated temperature. When working correctly, the flame is short, clear white, and no smoke issues from the stack; with imperfect combustion, the flame is long and dull red, and black smoke pours from the stack. Special provisions have to be made if a long flame is desired. Low-temperature furnaces require a special combustion chamber, maintained at the required high temperature for perfect combustion of the oil, from which flame and hot gases pass off into the furnace to be heated. In high-temperature furnaces, the furnace itself forms the combustion chamber.

The appliances[4] may be divided into three classes: (1) Those in which the oil is vaporized; (2) those in which it is sprayed into the furnace by purely mechanical means, and (3) those in which the spray is produced by means of compressed air or by steam or by both (atomizers). With all appliances the storage, pumping and piping of the oil have to be considered, and with the last two classes, also the heating and filtering.[5]

The Nobel Vaporizer.[6]—This was the first apparatus devised (1885) for burning liquid fuel. It is shown in Figs. 209 and 210 attached to a crucible furnace. The so-called grate consists of a number of superposed cast-iron pans. The oil (residuum, masut) is fed from an overhead tank into the top

[1] *Am. Chem. J.*, 1897, XIX, 162.

[2] Hodgets, E. A. B., "Liquid Fuel for Mechanical and Industrial Purposes," Spone, London, 1890.

Lew, I., "Die Feuerungen mit Flüssigen Brennmaterialien," Cotta, Stuttgart, 1890. "Production and Use of Petroleum in California," *Bull.* No. 32, State Mineralogist, Sacramento, Cal., 1904. "Oil Fields of the Texas-Louisiana Gulf Coastal Plain," *Bull.* No. 212, U. S. Geol. Survey, Washington, 1900, p. 162. *Report*, U. S. Naval Liquid Fuel Board, Washington, 1904.

Booth, W. H., "Liquid Fuel and its Apparatus," Constable & Co., London, 1912.

Ulrich, *Iron Trade Rev.*, 1909, XLIV, 1038.

Georgius, *Braunkohle*, 1910, IX, 613.

Butler, *Eng. Rev.*, 1911–12, XXV, 168, 243, 325.

Schiel, *Stahl u. Eisen*, 1908, XXXVIII, 1215.

[3] Best, *Eng. and Min. J.*, 1904, LXXVII, 771.

[4] Orde, *Cass. Mag.*, 1901, XX, 61; *Eng. Mag.*, 1901, XX, 779.

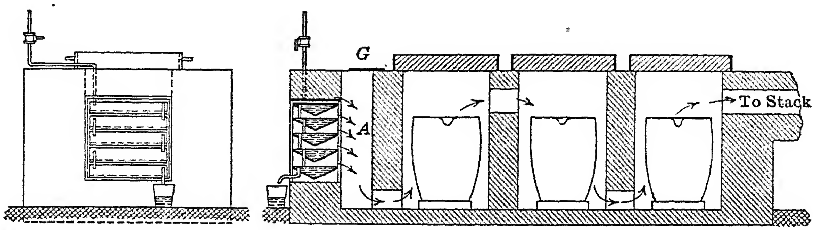
Winkel, *Berg. Hüttenm. Z.*, 1903, LXII, 301, 337, 361, 373, 385.

Peterrson, *Glück Auf*, 1905, XLI, 1405, 1443.

[5] Burt Oil Filter, *Iron Age*, 1909, LXXXIV, 767.

[6] *Stahl u. Eisen*, 1900, XX, 425; *Thon. Ind. Z.*, 1900, XXIV, 1163; 1906, XXX, 1949; *Cass. Mag.*, 1903–04, XXV, 71; Milwaukee type, *Foundry*, 1912, XL, 339; Oil Furnace & Engineering Co., *Eng. Min. J.*, 1912, XCIV, 122.

pan; an overflow pipe in each pan carries the excess oil to the next lower pan, and from the lowest the excess flows into a lower receiving tank, the contents of which are pumped into the upper distributing tank. In starting, the pans are filled with oil; a fire is made in chamber *A*; the vaporized oil is carried into



FIGS. 209 and 210.—Nobel vaporizer.

the chamber by air drawn in between the pans and burnt completely by additional air entering through a port at the top regulated by the damper *G*. The hot gases zig-zag through the three chambers, each of which holds a single crucible or a pair of crucibles; sometimes two to three single-crucible chambers are combined to a block for the better utilization of the heat. The temperature in the first crucible chamber reaches 1,800° C. One sq. ft. of oil surface will vaporize in a crucible furnace about 24 lb. of oil per hr.; in a boiler 1 lb. oil will evaporate 14.5 lb. water. The vaporizer was used at Worcester, Mass., for making Mitis castings, and in New Jersey for smelting nickel. It is in operation in Europe in ceramic plants.

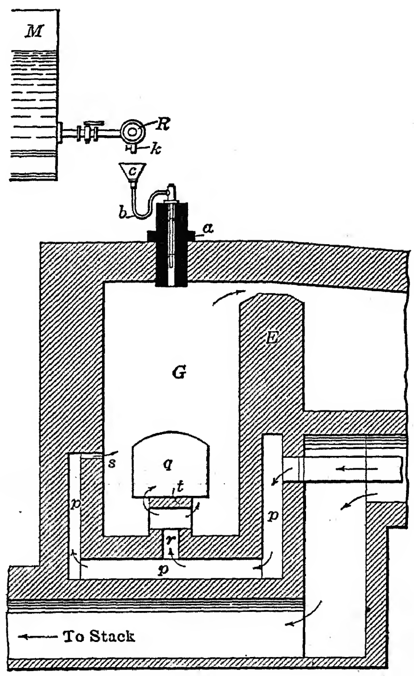


FIG. 211.—Krusell vaporizer.

THE KRUSELL VAPORIZER,[1] Fig. 211, represents another method of vaporizing and burning oil. This is stored in a tank *M*, passes into the distributing pipe *R* and through a number of feed pipes *k*, into funnels *c* connected by U-tubes with 0.5 in. dripping pipes *a*. The oil dripping into the generator *G* (1,200° C.) strikes the hot fire-clay tile *t*, is vaporized and burnt by air which, heated by the waste gases, arrives through flue *p* and enters the generator through ports *r* and *s*; the flame passes over the fire-bridge, *E*, built near the roof, and then on to the hearth of the furnace. The door *q* serves to start the furnace and to re-

[1] *Oest. Zt. Berg. Hüttenw.*, 1901, XLIX, 568; *Eng. and Min. J.*, 1901, LXXI, 435; *Glück auf*, 1905, XLI, 1443.

move any coke that may adhere to the tile *t*. In starting, kindling is ignited on tile *t* and oil allowed to drop on to the fire; in 1 to 1.5 hr. the furnace will have reached its normal temperature. The height of drop for the oil is about 3.3 ft., and the capacity of the generator 0.31 cu. ft. per lb. of oil to be gasified per hr. In a reverberatory smelting furnace, 1 lb. of oil has been found to do the work of 3 lb. of coal.

REGENERATORS AS VAPORIZERS.[1]—Vaporization methods of burning oil have been used in connection with furnaces having regenerative chambers, and perhaps still are in some instances, but they suffer from the deposition of oil-coke upon the checker-work the calorific value of which is lost when it is burnt out by the waste gases passing through the chambers. An improvement on the ordinary method is to vaporize the oil in a separate furnace; to pump the vapor into an accumulator, from which it is delivered under a pressure of 75 to 100 lb. into a Körting mechanical sprayer (see below) inserted into the side of a regenerator near the bottom; a small admixture of air burns off any C deposited on the checker-work. By this method the consumption of residuum oil in a 10 to 15 ton open-hearth furnace was found to be 18 to 20 per cent. on the weight of the steel produced.

MECHANICAL SPRAYING OF OIL.—The first mechanical burner (forsunka) is probably that of Körting, shown in Fig. 212, in which oil, heated in a tubular

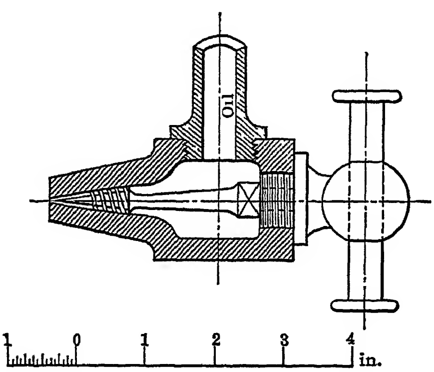


FIG. 212.—Original Körting mechanical burner.

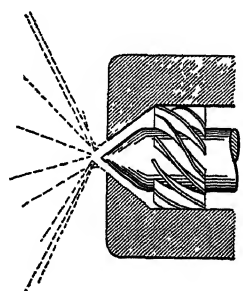


FIG. 213.—Spray from Körting mechanical burner.

boiler to 90° C. and purified by filtering, is pumped under a pressure of 70 to 120 lb. through a tapering screw-thread to the spraying orifice ranging from 0.04 to 0.12 in. diam. The spray formed by the oil under the combined effect of pressure and the centrifugal action of the screw guide-blades is shown in Fig. 213. Mechanical sprayers require a large excess of air for perfect combustion. They have the advantages of simplicity and noiselessness, and are used in ships where the loss of fresh water or the space occupied in the engine-room by the

[1] *Stahl u. Eisen*, XVI, 915; 1900, XXIV, 424; *Berg. Hüttenm. Z.*, 1903, LXII, 374; *Glück Auf*, 1901, XXXVII, 1018; 1905, XLI, 445.

compressor makes atomizing expensive. One pound of oil has been found to evaporate 12 to 16 lb. water at 100° C. and do the work of 7 to 10 lb. of coal.

Atomizing Oil.—An atomizer should break the oil into the finest drops with the least consumption of steam or air; it should be easy to instal and operate, which includes regulation, examination, cleaning and exchange of parts. Both steam and air are used for atomizing; the choice depends much upon the available air-pressure and the form and size of furnace to be heated. With low pressures (2 to 10 lb.), air is more economical[1] than steam; with high pressures (> 10 lb.) steam is more economical; for the same effect, the pressure of air has to be higher than that of steam. Steam reduces the temperature of the flame near the burner,[2] as decomposition absorbs heat, but the combustible gases burn further on in the furnace and thus produce a long flame.[3] Air requires pre-heating of oil or of air or of both to make the oil mobile; with steam this is not necessary, but the steam must be dry. In either case the temperature of the oil should not exceed 80 to 100° C. as otherwise parts begin to be volatilized. A combination of steam and air is being used, but the results have so far not been decidedly favorable. With low-temperature furnaces steam is more common than air; with high-temperature furnaces air is usually given the preference; with regenerative furnaces, the nozzles of the burners are frequently water-cooled.

Atomizing burners have been classed[4] under five heads as follows:

(1) Drooling Oil Burners, Fig. 214.—The oil oozes, drools out at the orifice over and on to a flaring jet of steam which expands within the layer or film

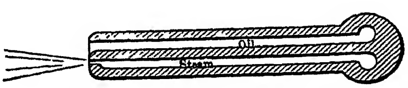


Fig. 214.—Drooling oil-burner.

of oil as this is being carried into the combustion chamber. Although apparently crude, the method is giving satisfactory results. The Booth oil burner, Fig. 215, represents this type. It is used extensively on the Atchison, Topeka and Santa Fé railway system. A similar burner is in operation in Southern Russia.[5]

(2) Atomizer Burners Proper, Fig. 216.—The orifice of the oil-delivery pipe is placed at an angle to the steam- or air-pipe, the stream of which impinges upon the oil and sprays it into a fine mist. This class of burners is the most

[1] *Compressed Air*, 1904–05, IX, 3315.

[2] Campbell, H. H., "The Manufacture and Properties of Iron and Steel," New York, 1903, p. 246.

[3] Examples of Oil-furnaces: Crane, *Eng. and Min. J.*, 1893, LVI, 644; Brent, *J. Can. Min. Inst.*, 1902, V, 288; Bell, *Cass. Mag.*, 1903, XXV, 70; Ropp, *Min. Sc. Press*, 1902, LXXXV, 308; *Eng. and Min. J.*, 1903, LXXV, 81.

[4] "U. S. Liquid Fuel Board," 1904, p. 328.

[5] *Stahl u. Eisen*, 1900, XX, 428.

common. The Best oil burner,[1] Fig. 217, corresponds closely to the diagrammatic sketch, except that the atomizer opening is above the oil-supply passage. The suction-effect of the atomizer is supposed to prevent the collection of solid

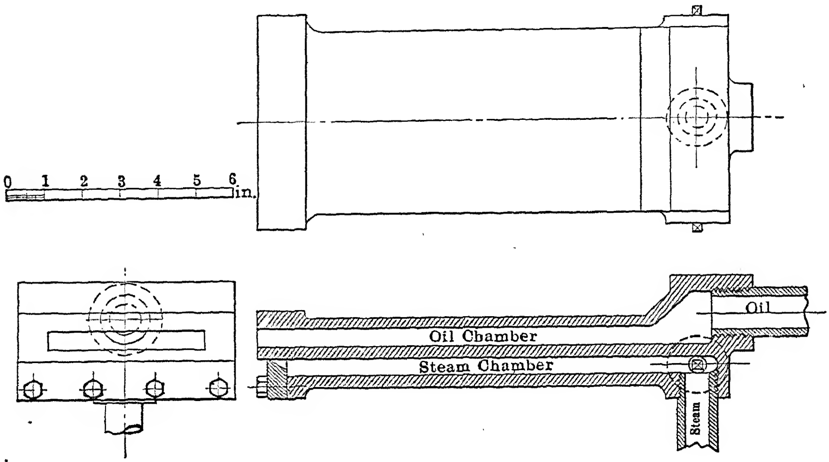


FIG. 215.—Booth oil-burner.

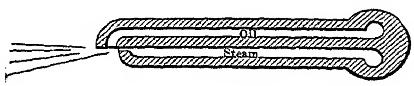


FIG. 216.—Atomizer oil-burner proper.

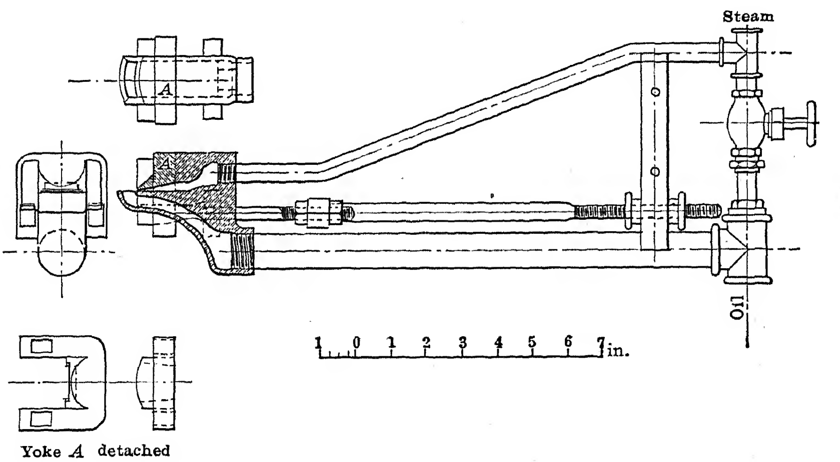


FIG. 217.—Best oil-burner

matter in the oil orifice. The figure shows the burner to be of the slot design.

THE PIPE OIL BURNER, Fig. 218, represents a very simple and common form. The size of the nozzle of the outer pipe is of leading importance; it is usually

[1] *Eng. and Min. J.*, 1904, LXXI, 771; *Iron Age*, 1908, LXXXII, 104.

made 3/16 to 1/4 in. in diam. at first and then reamed out from time to time until the best results are obtained.

THE WILLIAMS BURNER, Fig. 219, is constructed for steam as an atomizer

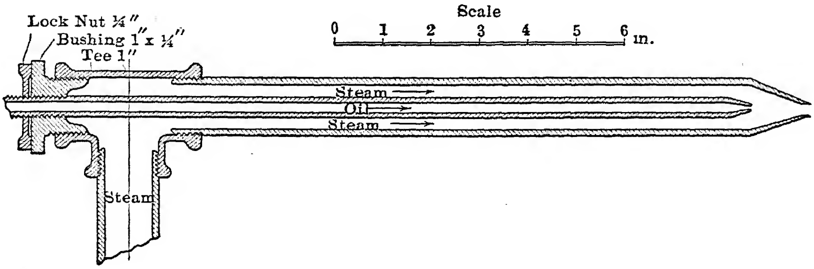


FIG. 218.—Pipe oil-burner.

while heated air is drawn in and mixed with the atomized oil. The oil enters vertically through *b* and is under control of cock *c*, regulated by handle *d*; steam

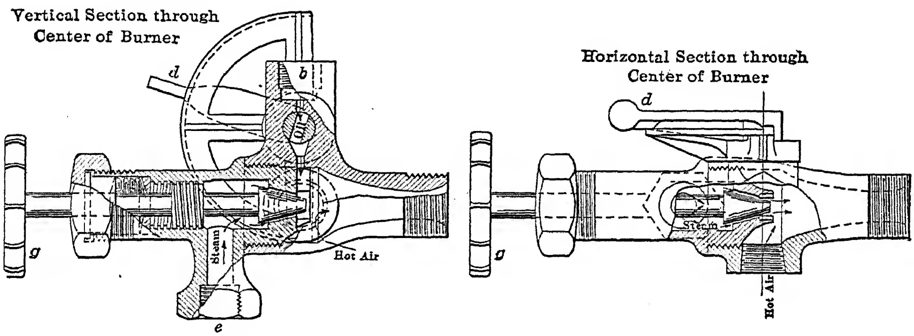


FIG. 219.—Williams oil-burner.

enters at *e*, flows through the conical opening *f*, regulated by hand-wheel *g*, and atomizes the oil as it flows between the vertical flanges *h*; air heated by the waste gases is drawn in through the opening *i*.

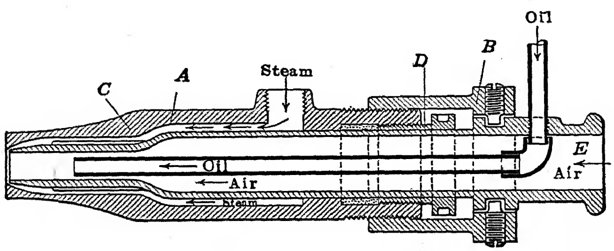


FIG. 220.—Reed oil-burner.

THE REED BURNER Fig. 220, is also intended for the combined use of steam and air. The oil-pipe is 3/8 in. diam.; the size of the opening through which the atomizing steam is to flow from the chamber *A* is regulated by turning the

sleeve *B* and thus moving the casting *C*; a limited amount of air is admitted through *E*.

Tar burners[1] are usually atomizers.

(3) Chamber Oil Burners, Fig. 221.—Oil and steam (air) are more or less mingled inside the body of the burner; the mixture passing out from the nozzle

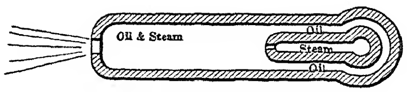


Fig. 221.—Chamber oil-burner.

is rapidly broken into small particles by the sudden expansion of the steam (air). The Hayes oil burner, Fig. 222, represents a simple form of this type. Another example is the Billow[2] burner used at Maurer, N. J., for heating retorts in

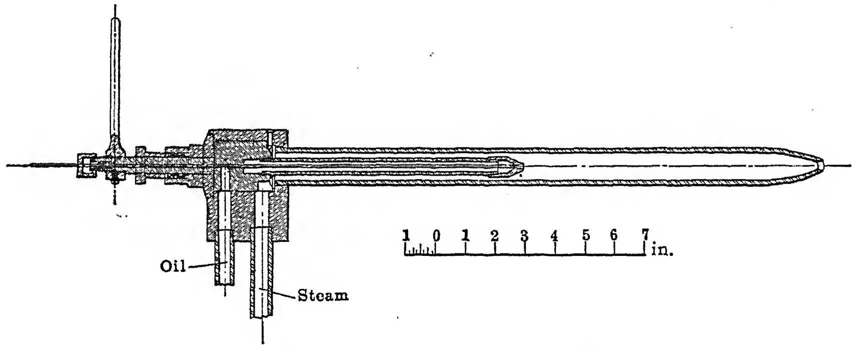


Fig. 222.—Hayes oil-burner.

distilling zinc-silver-lead crust, and the Kirkwood[3] burner used in open-hearth furnaces.

(4) Injector Burners, Fig. 223.—This type resemble in principle the injectors used for feeding boilers. The steam passing through the injector

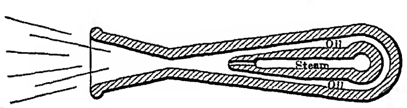


Fig. 223.—Injector oil-burner.

sucks the oil, mingles with it in the cone-shaped passage and forces the mixture outward through the reversed flaring cone with considerable velocity. This class of burners has become a standard in Southern Russia. Fig. 224 represents

[1] Burtonshaw, *Am. Mfr. and Iron World*, 1899, LXVI, 546.
Baker, *Iron-Steel Mag.*, 1904, LXXI, 21.
Hausenfelder, *Stahl u. Eisen*, 1912, XXXII, 772.

[2] *Eng. and Min. J.*, 1907, LXXXIII, 84.

[3] *Iron Age*, 1907, LXXIX, 192; LXXX, 492.

the burner of the *Pennsylvania Railroad*, which is a modification of the Urquhart (Russian) burner. Other examples are the Shelby burner of Cananea[1] and the Spur burner used in open-hearth furnaces.

(5) Projector Burners, Fig. 225.—The oil is pumped to the orifice and then caught and blown from this by a passing jet of steam. In the Oil City

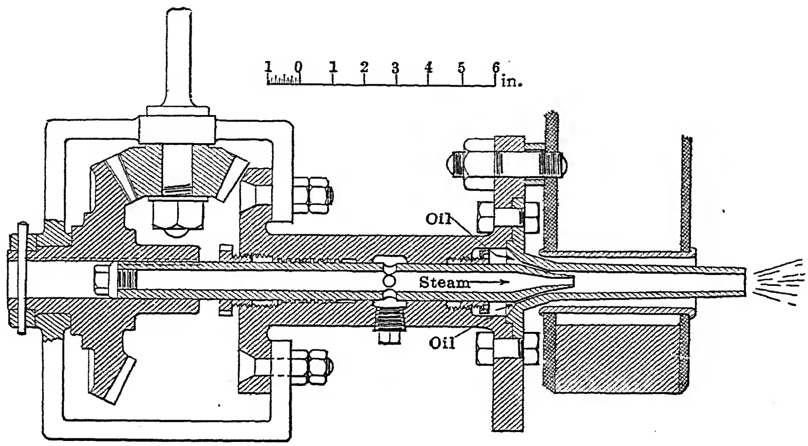


Fig. 224.—Pennsylvania Railroad oil-burner.

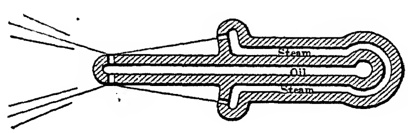


Fig. 225.—Projector oil-burner.

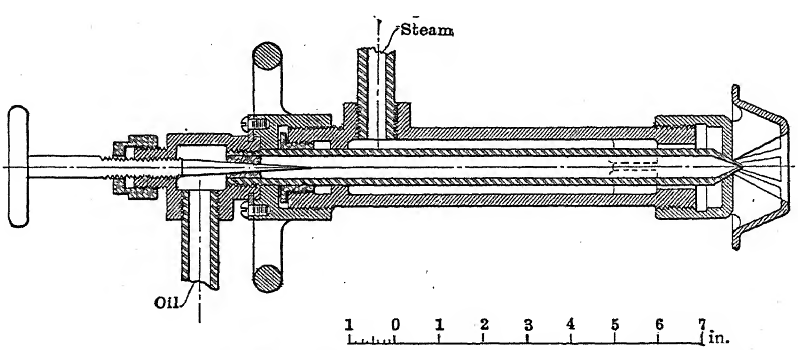


Fig. 226.—Oil City oil-burner.

Oil-burner, Fig. 226, a hollow jet of steam converges upon the stream of oil within a funnel-shaped bell from which the mixture is ejected giving a wide and short flame (the so-called rose-flame). Another example is the Kittle burner.[2]

[1] Shelby, *Eng. and Min. J.*, 1910, LXXXIX, 31; XX, Editor, *Iron Age*, 1909, LXXXIII, 731.
[2] *Mines and Minerals*, 1910, XXX, 367.

(6) Form of Flame.—The form of flame that these five classes of burners can throw depends upon the form of the tip. Thus with simple circular openings a conical flame will be formed; with oblong openings, the flame is flat; with ring-shaped openings, fan-tailed; with concentric slots, rose-form. The form of spray from a conical opening can be further modified by a cone reaching into the nozzle, by a sharp edge or an inverted cone from the outside, and other devices.

154. Working of Oil, Coal and Coke Compared.[1]—According to a series of tests carried on by Felton[2] at the Pennsylvania Steel Works, Steelton, Pa., using the Archer[3] gas-fuel producer, there were required, for heating 14-in. ingots, six to a charge, in two Siemens heating-furnaces, 6.5 gal. oil per ton of bloom; for smelting pig, scrap and ore charges in a 30-ton open-hearth furnace 48 gal. per ton of ingot, and in a 5-ton furnace 50 to 55 gal.; in generating steam in two 100 h.p. tubular boilers with feed water at 70° C., 12 lb. water were evaporated per lb. oil. The Archer producer consumed 0.50 to 0.75 lb. pea coal per gal. oil. Potter's records[4] of tests made at the South Chicago steel works with 14 tubular boilers, 16×5 ft., showed that 3.22 to 3.31 barrels (@ 31.5 gal.) were equivalent to 1 ton of Indiana block coal. Billings[5] found that in puddling with a furnace in good working order 1 gal. oil was required per ton of puddled bloom. Campbell[6] states that in regenerative furnaces and under boilers 50 gal. oil will do the work of 1,000 lb. of soft coal. Richmond-Hamilton[7] in melting zinc precipitate from the cyanide process, found that for 1,000 oz. precipitate, 4.46 gal. burned in 2.25 hr., were equal to 93 lb. coke in 3.40 hr. Young[8] in heating an assay muffle, found that the consumption of 25 lb. good or 34 lb. inferior coal per hr. was balanced by 2.75 gal. of oil of 24° Bé.

155. Apparatus for Burning Gaseous Fuels.[9]—In order to burn gaseous fuel successfully, it is essential that gas and air be intimately mixed and that the kindling temperature of the gas be maintained in the space where the gas is to be burnt. There are two methods of mixing gas and air.

(1) They are divided into a number of small parallel or slightly converging jets traveling in the same direction. When they issue from the nozzle of the burner, they become well mixed on account of the large contact surfaces. The

[1] Hamilton, "The Use of Oil for Smelting," *Eng. and Min. J.*, 1911, XCI, 224.

[2] *Tr. A. I. M. E.*, 1889–90, XVII, 809.

[3] *Eng. and Min. J.*, 1890, XLIX, 259, 678.

[4] *Tr. A. I. M. E.*, 1889–90, XVII, 807.

[5] *Op. cit.*, p. 808.

[6] "The Manufacture and Properties of Iron and Steel," New York, 1903, p. 247.

[7] *Mines and Minerals*, 1907, XXVII, 370.

[8] *Min. Sc. Press*, 1907, XCIV, 700.

[9] Ledebur, A., "Die Gasfeuerungen für Metallurgische Zwecke," Felix, Leipsic, 1891.

Pütsch, A., "Gasfeuerungen," Seydel, Berlin, 1880; "Neue Gasfeuerungen," Simon, Berlin, 1888; "Neuere Gas- und Kohlenstaub-feuerungen," Simon, Berlin, 1899 (tr. by C. Salter), Von Nostrand, New York, 1901.

Steinmann, F., "Compendium der Gasfeuerung," Felix, Leipsic, 1900.

more intimate the mixture, the higher will be the temperature at the meeting point and the shorter the flame. The method should, therefore, be applied to small fireplaces, such as forges, annealing ovens, muffles, crucible furnaces,[1] etc. Here the efficiency[2] with producer gas is 75 per cent.; with petroleum 30 per cent., and with solid fuel 20 per cent.

(2) They are admitted into the combustion chamber in the form of parallel or slightly converging sheets through oblong ports. These are placed either side by side, in which case a gas-port will have on either side an air-port, or above one another, in which case the air-ports will be placed above the gas-ports, as the gas, being lighter, has a tendency to rise. Occasionally air is admitted both above and below the gas. It is advisable to have air and gas travel at a different speed, as the friction between the two bodies favors mixing; usually the velocity of the gas is kept greater than that of the air. This method of mixing will be used in reverberatory furnaces requiring a long flame, the length of which is governed in part by the inclination of the gas- and air-flues. In a Siemens open-hearth furnace (§325) where the flame has to travel 30 ± ft. the inclination will be less than in a Siemens crucible furnace where the distance is 2 ft. and less.

Kindling Temperature.—The kindling temperature of gaseous fuel is about 650° C. With gas and air at ordinary temperature, the combustion is slow and likely to be imperfect on account of the slow speed of the flame; this is especially the case with producer gas or iron blast-furnace gas containing 70 per cent. inert gases. The speed increases with the temperature, and combustion becomes almost instantaneous at the kindling temperature. For high temperatures it is therefore necessary that gas and air be heated separately; heating a mixture would cause an explosion. If the furnace chamber is at an elevated temperature, 1,000+° C., as *e.g.*, in zinc, copper, iron, etc., smelting furnaces, the walls will radiate enough heat to raise gas and air to the kindling temperature; if the temperature is low (below 600° C.), as in some heating ovens and especially boilers,[3] there will have to be provided a small combustion chamber filled with checker-work which, first brought to the kindling temperature of the gas, will heat gas and air and, causing quick combustion, maintain the small chamber at or above the kindling temperature; the products of combustion then will pass into the large furnace chamber and keep this at the desired low temperature. Cool producer gas and iron blast-furnace gas used to be always heated to the kindling temperature by conducting them over a fireplace on the grate on which fuel was burning; at present, at least with heating stoves they are led into a hot combustion chamber containing more or less checker-work. Thus, in case the gas gives out momentarily or the supply is accidentally insufficient to furnish a flame, the coal fire or the combustion chamber will maintain the heat necessary to ignite the gas when it arrives again in sufficient quantity and thus prevent explosions from occurring. The auxiliary fireplace of gas-fired

[1] Loomis, *Tr. A. I. M. E.*, 1890–91, XIX, 995.

[2] Reichhelm, *Am. Machinist*, 1895, XVIII, 21.

[3] Rowan, *Coll. Eng.*, 1889–90, X, 52, 88, 114, 126 and 150.

boilers has the advantage that in case of prolonged disturbances of the iron blast-furnace, steam can be maintained independently. Hot producer gas kindles readily when it comes in contact with superheated air; water gas, as a rule, is not superheated, but the temperature of the air is raised.

AIR SUPPLY.—The air supply for gaseous fuel must be carefully regulated; the excess air is about 10 per cent. when gas and air have been superheated. It is larger when this is not the case.

PREHEATING.—Preheating of gas is discussed in §321. The details of the apparatus used for burning gas in metallurgical furnaces are so closely connected with the preheating (Boëtius, Bicheroux, Ponsard, Siemens, Pütch . . .) that they will be treated in connection with that subject (§§321–325).

156. Working of Industrial Fuels Compared.—Tests made by Livingston[1] in a double-deck tubular boiler 15×5 ft. to ascertain the relative values of solid, liquid and gaseous fuels showed that 1 lb. anthracite evaporated 9.70 lb. water from and at 100° C.; 1 lb. bituminous coal 10.14 lb.; 1 lb. fuel oil, 36° Bé, 16.48 lb., and 1 cu. ft. gas (obtained by distillation of oil) of 20 C.P. (equal in calorific power to natural gas or coal gas of same C.P.) 1.28 lb. of water. A second set of tests by Reichhelm[2] with small furnaces showed that in regular work, representing 1,000,000 heat units = 400,000 Cal.; the same amount of metal would be melted by 150 lb. coal; 10 gal. crude petroleum; 4.52 gal. naphtha (as gas); 9,000 cu. ft. producer gas; 7,000 cu. ft. mixed producer and water gas; 4,250 cu. ft. water gas (coke); 3,533 water gas (bit. coal); 2,030 cu. ft. carbureted water gas; 1,960 cu. ft. coal gas (20 C.P.); 1,922 cu. ft. gasoline gas, or 1,333 cu. ft. natural gas.

[1] *Proc. Eng. Club, Phila.*, 1892, IX, 82.

[2] *Am. Machinist*, 1895, XVIII, 22.

CHAPTER VII

REFRACTORY MATERIALS[1]

157. **In General.**—The ideal refractory material is one that resists heat, sudden change of temperature, pressure and chemical combination, and is at the same time a poor conductor of heat and electricity. No material exists which meets all these requirements. The inner walls of furnaces should be built of materials adapted to resist the destructive effects of the processes that are being carried on.[2] The outer walls usually are of non-refractory red brick, and the whole is ironed to meet expansion, when the furnace is heated, and to strengthen the structure.

Refractory materials are mainly composed of $Al_2O_3 . xSiO_2$, SiO_2, Al_2O_3, CaO, MgO, $Fe_xO_y-Cr_xO_y$, C and metal. With the exception of the last, every one of these constituents by itself is infusible in ordinary metallurgical furnaces, but becomes more or less fusible in contact with others. Most of them are mechanically so weak as not to be suitable as a building material. In order that they may resist change of temperature and pressure, it is necessary to mix refractory substances, if they do not occur already mixed in nature, and bear the consequences of diminished refractoriness. As acid materials in contact with basic readily form slags, it is advisable to have acid materials (quartzite, ganister, silica brick, etc.), for acid processes, and basic (magnesite, dolomite brick, etc.) for basic. Between these two come the so-called neutral materials (fire-clay, chromic iron ore, carbon, metal, etc.) suited for both acid and basic processes and therefore frequently used, especially fire-clay and water-cooled metal. Formerly siliceous refractory materials, such as quartzite, granite, mica-schist, were used quite generally in the natural state; at present most refractories are

[1] Percy, J., "Fuel, etc.," Murray, London, 1875, pp. 87–154.

Sexton, A. H., "Fuel and Refractory Materials," Blackie & Son, Manchester, 1909.

Bischof, C., and Kaul, H., "Die Feuerferten Tone," Quandt-Händel, Leipsic, 1904.

Bischof, C., "Gesammelte Analysen der in der Tonindustrie benutzten Materialien, etc.," Quandt-Händel, Leipsic, 1901.

Wernicke, F., "Die Fabrikation der Feuerfesten Steine," Springer, Berlin, 1905.

Kerl, B., Cramer E., and Hecht, H., "Handbuch der Gesammten Thonwaarenindustrie," Vieweg, Brunswick, 1907.

Werner, P., "Die Feuerfeste Industrie," Hartleben, Vienna, 1911.

Granger, A., "Fabrication et Emploi des Matériaux et Produits Réfractaires, etc.," Beranger, Paris, 1910.

Havard, F. T., "Refractories and Furnaces," McGraw-Hill Book Co., New York, 1912.

Baraduc-Muller, *Rev. Mét.*, 1909, VI, 701; *Thonind. Z.*, 1911, XXXV, 794; *Metallurgie*, 1910, VII, 314.

[2] Harbison-Walker, "Refractories Suited for Different Types of Furnaces," *Iron Trade Rev.*, 1909, XLIV, 108.

Blasberg, *Stahl u. Eisen*, 1910, XXX, 1055; *Met. Chem. Eng.*, 1910, VIII, 475.

first molded into bricks; in some cases fire-resisting materials are crushed to sand-size, and then either rammed or burned in place to constitute the working-bottom of a furnace. Refractory materials, when mined, have to be subjected to some preparatory mechanical treatment before they can be molded into bricks. The general operations of hand-sorting, crushing, screening, washing, drying, calcining, mixing, molding and baking are more or less common to all of them; they differ in detail with the several materials.

158. Fire-clay and its Properties.[1] CONSTITUTION AND FUSIBILITY.—Fire-clay is essentially a fine-grained mixture of a hydrosilicate of alumina, and fragments of other minerals, which possesses some plasticity when wet, becomes hard and rock-like when burnt at a red heat, and does not melt before Seger cone No. 26 (1,580° C.). Kanolt[2] found the mean melting point of 41 samples of fire-brick at 1,649° C.; other experiments by Ruff-Goecke.[3] The leading associate mineral is quartz; in addition there are usually present small amounts of feldspar, mica and other silicates, iron oxides, titanic acid, some colloidal material and organic matter.

KAOLINITE.—The common hydrosilicate of alumina is kaolinite[4] ($Al_2O_3.2SiO_2 + H_2O$; SiO_2 46.3, Al_2O_3 39.8, H_2O 13.9 per cent.); other hydrosilicates, such as halloysite, pholerite, allophane, etc., often replace kaolinite more or less. Kaolinite is often called the "clay-base" or "clay-substance,"[5] *i.e.*, that part of the clay which is decomposed with hot solutions of H_2SO_4 and Na_2CO_3. When pure, it is a white, soft (hardness 2.0 to 2.5), light (sp. gr. 2.2 to 2.65), compact,

[1] Branner, J. C., "Bibliography of Clay and the Ceramic Arts," published by American Ceramic Society, Columbus, O., 1907, pp. 451. Bischof, *op. cit.* Ries, H., "Clays, their Occurrence, Properties and Uses," Wiley, New York, second edition, 1909. Roland, P., "Die Tone," Hartleben, Vienna, 1909. Wernicke, *op. cit.* Granger, *op. cit.* *Ann. Reports*, U. S. Geol. Surv. "Mineral Resources," since 1882. "The Mineral Industry," since 1892. *Reports* of the several states. The leading ones are: Cook, G. H., Geol. Surv. of N. J., "Report on Clay Deposits," Trenton, 1878. Ries, H., Geol. Surv. of N. J., "Clay Industry," Trenton, 1904. Orton, E., Jr., Geol. Surv. of O., vol. VII¹, "Clay Working Industries," Columbus, 1893. Wheeler, H. A., Geol. Surv. of Mo., "Clay Deposits," Jefferson City, 1896. Blatchley, W. S., Dept. of Geol. and Nat. Res. of Ind., *Twentieth Ann. Rep.*, "A Preliminary Report on the Clays and Clay Industries," Indianapolis, 1896; *Twenty-second Ann. Rep.*, "The Clay and Clay Industries of Northwest Ind., 1898. Hopkins, T. C., "Clays and Clay Industry of Pa.," Pennsylvania State College, 1897. Buckley, E. R., Wisc. Geol. and Nat. Hist. Surv., *Bull.* No. 7, "The Clays and Clay Industries of Wisc.," Madison, 1901. Beyer, S. W., and Savage, T. E., Iowa Geol. Surv., vol. XIV, "Technology of Clays," Des Moines, 1904. Ladd, G. E., Geol. Surv. of Ga., *Bull.* No. 6, "Preliminary Report on a Part of the Clays of Ga.," Atlanta, 1898. Ries, H., *Bull.*, N. Y. State Museum, "Clays of N. Y.," Albany, 1900. Ries, H., and Leighton, H., "History of the Clay-working Industries of the U. S.," Wiley, New York, 1909. Sokoloff, "Determination of Physical Properties," *Thonind. Z.*, 1909, XXXIII, 1256, 1286, 1296. Heyn, Microscopical Examination, *Thonind. Z.*, 1907, XXXI, 530.

[2] *Technical Paper* 10, U. S. Bureau of Standards, 1912.

[3] *Thonind. Z.*, 1911, XXXV, 1395, 1424.

[4] Hickling, *Trans. Inst. Min. Eng.*, 1908, XXXVI, 10.
Mellor-Holdcroft, *Tr. Engl. Ceram. Soc.*, 1910–11, X, 94; *Thonind. Z.*, 1911, XXXV, 1383.

[5] Binns, *Tr. Am. Ceram. Soc.*, 1912, XIV, 815.

friable to mealy mass which as a rule is dull, but occasionally has a pearly luster; usually it has an unctuous feel and is weakly plastic; it has a clay odor, adheres slightly to the tongue, shrinks considerably upon heating and fuses at the temperature indicated by Seger cone No. 36 (1,790° C.).

SILICA.—Silica[1] is found in all fire-clays, ranging from <1 per cent. in pure highly refractory, to perhaps 30 per cent. in sandy semi-refractory, clay. It is usually invisible. Microscopic examination shows it to be angular in residual, and rounded in sedimentary clays; sometimes it is colorless, oftener, however, stained with iron. It expands upon heating (§161), thus counteracting the shrinkage of clay, and is nearly as infusible as kaolinite, melting with Seger cone No. 35 (1,770° C.) or 1,735 to 1,740°.[2] At a low temperature it therefore acts

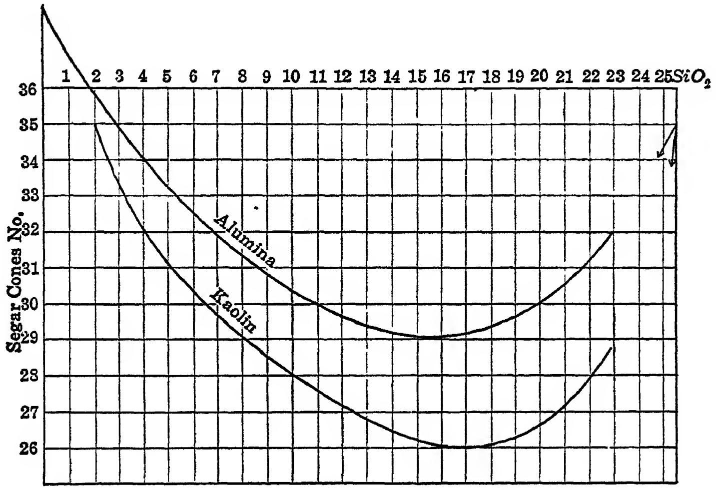


FIG. 227.—Seger's freezing-point curve of alumina-silica and kaolin-silica mixtures.

as a refractory agent, but at a high temperature it fluxes kaolinite. This is shown by Seger's curve[3] represented in Fig. 227. The ordinate represents the melting-points of Seger cones (Table 67) Nos. 26 to 36, the abscissa the proportions of Al_2O_3, or kaolinite, and SiO_2; the upper curve denotes the behavior of Al_2O_3, the lower that of kaolinite from Zettlitz, Bohemia (analysis, Table 142). In the upper curve the least fusible mixture consists of 1 Al_2O_3 and 2 SiO_2, and melts at a temperature indicated by cone No. 36. The fusibility increases with the amount of SiO_2 up to the proportion 1 Al_2O_3 : 17 SiO_2, and then decreases until the effect of Al_2O_3 has disappeared and the melting-point of SiO_2, equal to cone No. 35, has been reached. In the lower curve the minimum is reached with 1 kaolin : 18 SiO_2. Both curves show that we have to deal with a eutectic mixture. As SiO_2 not in combination with Al_2O_3 acts as a flux at high temperatures, a fire-clay low in free SiO_2 will be more refractory than

[1] Kraze, *Thonind. Z.*, 1908, XXXII, 934.
[2] Kanolt, *Technical Paper* 10, U. S. Bureau of Standards, 1912.
[3] *Thonind. Z.*, 1893, XVII, 391; *Tr. A. I. M. E.*, 1894, XXIV, 43.

one high in SiO_2, supposing the percentage of basic fluxes to be the same; or, the greater the percentage of Al_2O_3 the greater the infusibility.

If the Al_2O_3- and SiO_2-contents of Seger-cone mixtures are calculated in percentages and deg. C. substituted for cone-numbers, and the data plotted in a rectangular system of coordinates, as has been done by Neumann[1] there will be obtained the curve shown in Fig. 228. In it the chemical compound $Al_2O_3.SiO_2$ (andalusite, sillimanite, cyanite) with 62.9 per cent. Al_2O_3 and 37.1 per cent. SiO_2 is placed at 1,850° C., which appears justifiable from Seger's own record and from other researches, such as those of Stein,[2] Shepherd-Rankine[3] and others. Accepting this view there are present in the Al_2O_3–SiO_2 series, one eutectic Al_2O_3–$Al_2O_3.SiO_2$, the chemical compound $Al_2O_3.SiO_2$, and a second eutectic $Al_2O_3.SiO_2$–SiO_2.

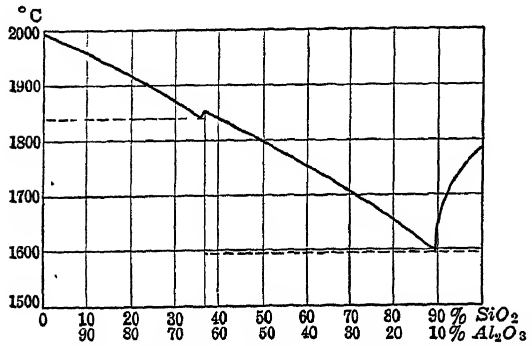


FIG. 228.—Seger-Neumann, freezing-point curve of alumina-silica mixtures.

FELDSPAR.—Feldspar, mica and other silicates in fire-clays are the cause of the presence of small quantities of MgO, CaO,[5] FeO, Na_2O, K_2O, all of which act as fluxes and lower the melting-points.[6] Richters[7] in 1868 propounded the theory that the fluxing effects of these oxides was proportional to their molecular weights; thus 40 MgO would have the same slagging effect as 56 CaO, 72 FeO (166 Fe_2O_3), 62 Na_2O, 94 K_2O. Later experiments by Cramer[8] showed that this theory holds good only for kaolinite, and that in the presence of uncombined SiO_2, the above fluxing order has to be changed to FeO, MgO, CaO, Na_2O, K_2O. Ludwig's[9] researches also confine Richter's theory to pure clays which when fused are to be considered as dilute solutions and therefore follow the general law that "equi-molecular quantities of different substances dissolved in equal amounts of the same solvent lower the melting-point to the same degree."[10]

MELTING-POINT OF CLAY.—As stated in the definition, a clay ceases to be a fire-clay when it melts before Seger cone No. 26; and a clay that contains, when

[1] *Stahl u. Eisen*, 1910, XXX, 1505.
[2] *Zt. anorg. Chem.*, 1907, LV, 159.
[3] *Am. J. Sc.*, 1909, XXVIII, 301.
[4] Hottinger, *Tr. Am. Cer. Soc.*, 1903, V, 130.
[5] Cramer, *Thonind. Z.*, 1901, XXV, 976.
[6] Cramer, "Volatility and Effect," *Thonind. Z.*, 1897, XXI, 288.
[7] *Dingler, Pol. J.*, 1896, CXCI, 59, 150, 229; 1870, CXCVII, 268, reprint, Thonindustrie-Zeitung, Berlin, 1897.
[8] *Thonind. Z.*, 1895, XIX, 633, 647; 1897, XXI, 288.
[9] *Op. cit.*, 1904, XXVIII, 773.
[10] Bleininger, *Tr. Am. Cer. Soc.*, 1905, VII, 276.

TABLE 141.—ANALYSES AND FUSIBILITY-TESTS OF SOME AMERICAN FIRE-CLAYS[1]

Locality	Al_2O_3	SiO_2 comb.	H_2O comb.	Total clay material	SiO_2 non-comb.	TiO_2	Total sandy material	MgO	CaO	K_2O	Na_2O	Fe_2O_3	Total fusing material	H_2O hygroscopic	Loss on ignition	Sum total	Seger cone No.
Zanesville, O.	22.95	64.26	6.74	93.95	incl.	in comb	. SiO_2	0.37	0.45	1.81	0.15	1.28	4.06	2.05	8.79	100.06	26
Salineville, O.	26.60	56.44	7.57	90.61	"	"	"	0.63	0.47	3.20	0.26	2.00	6.56	2.48	10.05	99.65	27/26
New Brighton, Pa.	24.88	63.55	6.96	95.39	"	"	"	0.47	0.56	2.27	....	1.17	4.47	1.38	8.34	101.24	28
East Palestine, O	25.12	59.54	7.75	92.41	"	"	"	0.51	0.57	1.95	......	1.57	4.60	2.63	10.83	99.64	29/28
St. Louis, Mo.	21.16	26.03	8.94	56.13	38.	32	38.32	0.30	0.61	0.86	0.88	2.72	5.37	2.62	11.57	99.82	31/30
Golden, Colo	31.72	45.99	13.30	91.01	5.	22	5.22	0.23	0.36	0.48	0.45	0.75	2.27	0.83	14.13	98.50	32/31
Parall, O.	24.14	32.03	10.22	70.39	26.	81	26.81	0.28	0.49	0.15	0.46	1.67	3.05	1.23	11.45	100.25	32
St. Louis, Mo.	23.26	27.56	10.20	61.02	31.	82	31.82	0.42	0.65	0.54	0.79	3.24	5.64	2.74	12.94	98.48	32/33
Mineral Point, O.	31.84	35.39	11.68	78.91	17.13	1.68	18.81	0.19	0.50	0.59	...	0.67	1.95	0.69	12.37	100.36	33
Sand Hills, N. J.	35.94	36.20	12.10	84.24	12.20	1.50	13.70	......	......	0.08	.....	0.96	1.04	1.10	13.20	100.08	34
Mt. Savage, Md.	38.14	41.93	12.62	92.69	5.90	5.90	5.90	0.17	tr.	0.49	0.42	1.31	2.39	1.18	13.80	100.98	34/35
Sayreville, N. J.	38.66	41.10	13.55	93.31	33.10	1.20	4.30	......	......	0.28	0.18	0.74	1.20	1.00	14.55	99.81	35

[1] Hofman, *Tr. A. I. M. E.*, 1895, XXV, 14, 16.
Other analyses, State Geological Surveys.

TABLE 142.—ANALYSES AND FUSIBILITY-TESTS OF SOME FOREIGN FIRE-CLAYS
(Bischof's Standard Series[1])

Fire-clay		Al_2O_3	SiO_2			Fluxing material						
Locality	No.	Per cent.	Combined per cent.	Non-combined per cent.	Total per cent.	MgO per cent.	CaO per cent.	K_2O per cent.	Fe_2O_3 per cent.	Total per cent.	Loss by ignition, per cent.	Corresponding Seger cone
Saarau, Silesia	I.	36.30	38.94	4.90	43.84	0.19	0.19	0.42	0.46	1.26	17 78	36
Zettlitz, Bohemia	II.	38.54	40.53	5.15	45 68	0 38	0 08	0.66	0.90	2.02	13.00	35
Andennes, Belgium	III.	34.78	39 69	9 95	49.64	0 41	0.68	0.41	1.80	3.30	12 00	33
Coblence, Rhen. Prussia	IV.	36.00	41.00	6.74	47.74	0.33	0.40	1.05	2.57	4.35	11.81	32
Grünstädt, Palatinate	V.	35 05	39.32	8 01	47.33	1.11	0.16	3.18	2.30	6 75	10 51	30
Cassel, Hesse[2]	VI.	27.97	33.59	24.40	57.99	0.54	0 97	0.53	2.01	4.05	9 43	28
Niederpleis, Nassau	VII.	28.05	30.71	27.61	58.32	0 75	0 72	1.39	1.89	4 75	8 66	26

[1] *Tr. A. I. M. E.*, 1894, XXIV, 45 and 46. Analyses of Other Foreign Clays: *Nineteenth Ann. Rep.*, U. S. Geol. Surv., 1897–98, VI[2], 626 (Ries); Bischof's "Gesammelte Analysen."
[2] "Recent Analysis of Grossalmerode Fire-clay," *Thonind. Z.*, 1912, XXXVI, 1471.

free from H_2O, more than 6 per cent. fluxes will do this. Clays of the highest degree of refractoriness fuse with Seger cone No. 36. Ries,[1] similarly to Jochum,[2] calls clays melting after Seger cone No. 33 highly refractory; melting between 31 and 33, refractory; between 27 and 30, semi-refractory, between 20 and 26, lowly refractory.

TiO_2.—TiO_2 is found in most clays, rarely, however, in amounts exceeding 2 per cent.; it begins to act as an acid flux at a temperature lower than does SiO_2. The usual small quantities present have little influence upon the fusibility. Seger,[3] Ries[4] and Riecke[5] have studied its effects.

ORGANIC MATTER.—Organic matter, frequently found in clays, gives them a dark color. As a rule, it is finely divided and harmless, being oxidized when the clay is burnt, and passing off with the vapors and gases.

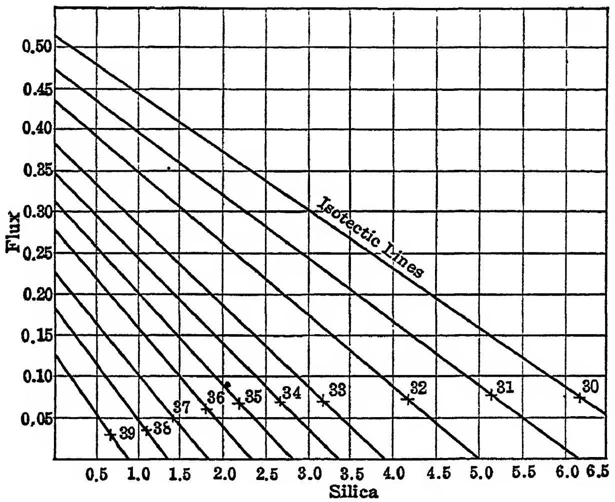


FIG. 229.—Ludwig's isotectic lines of refractory clays.

The interpretation of the ultimate analysis of a fire-clay as regards refractoriness is not simple. In a general way it is clear that the closer the percentages of Al_2O_3, SiO_2 and combined H_2O approach those of kaolinite, and the smaller the amount of free SiO_2 and of fluxing impurities, the purer and, therefore, the more refractory will be the clay. The first to give a numerical expression for refractoriness was Bischof[6] whose "refractory quotient" is expressed by $\frac{\text{O in } Al_2O_3}{(\text{O in RO})\times(\text{O in } SiO_2)}$ in which RO is the sum of fluxes. Its validity has been

[1] Geol. Surv. of N. J., "Clay Industry," 1904, p. 100.

[2] "Die Grenzen der Feuerbeständigkeit feuerfester Producte und die Definition der relativen Begriffes 'feuerfest,'" Thonindustrie Zeitung, Berlin, 1904, p. 7.

[3] *Thonind. Z.*, 1883, VII, 243, "Collected Works," I, 519.

[4] Geol. Surv. of N. J., "Clay Industry," 1904, p. 70.

[5] *Thonind. Z.*, 1908, XXXII, 1424.

[6] "Die Feuerfesten Thone," 3d, p. 63; Hofman, *Tr. A. I. M. E.*, 1894, XXIV, 44.

questioned by Seger,[1] Wheeler[2] and others who have proposed other formulæ. The most recent investigation of the subject is that by Ludwig[3] whose deductions are based upon 85 samples of analyzed fire-clays tested for fusibility with Seger cones. He starts with the fact that a clay, when melted, forms an igneous solution holding dissolved various silicates. This solution obeys the general law of solutions stated above, hence in interpreting fusibility from the ultimate analysis only the effects of gram-molecules of components can be considered, as was done by Richters, and not those of weight-percentages, as has been done by others. A kaolin of Grünstadt, *e.g.*, has the composition Al_2O_3 38.15, SiO_2 47.69, TiO_2 0.12, Fe_2O_3 0.77, CaO 0.21, K_2O 1.26. Dividing each component by its molecular weight gives Al_2O_3 0.37402, SiO_2 0.79484, TiO_2 0.00150, Fe_2O_3 0.00963, CaO 0.00374, K_2O 0.01340; making Al_2O_3 unity, changes these figures to Al_2O_3 1, SiO_2 2.125, TiO_2 0.004, Fe_2O_3 0.0257, CaO 0.0100, K_2O 0.01340. As equal gram-molecules of fluxes have equal effects upon fluxing, their combined effects can be expressed by their sum, hence the simplified refractory expression is 1 Al_2O_3+2.125 SiO_2+0.0755 F (lux). By plotting the calculated values of the 85 fire-clays examined in a system of rectangular coordinates, in which the abscissæ represent values for SiO_2, and the ordinates those for F, and marking the melting-points in values of Seger cones, there were obtained the "isotectic" (equal-melting) lines shown in Fig. 229. The numbers in these lines represent the locations of the Seger cones. In estimating the fusibility of a fire-clay from the ultimate analysis, one simply calculates the molecular proportions of SiO_2 and F to Al_2O_3 as unit, finds the position in the plot and thereby the fusibility. The dot in the diagram represents the clay from Grünstadt.

Another method of arriving at the refractoriness of a fire-clay is by making a "rational analysis"[4] which separates the clay into three constituents: kaolinite, quartz and feldspar (flux). The original method as improved by Sabeck[5] is as follows: boil 5 g. finely pulverized unburnt clay for 2 hr. with 50 c.c. conc. H_2SO_4 and 100 c.c. H_2O in a casserole to decompose and dissolve kaolinite, decant the solution with floating flakes of hydrosilicate into a beaker containing 2 lit. H_2O, allow to settle 1 hr. and decant the clear solution, join the residues, boil with 10 c.c of a 33-per cent. NaOH solution to dissolve amorphous SiO_2, dilute, decant, wash, treat for 5 min. with conc. HCl to dissolve AlO_3H_3. Repeat the treatment with NaOH and HCl, filter, ignite and weigh. The difference

[1] *Thonind. Z.*, 1877, I, 290, 296; 1889, XIII, 332; 1893, XVII, 339; "Collected Works," I, .468

[2] *Eng. Min. J.*, 1894, LVII, 224, 244; Geol. Surv. of Mo., "Clay Deposits," 1896, p. 146.

[3] *Thonind. Z.*, 1904, XXVIII, 783.

[4] Seger, "Collected Works," I, 46, 537.

Ries, *Ann. Rep.*, U. S. Geol. Surv., 1906–07, V, 1122; *Tr. A. I. M. E.*, 1898, XXVIII, 160.

Zschokke, *Thonind. Z.*, 1902, XXVI², 1742.

Binns, *Tr. Am. Cer. Soc.*, 1906, VIII, 198.

Bollenbach, H., "Laboratoriumsbuch für Tonindustrie," Knapp, Halle, 1910, p. 24, also *Chem..Ind.*, 1908, XXXI, 45.

"Purdy's Attacks," *Tr. Am. Ceram. Soc.*, 1911, XIII, 200; 1912, XIV, 359.

[5] *Chem. Ind.*, 1892, XXV, 90.

TABLE 143.—ULTIMATE AND RATIONAL ANALYSES OF CLAYS

	(1)	(2)	(3)	(4)	(5)	(6)	(7)	(8)	(9)	(10)	(11)	(12)
	Ultimate analysis											
SiO_2	62.40	62.52	63.17	64.87	63.07	54.51	53.10	47.60	46 61	58 39	46 82	57.08
Al_2O_3	26.51	25.57	25.09	23.83	24.67	31.41	33.06	34.00	36.47	27.52	38.49	29.94
Fe_2O_3	1.14	0.92	0.64	0.83	0.59	0 68	1.18	1 30	2 81	0 36	1.09	0.65
CaO	0.57	0.65	0.35	....	....	0.04	0.38	Tr.	0.14	1.52	....	....
MgO	0.01	0.10	0.26	0.50	0.40	0 43	0 08	0.50	....	0.41	Tr.	0.49
Alkalies	0.98	1.04	0.80	1.39	4.25	0.55	0.83	3.00	1.44	4 29	1 40	2 26
Loss by ignition	8.80	9.27	9.70	8.36	7.00	12.37	11.32	13.60	12.80	7.19	12 86	9 87
	100.41	100.07	100.01	99.78	99.98	99.99	99.95	100.00	100.27	99.68	100.66	100.29
	Rational analysis											
Clay-substance	66.33	72.05	67.82	63.77	54.92	83.04	83.39	88.34	96.08	55.88	96.55	74.09
Quartz	15.61	27.78	30.93	35.50	23.52	16.28	14.99	8.95	1.93	5.95	2.30	17.21
Feldspar	18.91	0.10	1.25	0.73	21.56	0.68	1.57	2.73	1.99	38.17	1.15	8.70
	100.85	99.93	100.00	100.00	100.00	100.00	99.95	100 02	100 00	100.00	100.00	100.00

(1) Crude kaolin, Springer mine, Webster, N. C. *Bull. N. C. Geol. Surv.*, on "Clays of North Carolina."
(2) Slip-clay from Rühle's mine, Löthain, Saxony. *Thonind. Z.*, 1892, p. 1031.
(3) Slip-clay from Kaschkau, Germany. *Ibid.*
(4) Kaolin from Sennewitz, Saxony. *Notizblatt*, 1876.
(5) Porcelain-clay mixture. *Ibid.*
(6) White earthenware-clay, Löthain, Saxony. *Seger's Ges. Schr.*, p. 887.
(7) Kaolin (unwashed), West Mills, N. C. *Bull. N. C. Geol. Surv.*, on "Clays of North Carolina."
(8) White earthenware-clay, Wiesau, Germany. *Thonind. Z.*, 1894, p. 358
(9) Fire-clay, Bautzen, Germany. *Ibid.*, 1894, p. 842.
(10) Kaolin, Limoges, France. *Seger's Ges. Schr.*, p. 552.
(11) Kaolin, Zettlitz, Bohemia. *Ibid.*, p. 50.
(12) Kaolin, Lettin. Saxony. *Ibid.*, p 50.

between original weight and residue gives the kaolinite. Determine Al_2O_3 in the residue (consisting of quartz and feldspar) and figure it as feldspar (102 Al_2O_3: 556 feldspar) and deduct this from the weight of the residue; the difference is quartz. Any mica[1] present is decomposed with the kaolinite as well as the usual small amounts of Fe_2O_3, $CaCO_3$, $MgCO_3$. If present in any appreciable amount, these bases would have to be determined by ultimate analysis, but then the clay would have ceased to be refractory. The amounts of mica and Fe_2O_3 can be arrived at by a more elaborate procedure[2] than outlined above. In Table 143 compiled by Ries[3] the clay containing the smallest amount of feldspar will be the most refractory. The ultimate analyses of clays Nos. 1 and 2 are very much alike, but the rational analyses show that No. 1 has 6 per cent. less clay-substance (kaolinite), 12 per cent. less quartz and 19 per cent. more feldspar (flux) than No. 2, and is therefore much less refractory. Samples Nos. 3 and 5 and 10 and 11 give similar evidence. With samples Nos. 6 and 7 and 9 and 11, both the ultimate and rational analyses are very similar, hence the clays show a similar general behavior as to fire-resistance and shrinkage.

The fusibility of a clay is, however, influenced not only by its chemical composition, but also by its texture, *i.e.*, the relative size and intimate contact of the particles of clay-substance and of fluxes. The smaller the particles and the more uniform the mixture, the lower the refractoriness. Thus Hofman-Stoughton[4] have shown that a mixture of raw and burnt clay of the same general chemical character varying in size from coarse grains to an impalpable powder behaved in practically the same way when subjected as such to the action of heat and of fluxes as when the whole had been finely ground. On the other hand, Ries[5] proved in a striking way how the admixture of the same amount of flux added to a given quantity of fire-clay caused fusion to take place at a much lower temperature when the flux was 150- than when it was 80-mesh size. Weber's[6] examination of fire-brick lays special stress upon the size of grain. In fact, when a clay is heated gradually to complete fusion, at first fritting takes place, which hardens the mixture; then the components (fluxes) which can combine into a readily fusible mixture (eutectic) will become liquefied; the liquid part will gradually dissolve adjoining particles, at first the siliceous, then the aluminous, the fine particles more readily than the coarse, until finally the whole has been liquefied to a homogeneous solution. This explains also the softening of a clay before it fuses.

The only conclusive way of arriving at the fire-resisting power of a clay is by means of a fire test. The simplest is the direct method of Seger.[7] The clay

[1] Melts at 1400° C., G. Vogt, *Bull. Soc. Chim.*, 1890, IV, 343; *Chem. News*, 1890, LXII, 315.

[2] *Nineteenth Ann. Rep.*, U. S. Geol. Surv., 1897–98, VI, 393 (Ries).

[3] *Tr. A. I. M. E.*, 1898, XXVIII, 160; *Tr. Am. Ceram. Soc.*, 1907, IX, 772.

[4] *Tr. A. I. M. E.*, 1898, XXVIII, 440.

[5] *Tr. A. I. M. E.*, 1904, XXXIV, 205, 956 (Hofman).

[6] *Tr. A. I. M. E.*, 1905, XXXV, 637.

[7] Hofman, *Tr. A. I. M. E.*, 1895, XXV, 4.
Ludwig, *Thonind. Z.*, 1906, XXX, 1477

to be tested is molded into the form of a small Seger cone, placed with two or three Seger cones in a specially refractory crucible and heated in a Deville furnace until a Seger cone is found showing the same behavior in the fire as the clay that is being tested.

The resistance of a fire-clay to fluxes[1] may be tested by mixing samples of say 1.5 g. severally with 5, 10, 15 . . . mg. flux, forming the mixtures into small Seger cones, heating them in a Deville furnace with Seger cone No. 26 to such a temperature that cone No. 26 will melt. The sample which shows the same behavior as the Seger cone will be the critical mixture, and the percentage of flux it contains will form the criterion of the clay's resistance to corrosion.

PLASTICITY AND SHRINKAGE.—The two other leading properties of a fire-clay are plasticity and shrinkage.

PLASTICITY[2] is the property which a raw clay possesses of absorbing water and then forming, when worked, a pasty mass that can be molded, and that will retain its shape when dried and burnt. The degree of plasticity varies very much. Residual clays[3] (kaolin, rock- or flint-clays) are practically non-plastic (lean); sedimentary clays, on the contrary, are as a rule plastic and are termed fat. Sandy (gritty) clays are less plastic than those that are fine (gritless); grinding[4] increases the plasticity; the more plastic a clay the more valuable. The amount of water required to form a plastic mass varies with the fineness of particles; the coarser they are the less water is needed. Coarse or sandy clays require 14 to 20 per cent. water; medium-fine clays 20 to 25 per cent., very fine clays 25 to 30, and occasionally 35 per cent. to develop the full plasticity. What causes the plasticity of certain clays is not definitely known. Johnson and Blake[5] found under the microscope that plastic clays were made up mainly of prismatic crystals and fine fan-shaped and curved plates, while flint-clays contained few plates only, from which it would appear that fine plates were the cause. Olchewsky[6] suggested the interlocking of clay particles and kaolinite plates as the cause. Cushman[7] worked out a theory that colloids[8] (glue-like particles) of hydrated aluminum silicates are the main cause. Grout and Pappe[9] hold that molecular attraction of colloids acting together are the cause.

[1] Hofman, *Tr. A. I. M. E.*, 1898, XXVIII, 435.

[2] Seger, *Thonind. Z.*, 1890, XIV, 201.
Simonis, *Sprechsaal*, 1905, XXXVIII, 597, 881, 1625; 1906, XXXIX, 1167, 1184.
Chatenet, *Rev. Mét.*, 1907, IV, 937.

[3] Galpin, *Tr. Am. Ceram. Soc.*, 1912, XIV, 301.

[4] Page-Cushman, *Bull.* No. 85, U. S. Dept. Agriculture, 1904.
Cushman, *Circular* No. 38, Office of Public Roads, U. S. Dept. Agriculture.

[5] *Am. J. Sc. and Arts*, 1867, XLIII, 351.

[6] Deutsche Töpfer und Ziegler Z., 1880, 385.

[7] *Tr. Am. Cer. Soc.*, 1904, VI, 65; 1908, VIII, 180.

[8] Ashley, *Bull.*, 388, U. S. Geol. Surv., 1909; *Tr. Am. Ceram. Soc.*, 1909, XI, 530.
Keppeler, *Zt. Electrochem.*, 1909, XV, 781, 540 (Rohland).
Soper, *Min. Sc. Press*, 1909, XCIX, 626.
Editor, *Tonind. Z.*, 1909, XXXIII, 1524.
Bleininger-Fulton, *Tr. Am. Ceram. Soc.*, 1912, XIV, 827.

[9] *Tr. Am. Ceram. Soc.*, 1912, XIV, 71.

Different methods have been suggested to determine the degree of plasticity. Thus Bischof[1] obtained relative figures by kneading, *e.g.*, two samples of finely ground clay, with the same amount of water, forcing them through a cylindrical die and measuring the length of the extruded pencil before it broke. Stover-Lindley[2] used this method at Trenton, N. J. Jochum[3] forms a pencil 5/8 in. square by 5 7/8 in. long and bends it in a registering apparatus until it cracks. The binding power (see below) gives good approximations. It was once held that tensile strength could be used to express the degree of plasticity, but high tensile strength and plasticity do not always go together.

TENSILE STRENGTH[4] has its own importance in that it gives an expression for the amount of handling and the addition of non-plastic material a molded clay can stand. The method employed is similar to that of cement-testing: The clay is ground, molded into the standard form of cement briquette, air-dried 2 to 5 days and tested. With very fine clays air-drying has to be supplemented by steam-drying. Tensile-strength data cover a wide range: Kaolin 5 to 20 lb. per sq. in.; common brick clays 30 to 100; pottery clays 100 to 500; ball clays and other very plastic clays 200 to 500.

Tensile-strength tests are often used to determine the *Binding Power:*[5] Measured amounts of clay passed severally through a 40-mesh sieve are mixed dry with measured increasing amounts of sand ground through 40- and on 100-mesh, moistened, kneaded, molded into briquettes, air-dried and then broken in the testing machine. The briquettes with increasing amounts of sand will show about the same tensile strength up to a sudden drop. The binding power is closely related to plasticity.

SHRINKAGE.—When a clay, mixed with water to form a paste, is exposed to the drying influences of air, part of the mechanically combined water is evaporated, and the paste shrinks. The linear air-shrinkage ranges from 2 per cent. with sandy to 12 with fat clays, and averages 5 to 6 per cent. One part of the water absorbed by the clay fills the pore-spaces of the single clay-particles (pore-water), the other (film-water) the interstitial spaces between the clay-granules forming films or irregular capillary tubes. In drying, the amount of pore-water, ranging from 1 to 5 per cent., remains unchanged, while the film-water is conveyed by the capillary tubes from the interior to the surface where it evaporates; at the same time the granules of clay are drawn closer together and fill up the emptied spaces. The larger the particles, *i.e.*, the coarser-grained, the more quickly and uniformly will a clay dry, and the less liable will it be to crack.

[1] *Feuerfeste Thone*, 3d ed., p. 83.

[2] *Tr. Am. Cer. Soc.*, 1907, VII, 397.

[3] *Zt. Ver. deutsch. Ing.*, 1895, XXXIX, 317.

[4] Wheeler, *Geol. Surv. Mo.*, vol. XI, p. 111.
Beyer-William, *Ia. Geol. Surv.*, vol. XIV, p. 83.
Ries, *N. J. Geol. Surv.*, vol. VI, p. 85.
Orton, *Tr. Am. Cer. Soc.*, 1900, II, 100; 1901, III, 198.

[5] Stoermer, M., "Fehler bei der Thonwaaren-Fabrikation," Craz and Gerlach, Freiberg, 1901, p. 37.

With fine-grained clays the small capillary tubes, while large in number and therefore absorbing much water, cannot draw the water as quickly from the center as the large-size tubes; hence fine-grained clays can be dried only slowly and are likely to crack more or less on account of the greater contraction at the surface than at the center, especially if the drying is hastened. A clay that has been air-dried retains its plasticity.

If a clay is heated to 200–300° C.,[1] a decided alteration takes place with many clays in that they become granular and begin to lose their plasticity; with others the temperature for loss of plasticity lies at about 450°; again dehydration is not always accompanied by loss of plasticity. Generally, however, dehydration means loss of plasticity; the clay shrinks, and shrinkage increases with the temperature and with it the hardness of the clay. When the chemically combined water is expelled, compounds are formed[2] which have a lower specific gravity than the raw clay; at 950° C., however, there is a sudden increase in specific gravity, caused by some exothermic reaction (Le Chatelier) in the substance.

The fire-shrinkage of Missouri clays averages 5.7 per cent. Coarse- and fine-grained clays will show a similar behavior in fire- as in air-shrinkage. In the fire a clay loses its plasticity, but still remains porous and readily again absorbs water. In order to make a fine-grained plastic clay shrink evenly without cracking, coarse non-shrinking substances (grog),[3] such as sandy clays, sand, burnt clay, graphite are added. The amount of grog to be added to a clay varies with its shrinkage and the size of the piece that is to be molded. Lean clays require 10 to 15 per cent. grog, fat clays 25 to 35 per cent.; with large pieces the amount may reach 50 per cent., with heavy furnace blocks even 65 per cent. Grog acts in part as a diluent reducing the amount of clay-substance; in part it increases the interstitial spaces thus enabling the capillary tubes to convey readily the water from the interior to the surface. The porosity of burnt clay gives fire-brick a certain elasticity which enables it to bear changes of temperature without cracking.[4]

Linear shrinkage, expressed in per cent. of original length of test-piece, is determined by molding clay with water into the form of a brick, say $8 \times 2 \times 0.5$ in., placing it on a weighed glass plate, drawing a fine center line, say 6 in. long, and a short one at either end |———|, weighing the sample, and then slowly drying, first in the open air, then in a steam-chest (130° C.) to constant weight and noting the reduction in length of the central line. The fire-shrinkage is determined by heating the air-dried brick in a muffle to 600 or 700° C., or a higher temperature until the maximum shrinkage has taken place. Wheeler,[5]

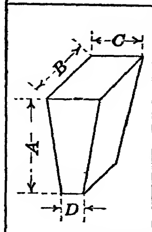
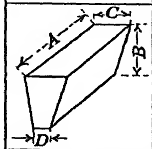
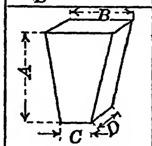
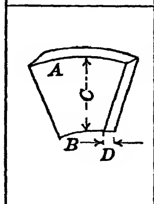
[1] Bleininger, Bureau of Standards, *Bull.* 152, 1911; *Tr. Am. Ceram. Soc.*, 1910, XII, 504.
Brown-Montgomery, *op. cit.*, 1912, XIV, 709.
Hursh, *ibid.*, p. 811.
Glasenapp, *Tonind. Z.*, 1907, XXXI, 1167; *Rev. Mét. Extr.*, 1909, VI, 731.

[2] Knote, *Tr. Am. Ceram. Soc.*, 1910, XII, 227.

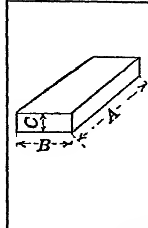
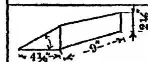
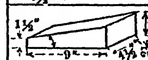
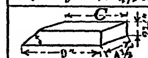
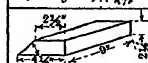
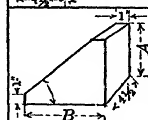
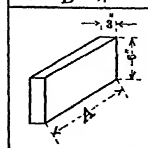
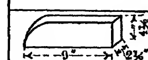
[3] Rigg, *Met. Chem. Eng.*, 1910, VIII, 523.

[4] Windszus, *Tond.*, 1912, XXXVI[2], 1205, 1625, 1656; *Met. Chem. Eng.*, 1912, X, 662.

[5] *Geol. Surv. of Mo.*, vol. XI, p. 123.

Sketch	Dimensions A	B	C	D	Name	Included Angle	No. of Brick to the Circle	Inside Circle
	9"	4½"	2½"	1¾"	No.1 Wedge	3° 32′	102	
	9"	4½"	2½"	1½"	No.2 Wedge	6° 21′	56	
	9"	4½"	2½"	2"	No.3 Wedge or Bullhead	3° 10′	113	
	9"	6¾"	2½"	1½"	No.2A Wedge	6° 21′	56	
	9"	6¾"	2½"	2"	No.3A Wedge	3° 10′	113	
	9"	13½"	2½"	1½"	No.2 Wedge Tile	6° 21′	56	
	9"	13½"	2½"	2"	No.3 Wedge Tile	3° 10′	113	
	9"	13½"	3½"	2¾"	13½" Arch			
	9"	4½"	2½"	2¼"	No.1 Side Arch	5° 0′	72	
	9"	4½"	2½"	2"	No.1A Side Arch	6° 21′	56	
	9"	4½"	2½"	1¾"	No.2 Side Arch	8° 23′	42	
	9"	4½"	2½"	1¼"	No.2A Side Arch	12° 40′	28	
	9"	4½"	2½"	1"	No.3 Side Arch	18°	20	
	9"	4½"	4"	2½"	No.1 Key	3° 10′	113	
	9"	4½"	3½"	2½"	No.2 Key	5° 43′	63	
	9"	4½"	3"	2½"	No.3 Key	9° 43′	38	
	9"	4½"	2¼"	2½"	No.4 Key	14° 24′	25	
	9"	6"	5⅜"	2½"	9 x 6 Key			
	9"	6½"	4½"	2½"	No.1 Circle	30°	12	33"
	9"	7 3/16"	4½"	2½"	No.2 Circle	25° 24′	14	45"
	9"	7⅜"	4½"	2½"	No.3 Circle	20°	18	51"
	9"	6 1/16"	6"	3"	No.1 Cupola Brick	24°	15	30"
	9"	6¾"	6"	3"	No.2 Cupola Brick	21° 10′	17	36"
	9"	7 3/16"	6"	3"	No.3 Cupola Brick	17° 10′	21	48"
	9"	7½"	6"	3"	No.4 Cupola Brick	14° 24′	25	60"

1-In all Tapers
2-For Regenerative Furnaces
3-All Sizes
4-For Inclined Wells and Skewbacks, etc.
5-Same as Feather Edge
6-Used for Supporting Checker Work of Regenerative Furnaces
7-Used around Door Openings

Sketch	Dimensions A	B	C	Name	Included Angle
	9"	4½"	2½"	9' Straight	
	9"	4½"	1"	1" Split	
	9"	4½"	1¼"	1¼" Split	
	9"	4½"	1½"	1½" Split	
	9"	4½"	2"	2" Split	
	9"	2¾"	2½"	Soap	
	9"	2¾"	2¾"	Checker Brick[2]	
				Blocks and Straights[3]	
				Feather Edge[4]	29° 3′
				Edge Skew[5] or	18° 26′
				No.3 Skew[5]	
			6⅛"	No.1 Skew[5]	41°
			7"	Bevel End[5]	51° 20′
				Bevel Edge[5]	48° 20′
	7"	7"		7"x 7" Skew Back	45°
	7"	9"		7"x 9"	36° 25′
	9"	9"		9"x 9"	45°
	9"	12"		9"x 12"	36° 1½′
				18" Regenerator Tile[6]	
				21"	
				22"	
				24"	
				27"	
				Jamb[7]	

FIGS. 230 and 231.—Leading shapes of fire-brick.

in determining the shrinkage of Missouri clays, made bricks 0.5×4×4 in., 1×1×4 in., 2×2×4 in. and 3×4×8 in., so as to have approximately the sizes of ordinary ware. The cubical shrinkage, expressed in per cent. of original volume, may be determined with a Seger volumeter.[1] Experiments upon the specific heat of clays have been made by Knote.[2] Thus the specific heat of raw clay from Olive Hill, Ky., was found to be 0.237; burnt at 650° C. it decreased to 0.204, at 1,050° to 0.200.

159. Fire-brick Manufacture, Properties and Laying.[3]—The fire-brick used in furnace construction have various forms; most of them have become standardized; many, however, have special shapes as, *e.g.*, the brick used in retort coking ovens, zinc smelting furnaces, various recuperators and regenerators. Figs. 230 and 231[4] give the leading shapes with the common dimensions. The same reference contains the common rules for calculating the number of brick required for the construction of arches, etc. In the manufacture of fire-brick five steps have to be considered: the composition, preparation and molding of the mixture, and the drying and burning of the brick.

Composition of Mixture.—The mixtures ordinarily used for making fire-brick consist of refractory raw and burnt flint-clay and raw refractory plastic clay. The burnt flint-clay is often replaced in part by ground fire-brick and sometimes by coarse quartz. The flint-clay which is abundant and cheap constitutes the largest part of the mixture. As it shrinks considerably upon drying and burning, and thus has a tendency to crack and lose its shape, part of it is always calcined.

The proportion of flint-clay to be deprived of its shrinking power will depend upon the amount of raw plastic clay that is required to form the bond and upon its shrinkage, the principle to be followed being that for the soundness and strength of the ensuing brick, it is essential that the components of the mixture shall approximately show the same amount of shrinkage. Plastic clays suited to serve as a bonding material are often sandy and have a low shrinkage. As the flint-clay furnishes the fire-resisting power and the plastic clay the physical strength to the brick, it will be seen that it is not possible to make a brick that shall combine to perfection the two properties; one has to be sacrificed to the other according to the use to which the brick is to be put.

A mixture for fire-brick that is to resist intense heat consists of about 45 per cent. vol. of raw flint-clay, 45 per cent. calcined flint-clay and 10 per cent. plastic clay. Such a brick is loose and porous, even when hard-burnt. Replacing part of the calcined clay by quartzite weakens the brick physically and

[1] *Thonind. Z.*, 1881, v, 2; 1891, xv, 317; *Ia. Geol. Survey*, vol. xiv, p. 107.
Crossley, *Clay-worker*, 1899, xxii, 115, 199, 265, 367, 449; 1900, xxxiii, 22, 428, 520.

[2] *Tr. Am. Ceram. Soc.*, 1912, xiv, 394.

[3] Wear of Fire-brick in Metallurgical Furnaces : *Thonind. Z.*, 1895, xix, 380 (Dürre); 1903, xxvii, 335 (Jochum), 336 and 775 (Osann); 1906, xxx, 145; 1907, xxxi, 1382 (Osann); *Stahl u. Eisen*, 1892, 265; 1898, xviii, 168 (Lürmann); 1903, xxiii, 826; 1907, xxvii, 1626, (Osann); 1905, xxv, 870 (Ludwig); 1906, xxvi, 336 (Osann); 1912, xxxii, 231 (Kinder); *Tr. A. I. M. E.*, 1904, xxxiv, 427 (Firmstone).

[4] Peters, *Iron Age*, 1908, lxxxii, 1074; *Eng. Min. J.*, 1909, lxxxvii, 447.

to a certain extent reduces the refractoriness. A brick that is to be physically strong will be made of a mixture of about 50 per cent. vol. raw and calcined flint-clay and 50 per cent. plastic clay.

Preparation of Mixture.[1]—The clay in a bank is, as a rule, not uniform.[2] It has therefore to be first sorted into different grades and perhaps cobbed to remove impurities, such as quartz, feldspar, mica, pyrite, oxide of iron, etc. Sometimes clays are washed,[3] especially when they are mined in open cuts and are thus liable to become contaminated with mud; or, the clays may be too sandy or contain other impurities which have to be removed. Log-washers,

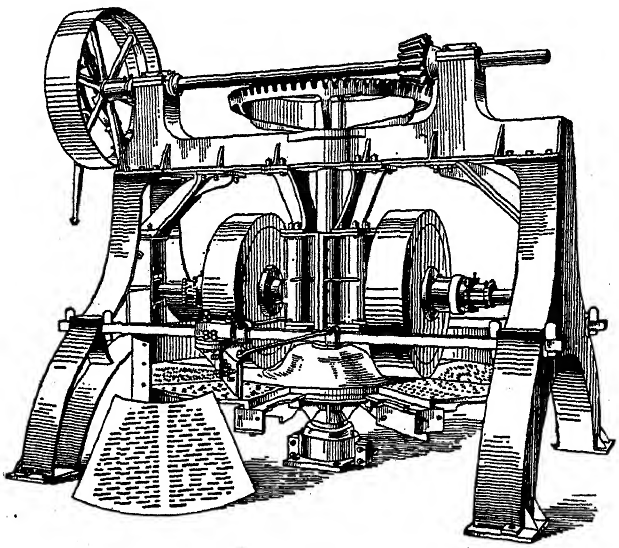


Fig. 232.—Dry-pan.

pit-washers or some other simple disintegrating and stirring devices separate the clay-particles satisfactorily from the rest. They are recovered from the water by settling or filter-pressing. Often clay is weathered, as spreading it out and exposing for months to the weather causes the texture to become more even and the toughness and plasticity to increase.

Ordinarily the operations of preparing a mixture consist in crushing and grinding the components and then tempering the mixture that it may be ready for molding. The clay is ground in a dry- or wet-pan after it has been crushed to a limiting size by breakers, rolls, disintegrators; it is then tempered in a pug-mill or a wet-pan; before pugging, it may have to be soaked in a pit.

[1] Dümmler, K., "Handbuch der Ziegel-fabrikation," Knapp, Halle, 1913.

[2] Soper, "Geology and Mining of Clay," *Eng. Min. J.*, 1912, XCIII, 263.

[3] Ries, *Nineteenth Ann. Rep.*, U. S. Geol. Surv., 1897-98, VI, 387.

The Slumming Process, *Clay Worker*, 1904, XLII, 244.

Bleininger, *op. cit.*, 1909, LII, 142, 269; *Tr. Am. Ceram. Soc.*, 1909, XI, 467 (Parmalee-Moore); 1912, XIV, 399 (Thomas).

Bollenbach, *Tonind. Z.*, 1911, XXXV, 1780.

The dry-pan, Fig. 232, is a horizontal cast-iron pan, 7 to 9 ft. in diam. pivoted to a vertical revolving shaft geared either from above or below. It has two iron edge-rollers (6 to 14 in. face, 48 in. diam, each 2,000 to 5,000 lb.) that are keyed to a horizontal shaft which rests in grooves of the framework. The rollers usually have removable chilled tires. The floor of the pan outside of the path of the rollers is made up of exchangeable sectional plates perforated with holes 1/8×3/16 in. The rollers, revolved by the friction of the rotating pan, crush the clay fed near the center. From the rollers the clay passes outward to the periphery and falls through the perforations into a wooden box underneath the pan (not shown) whence it is delivered by scrapers, attached to the bottom of the pan, to a discharge-chute. The clay which did not pass through the pan-bottom is returned by scrapers to the path of the rollers. An average record of a 9-ft. dry-pan with 1/8×3/16 in. holes making 25 to 30 r.p.m. and requiring 20 to 30 h.p. is 100 tons rock-clay in 10 hr. The ground clay may or may not be screened. Screening insures greater uniformity. From the ground materials the mixture is made up by bedding, *i.e.*, the components measured by volume are spread out in horizontal layers, one on top of the other, and removed in vertical sections. A bed is usually made up on the floor, watered and then fed into a pug-mill; sometimes it is prepared in a pit, soaked with water and allowed to stand for 24 hr. before it is pugged.

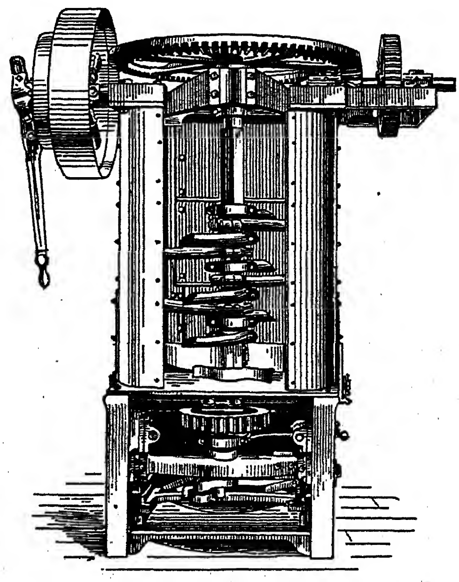


FIG. 233.—Vertical pug-mill.

A pug-mill is a vertical (Fig. 233) or horizontal (Fig. 234) iron casing in which revolves a shaft equipped with knives (cutting bars, paddles, propellers) set so as to form an interrupted Archimedian screw. The clay-mixture fed at the top or the upper end and wetted with the necessary amount of water is cut up, mixed, and thoroughly kneaded in its passage through the casing until it is delivered at the lower end. Vertical pug-mills have a discharge at the side; horizontal mills which discharge at the end may be open or closed, thus delivering the mixture either free or under pressure. In the closed-delivery pug-mill, shown in Fig. 234, the mixture is fed to a screw in the conical end which forces it through the die. The resulting rectangular bar is received on a cutting-table and cut automatically into bricks. The vertical pug-mill, shown in Fig. 233, weighs 6,000 lb., occupies a floor-space 3 1/2×6 1/2 ft. and is 9 ft. high; it has eight knives and two wipers, and treats 3 to 4 tons of material in 10 hr. The horizontal mill, shown in Fig. 234, weighs 20,000 lb., is 52 ft. long and 8 ft. wide

at the upper end; it has 16 knives, makes 22 r.p.m., and treats 6 to 7 tons of material in 10 hr.

A wet-pan resembles very much a dry-pan; the bottom is not perforated; the rollers are narrower and lighter (8 to 12 in. face, 24 to 36 in. diam., 2,000 to 3,000 lb., pan 9 ft. diam.); grinding and tempering go on simultaneously. A charge of mixture (1,500 to 4,000 lb.) is dumped into the revolving pan and

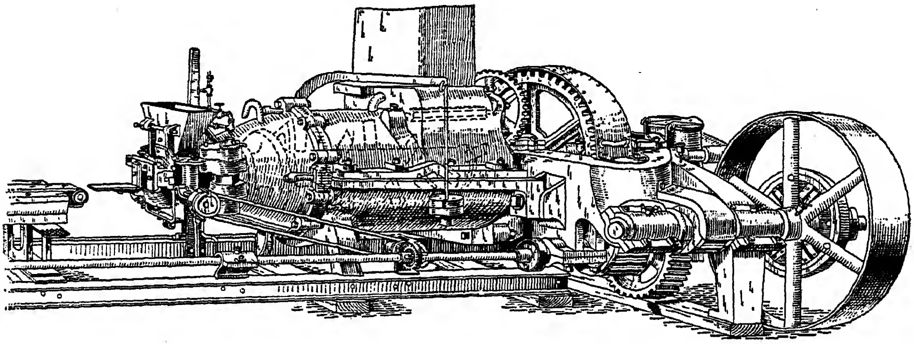


FIG. 234.—Horizontal closed-delivery pug-mill.

water added as often as necessary; after 10 to 15 min. it is sufficiently worked and taken out by means of a long-handled wooden paddle pivoted on a ring-bolt near the blade. The paddle is lowered into the pan, the mixture glides onto it and is discharged. The wet-pan is sometimes used for tempering materials ground in the dry-pan. The dry-pan has the advantage that its material is crushed to a limiting size, the wet-pan that it develops plasticity.

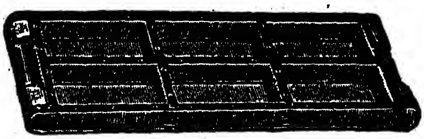


FIG. 235.—Wooden mold.

MOLDING.—Bricks are molded either by hand or by machine. In machine molding there are distinguished the soft-mud, the stiff-mud, and the dry-processes.

The best fire-brick are *hand-molded*, for which purposes the temper has to be especially soft. A ball of clay holding more material than will fill a wooden mold (Fig. 235) is thrown forcibly into the mold after the latter has been "sanded" with burnt clay, the excess of mixture is cut off by means of a wire, the surface smoothed, the mold dumped and the brick transferred to a drying floor, when it is re-pressed in a movable hand-press[1] after having dried for about 6 hr. Re-pressing is necessary to give the brick the required shape, to smooth the sides and to strengthen it. A man will make 4,000 bricks in a day, supposing the mixture and sanded molds to be delivered and the filled molds removed.

SOFT-MUD MACHINES are sometimes used for making fire-brick. A machine[2] consists of a vertical pug-mill which repugs the mixture, delivers it at the bottom and forces it into sanded molds fed in empty on one side and pushed out auto-

[1] Illustration, *Min. Ind.*, 1895, IV, 119.
[2] *Op. cit.*

matically at the other. The bricks are rough and unfinished and have to be re-pressed. The capacity of a machine is 25,000 to 30,000 brick per day.

THE STIFF-MUD MACHINE principle has been shown in connection with the horizontal pug-mill, Fig. 234. It is rarely used in connection with fire-brick manufacture. Its capacity is 60,000 to 70,000 brick per day. It is employed for making flue-dust briquettes at Anaconda.[1]

THE DRY-PROCESS is hardly used at all. Here the raw material, ground rather fine, is pressed without moistening by a powerful machine into the form of a brick and burned. When the natural moisture is not sufficient to bind the particles together, it may have to be steamed. The natural bond of such a brick must be weak; it can become strong only through the presence of fusible silicates, but these must be absent in a fire-brick.

DRYING.[2]—Fire-brick are dried on drying floors, rarely on pallets, sometimes in tunnel-dryers. Drying floors are brick floors heated by flues passing underneath. Pallet-dryers are covered racks or cribs for holding boards (pallets) 3 or 4 in. apart, each with about six brick. The time required varies. Pallets have the advantage of cheapness, capacity and economy of space, but suffer from being dependent upon the weather. Tunnel-dryers consist of a set of brick flues, say $4 \times 5 \times 100$ ft., heated by fireplaces, steam-pipes or superheated air. The green brick are placed on trucks, a train of which moves on tracks slowly through a tunnel in 24 to 36 hr.

BURNING.—The dried fire-brick are burned in brick kilns.[3] According to the method of firing they are classed as down-draft, up-draft and continuous kilns. The down-draft kiln[4] is in common use in burning fire-brick. It is a circular or oblong brick chamber 20 to 30 ft. in diam., or 20×30 ft. long, and 15 ft. high which has a number of fireplaces at the periphery, or at the sides, from which the flames, upon entering the kiln, rise in vertical flues reaching to near the spring of the arch (usually of silica brick), then spread over the tops of the bricks, descend in the open space left between them and through the perforated bottom, on which the bricks rest, into underground flues connected with the main flue and the chimney. The kiln shown in Figs. 236 to 238 is used for baking electricial porcelain and therefore too small for brick; for burning brick it would have to be 40 feet in diameter instead of 18, and the other dimensions correspondingly larger. The temperature is kept at Seger cone Nos. 9 to 12 (1,280 to 1,350° C.); it takes 5 to 6 days to burn the charge and a few more days to cool it. With up-draft kilns,[5] the flame enters below the perforated bottom; the products of combustion enter the chamber through the perforations and pass off through a pipe in the roof.

[1] Austin, *Tr. A. I. M. E.*, 1906, XXXVII, 460.

[2] Lines, *Tr. Am. Ceram. Soc.*, 1908, X, 146.
Farnham, *op. cit.*, 1910, XII, 392.

[3] Glasenapp: Chemical Phenomena in Burning, *Thonind. Z.*, 1907, XXXI, 1167.
Mellor-Austin, *op. cit.*, 1907, XXXI, 904; 1908, XXXII, 1123.

[4] Montgomery, *Tr. Am. Ceram. Soc.*, 1911, XIII, 311.
Grave, *op. cit.*, 1912, XIV, 113.

[5] Drawing: Riddle, *Trans. Am. Ceram. Soc.*, 1911, XIII, 385.

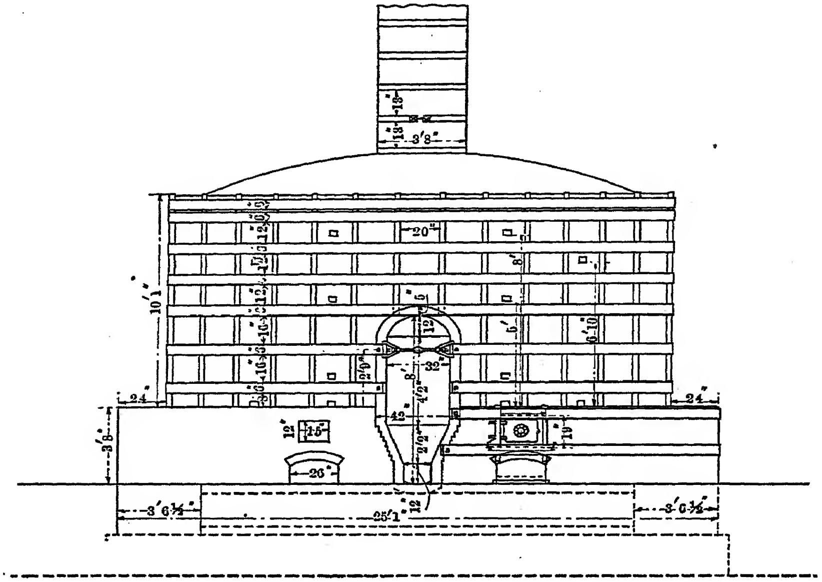


FIG. 236.—Down-draft kiln with central stack, elevation.

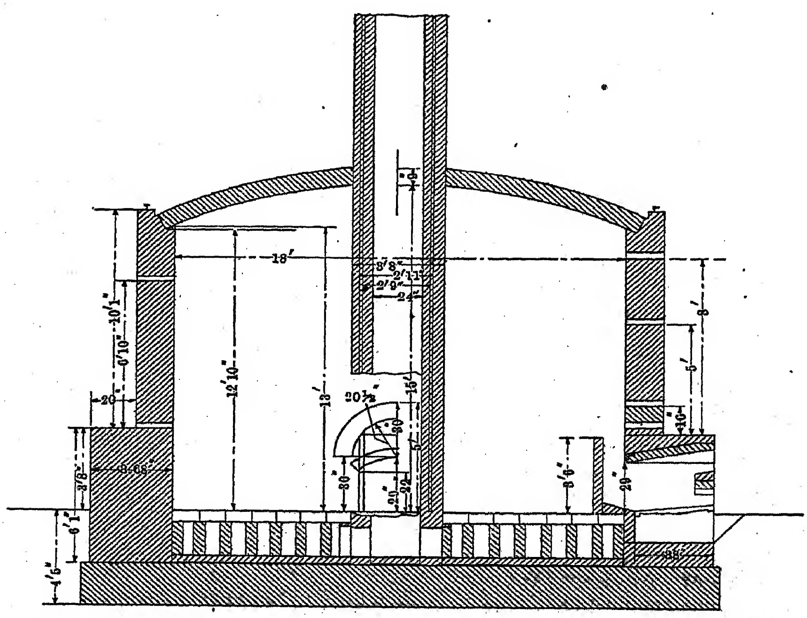


FIG. 237.—Down-draft kiln with central stack, vertical section.

In continuous kilns[1] which form a series of connected chambers, the waste heat from one chamber which, *e.g.*, is at a bright-red, serves to partially burn and to dry the green bricks of the others.

PROPERTIES OF FIRE-BRICK.[2]—The color is usually a light-gray; (this cannot serve as a guide to refractoriness;)[3] the specific gravity 1.8 to 2; (this increases with the rise of temperature;)[4] the hardness 6.5 to 7.0, (Rattler-tests;)[5] it absorbs

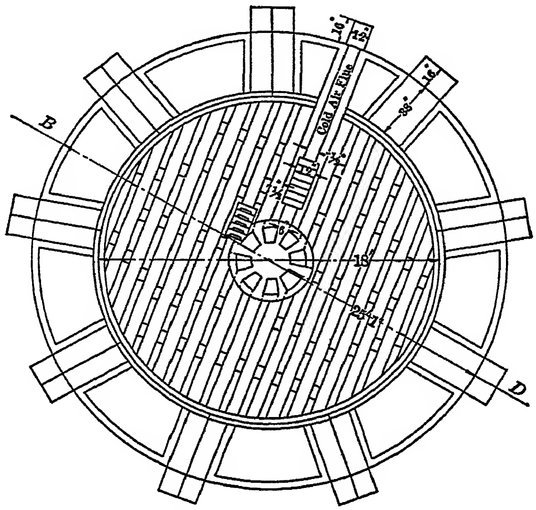


FIG. 238.—Down-draft kiln with central stack, horizontal section.

11 to 14 per cent. water when soaked for 48 hr. The absorbing power is about inversely proportional to the crushing strength.[6] The crushing strength ranges from 1,600 to 2,000 lb. per square inch;[7] it increases in the furnace with some

[1] Egleston, *Tr. A. I. M. E.*, 1886–87, xv, 448 (Dunnachie); *School Min. Quart.*, 1886–87, VIII, 34 (Mendenheim); *Clay Worker*, 1908, L, 492 (Duemmler).
Richardson, *Tr. Am. Ceram. Soc.* 1912, XIV, 778 (producer gas).
Lange, *Stahl u. Eisen*, 1912, XXXII, 1729 (Mendenheim).

[2] Rigg, Defects of Fire-brick, *Met. Chem. Eng.*, 1910, VIII, 237.

[3] Leander, *Tonind. Z.*, 1912, XXXVI, 1725.

[4] Ogan, *Tr. Am. Ceram. Soc.*, 1911, XIII, 610.

[5] *Trans. Am. Ceram. Soc.*, 1909, XI, 501 (Hope); 1911, XIII, 792; 1912, XIV, 180, 228 (Orton); 1912, XIV, 270 (Brown).

[6] Bleininger, *Tr. Am. Ceram. Soc.*, 1910, XII, 564.
Tests of Wolgodine, *Rev. Mét.*, 1909, VI, 767; *Tonind. Z.*, 1910, XXXIV, 84; *Met. Chem. Eng.*, 1909, VII, 383, 433.
Douty-Beebe, *Proc. Am. Soc. Test. Mat.*, 1911, XI, 767.

[7] Lürmann, *Stahl u. Eisen*, 1901, XXI, 786; 1907, XXVII, 1423, 1662.
Wernicke, *op. cit.*, 1907, XXVII, 1661.
Seger-Cramer, *Thonind. Z.*, 1907, XXXI, 1383; 1908, XXXII, 69; *op. cit.*, 1908, XXXVIII, 414.
Bleininger, *Tr. Am. Ceram. Soc.*, 1910, XII, 337; 1911, XIII, 210; *Tonind. Z.*, 1912, XXXVI, 815.
Gary, *op. cit.*, 1910, XXXIV, 633.

brands, and decreases with others;[1] the load-carrying power[2] is very much diminished when heated to say 1,500° C. Cramer[3] devised a method for determining the softening temperature by molding pencils 3/8×3/4×10 in., baking them at Seger cones Nos. 16–17, placing the rods horizontally on sharp-edge brick supports 8 in. apart, and then heating to bending and measuring the height of the segment. Fire-brick are poor conductors of heat;[4] with Au as 1,000,[5] the heat conductivity is 12. Table 144 by Wolgodine[6] gives the heat conductivities of some of the leading refractory materials. Other tests are those by Fitzgerald.[7] Stansfield-McLeod-McMahon[8] have studied the electric restivity of specimens of fire-brick, silica, magnesite and chrome brick. The shrinkage, and expansion, vary greatly with the porosity and percentage of free SiO_2. Lürmann[9] found the range of linear shrinkage of leading European brands to be from 0.54 to 2.52 per cent. Ogan[10] tested the shrinkage at different temperatures. Foote-Stockman[11] found that some of the leading American brands expanded about 0.5 per cent. A fire-brick with 40 per cent. Al_2O_3 satisfies the most rigid demands of the iron blast-furnace.[12] A brick to be acid-proof must be made of a dense-burning mixture, free from Fe_xO_y and CaO, as porous brick never resist acid any length of time.

LAYING OF FIRE-BRICK.—In laying fire-brick, it is essential to make as thin a joint as possible. The mortar used is a thin gruel-like mixture of finely ground raw and burnt clay in proportions similar to those of which the brick was made. The brick which are to be adjacent are rubbed against one another until no light can be seen through the joint. In building flues of by-product coke-ovens the brick are sometimes ground with carborundum wheels to certain

[1] Seger-Cramer, *Stahl u. Eisen*, 1907, XXVII, 521.

[2] Parker, *Tr. Am. Ceram. Soc.*, 1907, VII, 185.

[3] *Thonind. Z.*, 1901, XXV[1], 706; 1902, XXVI, 300, 1065; also Stoermer, *op. cit.*, 1901, XXV[2], 1628, 1699.

Bleininger, *Tr. Am. Ceram. Soc.*, 1910, XII, 237.

[4] Pennock, *Tr. A. I. M. E.*, 1896, XXVI, 263.

Wheeler, *Tr. Am. Ceram. Soc.*, 1904, VI, 119.

Page, *op. cit.*, 1905, VII, 434.

Römer, *Stahl u. Eisen*, 1908, XXVIII, 883.

Lürmann, *ibid.*, 1180.

Cramer, *Tonind. Z.*, 1908, XXXII, 28.

Hering, *Met. Chem. Eng.*, 1908, VI, 495; 1909, XII, 72.

Osann, *Tonind. Z.*, 1909, XXXIII, 1405; *Stahl u. Eisen*, 1910, XXXIX, 1060, 1107, 1147, 1692.

Clement-Egy, *Bull.* 36, Univ. Ill., 1909.

[5] Kerl-Cramer-Hecht, *op. cit.*, p. 475.

[6] Internat. Congress Appl. Chem., London, 1909; *Rev. Mét.*, 1909, VI, 767; *Tonind. Z.*, 1910, XXXIV, 84; *Met. Chem. Eng.*, 1909, VII, 383, 433.

[7] *Met. Chem. Eng.*, 1912, X, 286.

Randolph, *ibid.*, 287.

[8] *Met. Chem. Eng.*, 1912, X, 746.

[9] *Stahl u. Eisen*, 1901, XXI, 786.

[10] *Tr. Am. Ceram. Soc.*, 1911, XIII, 602; *Tonind. Z.*, 1912, XXXVI, 775.

[11] *Thesis* No. 226, 1902, *Mining Department*, Mass. Institute of Technology.

[12] Jochum, *Thonind. Z.*, 1901, XXI., 789.

TABLE 144.—HEAT CONDUCTIVITIES OF REFRACTORY MATERIALS

Material	Heat conductivity		
	g. cal. sec. p. sq. cm. p. cm. p. 1° C. diff.	Kg. cal. hr. p. sq. m. per m. p. 1° C. diff.	Relative per cent.
Graphite brick	0.025	9.0	100.0
Carborundum brick	0.0231	8.32	92.4
Magnesia brick	0.0071	2.54	28.4
Chromite brick	0.0057	2.05	22.8
Fire-brick	0.0042	1.50	16.7
Checker brick	0.0039	1.42	15.8
Gas-retort brick	0.0038	1.36	15.2
Building brick	0.0035	1.26	14.0
Bauxite brick	0.0033	1.19	13.2
Glass pot	0.0027	0.96	12.4
Terra cotta	0.0023	0.84	9.3
Silica brick	0.0020	0.71	7.8
Kieselguhr brick	0.0018	0.64	7.1

standards in order that there may be no leakage of gas through the joints. The brick to be laid is dipped into water, then into the clay gruel, put in position and hammered into place with a wooden mallet in order to squeeze out any excess mortar and make a close fit. After the brick have been laid, they are grouted with a slurry of the mortar to fill every crevice. The suggestion of Lürmann[1] to use, for an iron blast-furnace, a mortar less refractory than the brick with the idea that the fusing mortar would cement the brick and thus form a monolithic wall has not found favor, as, while it seemed to work in water-cooled brick-work, the mortar began to ooze out from walls subjected continuously to high temperatures.[2]

160. Fire-clay Used in Other than Brick Form.—Fire-clay is used in the form of a loose mass in ramming the bottom of a furnace. The ground raw clay is mixed with ground calcined clay or brick in proportions similar to those in brick-making, moistened and worked to form a mass that will cohere to a lump when pressed in the hand, but not adhere to the hand.[3] Sometimes ground raw fire-clay is mixed with equal volumes of charcoal or coke to form "*brasque;*" charcoal-brasque is weaker than coke-brasque. Brasque is a poor conductor of heat and is not readily attacked by heat or slag, but its use presupposes a reducing fusion; it serves for lining furnace bottoms, less so at present than in former years.

[1] *Thonind. Z.*, 1882, VI, 447.
[2] *Op. cit.*, 1884, VIII, 113.
[3] Clay Crucibles, Greaves-Walker, *Tr. Am. Ceram. Soc.*, 1910, XII, 5.

TABLE 145.—ANALYSIS OF CLAY FIRE-BRICK

	New Jersey	Anaconda, Mont.	Evans and Howard, St. Louis, Mo.	Rock Spar, Pa.	La Clede, St. Louis, Mo.	Harbison-Walker, Pittsburgh, Pa.	Stourbridge, England	Gemboig, England	German		
									Iron blast-furnace	Iron blast-furnace	Coking oven
SiO_2	78.24	72.8	63.3	60.11	60.9	50.4	69.50	54.04	59.79	69.54	84.80
Al_2O_3	19.24	23.7	33.6	29.60	33.6	42.8	27.52	38.81	25.50	25.34	11.26
Fe_2O_3	0.68	3.0	1.9	3.94	4.6	3.7	1.90	3.07	3.45	1.20	0.89
CaO	......	......	0.7	0.18	1.6	1.0	0.32	0.34	0.56	0.50	0.35
MgO	......	......	......	0.86	......	......	0.10	0.32	0.36	0.54	0.31
TiO_2	1.66	......	......	1.86	......	......	......	......	......	......	......
K_2O	......	......	......	3.21	......	......	......	......	10.34	2.68	2.39
Na_2O	......	......	......	0.36	......	......	......	......	......	......	......
Reference	(1)	(2)	(2)	(5)	(2)	(2)	(3)	(3)	(4)	(4)	(4)

(1) Geol. Surv. of N. J., "Clay Industry," 1904, p. 333.
(2) Austin, *Tr. A. I. M. E.*, 1906, XXXVII, 485.
(3) Bischof, "Collected Analyses," p. 135.
(4) Blasberg, *Stahl u. Eisen*, 1910, XXX, 1055.
(5) Private notes.

Other analyses: Geol. Surv. of N. J.; Kerl-Cramer-Hecht, *op. cit.*, p. 888; Bleininger, *Tr. Am. Ceram. Soc.*, 1911, XIII, 213; "Yorkshire Steel Crucibles," *Tonind. Z.*, 1908, XXXII, 498.

161. Silica[1] and Silica Brick.—Silica melts approximately at 1,750° C.[2] Heraeus[3] found that quartz decrepitated at 570° C., softened at 1,500°, became glassy at 1,700°, could be maintained in the fluid state at 2,000° C., and was somewhat volatile at 2,300 to 2,400° C. Cramer[4] volatilized SiO_2 completely in a Deville furnace[5] as did Moissan[6] and Hirsch[7] in an electric furnace. Silica expands upon heating. The results of Le Chatelier's[8] experiments are represented graphically in Fig. 239. Challendar[9] compared the expansion of SiO_2 with that

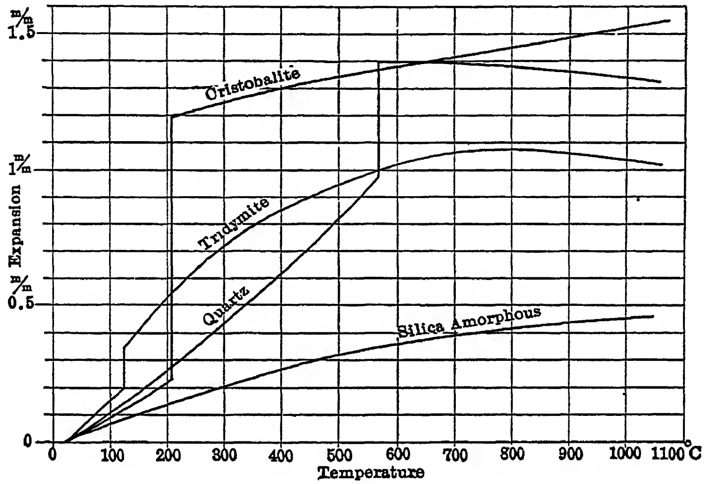


FIG. 239.—Expansion of silica by heat (Le Chatelier).

of Pt. Cramer[10] found that the expansion was progressive, *i.e.*, when sand was heated (SiO_2 98.25, Al_2O_3 1.04, Fe_2O_3 0.04, K_2O 0.40) ten times to the temperature for baking porcelain, the specific gravity diminished with every heat, decreasing from 2.662 to 2.398. The behavior of quartzite[11] is not as regular as that of SiO_2; some varieties swell considerably in the first heating and less so in the second; with others, just the reverse is true.[12] The expansion of sandstone and quartz-schist is more even than that of quartzite.[13]

[1] Le Chatelier; "La Silice," *Rev. Un. Min.* 1913, I, 85; forms part of "La Silice et les Silicates" to be issued by Hermann, Paris.

[2] Kanolt, *Technolog. Paper* 10, U. S. Bureau of Standards, 1912.

[3] *Zt. Electrochem.*, 1903, IX, 848.

[4] *Thonind. Z.*, 1892, XVI, 747.

[5] Hofman, *Tr. A. I. M. E.*, 1895, XXV, 5.

[6] *Compt. Rend.*, 1893, CXVI, 1222.

[7] *Stahl u. Eisen*, 1908, XXVIII, 413.

[8] *Compt. Rend.*, 1890, III, 123.

[9] *Am. Mfr.*, 1904, LXXIV, 469.

[10] *Thonind. Z.*, 1886, X, 355.

[11] Cramer, *Thonind. Z.*, 1901, XXV, 864; *Stahl u. Eisen*, 1901, XXI, 773.
Lattermann, *Thonind. Z.*, 1901, XXV, 659.

[12] Wernicke-Wildberg, *Tonind. Z.*, 1910, XXXIV, 262, 688, 723, 769.

[13] Cramer, *Thonind, Z.*, 1901, XXV, 864; *Stahl u. Eisen*, 1901, XXI, 773.

The research of Grum-Grzimailo[1] has shown that quartz (sp. gr. 2.65) heated for some time to 1,000° C. is changed into tridymite (sp. gr. 2.32) with an increase in volume of 14.2 per cent.; quartzite is less affected than quartz. Silica brick, which upon continued heating increases in volume from 17 to 20 per cent., consists of quartz, quartzite and a bond (a silicate of $CaO.MgO.FeO.AlO_3$). The more perfectly the quartz has been converted to tridymite in baking the better and stronger will be the silica brick; large pieces of quartz require a prolonged heating to be changed into tridymite; hence a rock with small grains of quartz is preferable to one with large, and an impure quartzite furnishing a larger number of points of attack for the conversion of the quartz than a pure milky quartzite, will be a more satisfactory raw material. Other investigations bearing upon the conversion and expansion are those of Holmquist,[2] Endell,[3] who also give cross-references, Kaye,[4] and Shepherd-Rankin.[5]

Silica coming in contact with bases at elevated temperatures forms more or less fusible slags.

Quartzite (Middlesbrough: SiO_2 95.40, Al_2O_3 3.10, CaO 1.08),[6] sandstone, ganister (Sheffield: SiO_2 98.94, Al_2O_3 0.57, Fe_2O_3 0.67, CaO 0.62, MgO 0.21, H_2O 0.42),[7] etc., were formerly used quite extensively in the natural state in furnace construction; at present their field is limited to foundations on account of the difficulty of cutting the blocks and the liability to crack and decrepitate upon heating. When employed, they should be laid with the bedding-planes horizontal. Silica brick are used almost exclusively in the roofs of high-temperature reverberatory furnaces or in regenerative chambers where they have little load to bear and where they can expand freely. Quartzite crushed to a sand is burnt in to form the bottom of copper reverberatory and acid open-hearth furnaces, is rammed in to line steel ladles and Manhès copper-matte converters, and thrown into copper reverberatory furnaces and patted down to serve as fettling.

Manufacture and Properties of Silica or Dinas[8] Brick.[9]—Silica brick is composed of a mixture of crushed quartzite and lime. Ordinary fire-sand cannot be used as raw material on account of the lack of angularity of the grains.[10] The usual material is a quartzite of say SiO_2 97.5, Al_2O_3 1.5, Fe_2O_3 0.5 per cent.,

[1] *Stahl u. Eisen*, 1911, XXXI, 224; *Tonind. Z.*, 1911, XXXV; *Rev. Mét.*, 1911, VIII, 275.

[2] *Tonind. Z.*, 1911, XXXV, 1323.

[3] *Stahl u. Eisen*, 1912, XXXII, 392; *Rev. Mét.*, *Extr.*, 1912, IX, 464.

[4] *Phil. Mag.*, 1910, XX, 718; *Rev. Mét.*, *Extr.*, 1911, VIII, 36.

[5] *Am. J. Sc.*, 1909, XXVIII, 322.

[6] *Tonind. Z.*, 1893, XVII, 425.

[7] *J. I. and St. I.*, 1875, II, 522.

[8] The term "dinas brick" originated in England in 1822 with W. W. Young, who made the first silica brick in Wales from so-called dinas clay, containing 96.7–98.3 per cent. SiO_2.

Percy, "Fuel," 2d ed., p. 146.

[9] Crossley, *Clay-worker*, 19,000, XXIV, 268.

Plant of Robinson Co., Akron, O., *Iron Trade Rev.*, 1907, XLI, 339; *Clay Worker*, 1907, XLVIII, 259.

Plant of Anaconda, Austin, *Tr. A. I. M. E.*, 1906, XXXVII, 484; Greaves-Walker, *Tr. Am. Ceram. Soc.*, 1911, XIII, 152.

[10] Photomicrograph of silica brick, Rinne, *Thonind. Z.*, 1903, XXVII, 195, Fig. 5.

melting at Seger cone No. 35 or 35–36.[1] It is essential that some Al_2O_3 and Fe_2O_3 be present to form a bond. Wernicke[2] states that chemical composition and refractoriness alone do not determine the suitability of a quartzite for making silica brick, but that the character of the individual grains is of leading importance. The microscopical examination of a thin section in polarized light shows that in an unsuitable quartzite the individual grains are in close contact, and that in a suitable quartzite they are separated from one another by an amorphous silicious bond. The quartzite, calcined in a shaft furnace if necessary, is crushed in a rock-breaker and then reduced to the required size in a wet-pan with the addition of 2 per cent. $Ca(OH)_2$,[3] in the form of milk of lime, and the necessary water to form a moldable paste. Some quartzite contains just the right proportion of flux so as to require no bonding material. The lime ought to be pure, *i.e.*, as free as possible from MgO and absolutely free from CO_2. The calcining temperature of limestone is 1,000° C.;[4] overburnt lime does not slake satisfactorily; underburnt lime retains CO_2 (800° C.). In crushing in the wet-pan, the aim is to obtain part of the quartzite as a fine flour, *e.g.*,[5] 1/3 flour, 1/3 grains 1 to 3 mm, 1/3 grains 3 to 7 mm. The rollers of the pan ought to be heavy (5 tons, 5 ft. diam., 16-in. face); a 7-ft. pan making 15 r.p.m. and requiring 25 h.p. will prepare 10 tons of mixture in 10 hr. The molding used to be done exclusively by hand, but a pneumatic hammer was sometimes used in filling the mold as with fire-clay brick. At present brick-presses have replaced hand-work to some extent. On account of the expansion in burning, the molds are made about 3 per cent. smaller than the desired brick. The brick are dried on floors or in tunnels; they can be dried more rapidly (say in 3 days) than clay-brick. The molded brick has very little strength, but to overcome the fault is repressed after drying.

The burning is carried on in down-draft kilns.[6] A circular kiln holding 100 tons of brick is about 20 ft. inside diam., has eight fireplaces, height to spring of arch 10 ft., total height 15 to 16 ft. The burning temperature is at Seger cone Nos. 16 to 17; it takes 20 days to burn a charge, including filling and emptying. It is essential that the kiln be brought quickly to the desired temperature, that a minimum of excess air pass through the grate, that the charge be cooled slowly, and air again be excluded, as otherwise the brick are likely to crack. Silica brick are highly refractory, rarely fusing below Seger cone No. 33; tests with three samples gave Kanolt a range of 1,700 to 1,750° C.; they are light-brown in color, and are speckled from particles of Fe abraded from the grinding machinery; they are porous, light in weight, brittle and difficult to cut, poor conductors of heat (see Table 144) and inelastic; they crack when exposed to sudden change of temperature; the linear expansion which is progressive is

[1] Analyses: Lange, *Stahl u. Eisen*, 1912, XXXII, 1729; Grum-Grzimailo, Holmquist, and Endell, quoted above.

[2] *Stahl u. Eisen*, 1913, XXXIII, 235.

[3] Hursh, Dissociation of Calcium Hydrate, *Tr. Am. Ceram. Soc.*, 1912, XIV, 792.

[4] Bleininger, *Tr. Am. Ceram. Soc.*, 1911, XIII, 618.

[5] *Thonind. Z.*, 1881, 238.

[6] Lange, *Stahl u. Eisen*, 1912, XXXII, 1734 (single circular and Mendheim kilns).

about 3 per cent.;[1] the compressive strength[2] shows a range of 1,900 to 4,000 lb. per sq. in; the former figure refers to pure, the latter to impure brick; the refractoriness has been sacrificed (melting-point below Seger cone No. 33) to strength. The heat conductivity is a little greater than that of fire-clay brick.[3]

TABLE 146.—ANALYSES OF SILICA BRICK

	Harbison & Walker "Star Brick"	Anaconda, Mont.	Haws Fire-brick Co., Mt. Union, Pa.	Fayette Mfg. Co.	Sheffield, England	Germany	Germany
SiO_2	95 498	97.2	97.2	96 6	95 40	95.96	95 95
Al_2O_3	1.527	... ...	1.4	0.6	3 10 (with Fe_2O_3)	1.88	1.69
Fe_2O_3	0.744	1.1	0 2	0.55		. . . . . .	0.51
CaO	2.128	1.0	0.76	1.8	2.00	2.06 (with MgO)	1.20
MgO	0 108	....	0.35	0.33	. .....		0.16
Ref.	(1)	(2)	(2)	(2)	(3)	(4)	(5)

(1) Circular; (2) *Tr. A. I. M. E.*, 1906, XXXVII, 485; (3) *Berg. Hüttenm. Z.*, 1890, XLIX, 117; (4) Bischof, Collected Analyses, p. 139; (5) Kerl-Cramer-Hecht, *op. cit.*, p. 890, gives thirty analyses. Blasberg, *Stahl u. Eisen*, 1910, XXX, 1059, gives also analyses of different parts after use.

In laying silica brick it is important to leave room for expansion; thus in a reverberatory furnace for melting sulphide-copper ores, having 12-in. silica brick, 1/4 in. is allowed to 1 ft. in the roof, and 5/16 in. to 2 ft. in the bottom; the expansion space is occupied by cardboard when the brick is being laid. The mortar used is a slurry of finely ground fire-sand and fire-clay in the proportion of 60 : 40.

162. Bauxite and Bauxite Brick.—Bauxite is AlO_3H_3 contaminated with FeO_3H_3, SiO_2, TiO_2, etc. It is mined mainly in France, United States[4] and Ireland, and serves as an aluminum ore, and as a refractory material.

Al_2O_3 melts at 2,010° C.[5]; with SiO_2 it combines only at temperatures not reached in C-heated furnaces; if other bases, such as CaO are present, readily fusible slags are formed. Bauxite melts at 1,820° C. and bauxite clay at 1,795° C. (Kanolt). The variety best suited for a refractory is one containing little Fe_2O_3 and SiO_2. The mineral, however, shrinks as much as 30 per cent. and crumbles in calcining, hence brick cannot be made of bauxite alone. An addition of SiO_2 counteracts the shrinkage. A mixture[6] with Al_2O_3 49, SiO_2

[1] Seger-Cramer, *Thonind. Z.*, 1901, XXV, 496.
Lürmann, *Stahl u. Eisen*, 1901, XXI, 790.
[2] Brown, *Tr. Am. Ceram. Soc.*, 1912, XIV, 393.
[3] Pennock, *Tr. A. I. M. E.*, 1896, XXVI, 263.
[4] Judd, *Eng. Min. J.*, 1907, LXXXIII, 574.
[5] Kanolt, *Technolog. Paper* 10, U. S. Bureau of Standards, 1912.
[6] *Thonind. Z.*, 1903, XXVII, 2132.

TABLE 147.—ANALYSES OF BAUXITE

	France		United States				Ireland		
	Light	Red	Ga. (*a*)	Ala.	Ark.	Ark. (*b*)	No. 1	No. 2	No. 3
Al_2O_3	56–75	60–68	58.62	61.00	60–61	87.30	49–54	51–52	45–46
Fe_2O_3	1–3	12–16	1.51	2.20	2–3	1.43	1.75–2.25	4–5	15–16
TiO_2	3–5	3–4	3.79	3 12	2–3	3.99	5–6	6–7	4–5
SiO_2	10–20	1–3	4.27	2 10	0.5–1	6.40	8–16	11–12	10–11
CaO	......	......	......	..	......	....	0.5–0.6	0 7–0.8	0.15–0.20
H_2O	14–24	11–14	31.44	31.58	32.33	0.88	26–30	23–24	23–24
Reference	(1)	(1)	(2)	(2)	(3)	(4)	(5)	(5)	(5)

(*a*) Average of 28 samples. (*b*) Washed and calcined.

(1) *Thonind. Z.*, 1903, XXVII, 37; (2) *Min. Ind.*, 1904, XIII, 40; (3) *Thonind. Z.*, 1897, XXI, 132; (4) *Eng. Min. J.*, 1906, LXXXI, 217; (5) *Thonind. Z.*, 1897, XXI, 132.

28, H_2O 18 per cent. (calcined, $Al_2O_3=63.6$ per cent.) was found to be more refractory than Seger cone No. 38, and a brick made of it, with Al_2O_3 57.76, SiO_2 39.75, Fe_2O_3 2.30, MgO 0.54, Ign. loss 0.17 per cent., and melting at Seger cone No. 37, stood well as a basic lining. A brick of the composition Al_2O_3 74.78, SiO_2 21.20, Fe_2O_3 3.83, was more refractory than Seger cone No. 39 and also stood well in a basic furnace.

The procedure of manufacture in outline is as follows: the washed bauxite is calcined between 1,350 and 1,400° C. (water is given off at 1,310° C.), ground, pugged with about 4 per cent. of a highly aluminous plastic clay, balled, dried and calcined at a high temperature. The calcined mixture is again ground and pugged as above and molded by hand; the brick are partly dried, re-pressed and burnt at a high temperature. Brick to be used in a basic open-hearth ought not to contain over 12 per cent. SiO_2. Kanolt[1] found the melting-points of commercial brick to range from 1,565 to 1,785° C.

Bauxite is used to tone up fire-clay, *i.e.*, to increase the percentage of Al_2O_3. Heucken[2] has prepared iron-blast-furnace blocks with up to 70 per cent. Al_2O_3, fusing at Seger cone No. 36, which have stood for five to six years. Ordinary pure clay with about 40 per cent. Al_2O_3 (kaolinite =39.8 per cent. Al_2O_3) will form a brick that fulfils any blast-furnace requirement; in furnaces exposed to high temperatures and basic corroding influences, brick with 60 per cent. Al_2O_3 are in place as long as they are sufficiently strong. Increasing the percentage of Al_2O_3 of a high-grade fire-clay for special purposes differs from toning up a low-grade clay to raise the Al_2O_3 content and pass it off as a high-grade article. Bauxite brick[3] are used at present in rotary cement kilns and in lead-softening and refining furnaces. The modification of Bischof's method for determining

[1] *Loc. cit.*

[2] *Thonind. Z.*, 1901, XXV,[2] 1659.

[3] Stone, *Min. Eng. World*, 1912, XXXVII, 341.

the fusibility of fire-clay by Hofman[1] forms a satisfactory mode of procedure for laboratory experiment. Bauxite fused in the electric furnace forms a new refractory called alundum.[2]

163. Lime, Magnesia, Dolomite.[3]—LIME is fusible at 2,000° C. (Moissan); small amounts of impurities, such as clay, SiO_2, Fe_xO_y, lower the melting-point quickly so that impure CaO will frit at 1,000 to 1,400° C. Pure lime is used as a refractory material only in the form of a lime crucible. Thus Deville in 1859 first fused Pt with the O–H gas blowpipe in a lime crucible. Small crucibles are prepared by turning a piece of well-burnt lime in a lathe. Large crucibles are made by ramming finely-divided carbon into a clay crucible, reaming out a cavity so as to leave a lining 0.5 in. thick, ramming in powdery lime and reaming out the cavity that is to hold the charge. Crucibles of electrically fused CaO have not come into use as yet.

LIMESTONE crushed to pass a 5- or 8-mesh sieve and mixed with 1/3 to 1/4 of its volume of fire-clay, or the natural mixture "marl" ($CaCO_3$ 65–66, Al_2O_3 5–7, SiO_2 21–24, Fe_2O_3 3–5, $MgCO_3$ 1–2)[4] is commonly used for lining the test of a cupelling furnace. Bone-ash ($Ca_3P_2O_8$ 84, $CaCO_3$ 10, $MgCO_3$ 2, CaF_2 4), formerly used for the same purpose, is confined almost wholly to assay-cupels.

MAGNESIA.[5]—MgO is more refractory than CaO, melting at 3,000° C. (Moissan); Goodwin-Mailey[6] found a lower figure, 1,910° C. It has always been held that MgO could not be reduced to Mg by C; this is still true in part. Watts[7] found that the reaction $MgO+C \rightleftarrows Mg+CO$ was reversible, at a high temperature it went from left to right, and at a low from right to left.

The raw material is magnesite, $MgCO_3$, with 47.6 per cent. MgO which begins to lose CO_2 at 450° C.[8] Prepared $MgCO_3$ gives up CO_2 at a lower temperature. Magnesia crucibles are used in the laboratory.[9] Calcined magnesite is used in the form of sand and brick in basic open-hearth furnaces; brick are occasionally employed in furnaces for softening, refining and cupelling lead, and for smelting, converting and refining copper. The leading deposits occur in Veitsch, Styria, and Euboea, Greece. Some magnesite is mined in Hungary,[10] India[11] and in California.[12] Styrian magnesite[13] averages: $MgCO_3$ 90.0 to 96.0,

[1] *Tr. A. I. M. E.*, 1898, XXVIII, 435.
[2] Norton Co., *Met. Chem. Eng.*, 1911, IX, 225.
Boeck, *Metallurgie*, 1912, IX, 341.
Fitzgerald, *Met. Chem. Eng.*, 1912, X, 128.
[3] Havard, Basic Refractory Materials, *Eng. Min. J.*, 1908, LXXXVI, 802
[4] Kerl-Wimmer, *Berg. Hüttenm. Z.*, 1853, XII, 245, 246.
Scherr, R., "Der Magnesit," Hartleben, Vienna–Liepsic, 1908.
[5] Scherr, R., "Der Magnesit," Hartleben, Vienna, 1908.
[6] *Trans. Am. Electrochem. Soc.*, 1906, IX, 89.
[7] *Tr. Am. Electrochem. Soc.*, 1907, XI, 279.
[8] Brill, *Zt. phys. Chem.*, 1905, XLV, 283.
[9] Watts, *Min. Sc.*, 1912, LXVI, 413.
[10] *Stahl u. Eisen*, 1909, XXIX, 294.
[11] *J. Soc. Chem. Ind.*, 1909, XXVIII, 503; *Thonind. Z.*, 1909, XXXIII, 1416.
[12] *Eng. Mag.*, 1906, XXXI, 691; *Bull.* 355, U. S. Geol. Surv., 1908.
[13] Bischof, *Oest. Zt. Berg. Hüttenw.*, 1893, XLI, 27.

$CaCO_3$ 0.5 to 2.0, Fe_2O_3 3.0 to 6.0, SiO_2 up to 1.0, Mn_2O_3 up to 0.5 per cent. Grecian magnesite[1] contains $MgCO_3$ 95.151, $CaCO_3$ 4.02, Fe_2O_3 trace, SiO_2 0.5, H_2O 0.34; an average analysis of 4,300 tons sintered Styrian magnesite[2] gave MgO 85.30, CaO 1.76, Fe_2O_3 7.79, SiO_2 3.40, Al_2O_3 0.82; calcined Grecian magnesite[3] has the following range of composition: MgO 82.46 to 95.36, CaO 0.83 to 10.92, $Fe_2(Al_2)O_3$ 0.56 to 3.54, SiO_2 0.7 to 7.98.[4] Styrian magnesite has a sufficient amount of impurity to cause it to sinter when it is calcined. The sintering is believed to be caused mainly by the Fe[5] that is present. This property causes it to be preferred as a refractory material to the purer Grecian which does not sinter, but crumbles after it has been calcined.

Grecian magnesite is often calcined in retorts to obtain the CO_2 for the manufacture of effervescent beverages; the residue is then re-calcined at an elevated temperature and used either to tone up low-grade magnesite or made into brick with serpentine as a bonding material.[6] When magnesite is calcined at a red heat[7] it loses its CO_2 (more readily than $CaCO_3$); it shrinks considerably, but still retains the property of forming with H_2O a $Mg(OH)_2$ which is slightly plastic and can be molded under a high pressure. When calcined at a high white heat,[8] its shrinkage amounts to 50 per cent., and the residue is hard, dense, and has lost all plasticity and nearly all shrinking power; it is not attacked by CO_2 or H_2O at ordinary, nor by SiO_2 at elevated temperatures. The method of making a brick mixture of soft- and hard-burnt magnesite under a high pressure has fallen into disuse on account of the high shrinkage in burning.[9]

In the present mode of manufacture[10] hard-burnt magnesite is used exclusively. When this is exposed to the air, the lime present will slake and can be in part removed by cobbing and blowing off the dust. Burnt magnesite to be used for tamping must be protected from moisture. The dense burning[11] of magnesite is carried in direct or gas-fired shaft furnaces and continuous gas-fired kilns, such as the Mendenheim (p. 361); the range of temperature lies between Seger cones Nos. 17 and 24. The calcined material is broken in crushers, pulverized in ball mills to 1.5 mm., moistened with 4 to 5 per cent. water, thoroughly pugged, molded in a hydraulic press with steel shoes and dies at a pressure of 400 tons, air-dried and burnt at Seger cone Nos. 18 to 22; some varieties are burnt at cone No. 26. The bricks have a reddish color, are hard, strong, give a clear ring, are porous (20 per cent. absorption), have a specific gravity 3.03 to 3.05; they, however, still shrink at the temperature of the open-

[1] McGinley, *Eng. Min. J.*, 1899, LXVIII, 665.
[2] Wedding, *Stahl u. Eisen*, 1893, XIII, 283.
[3] Bischof, *Oest. Zt. Berg., Hüttenw*, 1893, XLI, 27.
[4] Collection of Analyses, *Min. Ind.*, 1911, XX, 498.
[5] Bleichsteiner, *Oest. Zt. Berg. Hüttenw.*, 1892, XL, 355, 486.
[6] Schmatolla, *Thonind. Z.*, 1900, XXIV, 1470.
[7] Caron, *Compt. rend.*, 1868, LXVI, 839.
[8] Fitzgerald, *Met. Chem. Eng.*, 1912, X, 131.
[9] Schimm, *Thonind. Z.*, 1905, XXIX, 1968, 1981; *Stahl u. Eisen*, 1906, XXVI, 303.
[10] Hörhager, *Stahl u. Eisen*, 1911, XXX, 955.
[11] Schmatolla, *Thonind. Z.*, 1900, XXIV, 1470.

hearth furnace. They scale when subjected to sudden change of temperature.[1] As manufactured by the Harbeson-Walker Refractories Co. the brick expand in the furnace. Kanolt[2] found their melting-point to be 2,165° C. Brown[3] found that at 1,550° C. the brick yielded suddenly to a sheering stress. The heat conductivity[4] is greater than with either of fire-clay or silica brick (Table 144).

TABLE 148.—ANALYSES OF MAGNESITE BRICK

Locality	MgO	CaO	Fe_2O_3	Al_2O_3	Mn_3O_4	SiO_2	S	P_2O_5	
Styria	85.31	tr.	9.	05	0 52	4.70	... .	......	Pittsburgh Testing Laboratory.
Norway ...	83.6	....	4.6	2.0	0.05	9.3	0.003	0.046	*Electrochem. Met. Ind.*, 1906, IV, 30.
Greece	86 5	3 76	6	64	.....	3.10	....	......	*Tr. A. I. M. E.*, 1896, XXVI, 263.
U. S..........	93.03	4.20	0	72	......	2 16	....	....	*Tr. A. I. M. E.*, 1896, XXVI, 263.
U. S.........	87.66	1.19	8 10	1.19	.....	2.05	....	......	Private notes.
U. S.........	90.22	0.00	6.60	1.68	......	1.38	....	..... .	Private notes.

The mortar used in laying the brick is a slurry of fine magnesite, which has been calcined but not fritted, with tar or linseed oil. Only hard-burnt magnesite can be used for tamping in a hearth; the binder is either tar freed from water or preferably finely ground calcined, but not fritted, magnesite. Several attempts have been made to use prepared MgO as a refractory material, but without success.

DOLOMITE is used as a refractory material in the natural state for the same purposes as is limestone. Calcined dolomite came into prominence in 1878 with the introduction of the basic Bessemer and the basic open-hearth processes. It is used in the form of sand, to be rammed or burnt in, and in the form of brick. Normal dolomite, $(MgCa)CO_3$, contains MgO 21.7 and CaO 30.4 per cent. The ratio of MgO and CaO varies in different localities; the range of composition is MgO 16 to 22, CaO 28 to 36, Al_2O_3 0 to 4, Fe_2O_3 0.3 to 4, SiO_2 0.5 to 4, CO_2 43 to 48 per cent. Within limits, the richer in MgO, the more valuable the stone.[5] The presence of about 20 per cent. CaO has been found to be of importance;[6] further the rock ought not contain over 4 per cent. $Fe(Al_2)O_3$ nor over 3 per cent. SiO_2, if it is to serve as a refractory material.

[1] *Eng. Min. J.*, 1912, XCIV, 1018.

[2] *Technolog. Paper* 10, U. S. Bureau of Standards, 1912; *Proc.* VIII, Internat. Cong. Appl. Chem., 1912, Vol. XXII, p. 171.

[3] *Tr. Am. Ceram. Soc.*, 1912, XIV, 391.

[4] Massenez, *Stahl u. Eisen*, 1881, I, 99.
Ramdohr, *Thonind. Z.*, 1883, VII, 4.
Pennock, *Tr. A. I. M. E.*, 1896, XXVI, 263.

[5] Zyromski, *Compt. rend. Soc. Ind. Min.*, 1886, p. 106. *Stahl u. Eisen*, 1886, VI, 623.

[6] Zyromski, l. c.
Editor, *Thonind. Z.*, 1900, IV, 638.

TABLE 149.—ANALYSES OF DOLOMITE USED IN BASIC STEEL PROCESSES

	Tenn.	Pa.	Pa. (Reading)	Pa.	Sweden		France Bessèges	England Whitehaven	Germany	
									Hörde	Duisburg calcined
SiO_2	....	5.52	3.65	0.77	3.70	1 66	3 80	1 10	1 35	5 58
Fe_2O_3	.	3.74	0 91	....	0.88	0 82	4 00	1 14	0 29	2 84
Al_2O_3	..		0 71	0.10		1.04			2 05	1 34
MnO	.....	.....	....	....	....	0.36	.....	....	...	...
MgO	19.50	17.31	20.43	21 39	20 4	19 15	17.0	20 0	19 21	35 12
CaO	29.60	29 20	30.36	30 64	30.24	30.08	28 0	31.42	30.12	55 27
P_2O_5	.....	......	....	.....	0.022	0.007	.....	......	.....	....
S	....	......	....	....	....	0.034	......	....	....	.....
CO_2	45 11	43.82	.....	....	46.24	......	45.0	46.2	44 97	....
H_2O	......		.....	......	....	....	......	....	2.00	....
Ref	(1)	(7)	(8)	(8)	(2)	(2)	(3)	(4)	(5)	(6)

(1) *Iron Age*, 1894, LIII, 845; (2) *Thonind. Z.*, 1900, XXIV, 638; (3) *Compt. rend. Soc. Ind. Min.*, 1886, 106; (4) Harbord, "Steel," p. 58; (5) Ledebur, "Eisenhüttenkunde," 4th ed., p. 187; (6) *Stahl u. Eisen*, 1882, II, 120; (7) Harbison-Walker Refractories Co.; (8) Private notes.

Dolomite is calcined[1] in a cupola similar to that used in melting pig iron, excepting that near the bottom there are four to six drawing doors to remove the calcined material. A cupola is used, as it is essential to calcine at a white heat in order that the product may be slightly fritted and thus the property of slaking impaired. Such a cupola has, *e.g.*, the following dimensions: inside diam. 6 ft., height to tuyeres 4 ft. and to feed-door 16 ft.; there are eight 4-in. tuyeres; the lower 5 ft. of the cupola are lined with dolomite, the rest with fire-brick followed by red brick; the capacity for 24 hr. is 14 tons calcined material. A larger cupola has the following dimensions: inside diam. 9 ft., height to tuyeres 5 ft. and to feed-door 14 ft., 7 tuyeres, 4 in. diam. The shell is lined with 4 1/2 in. clay and followed by 9 in. of chrome brick in the hot zone; the rest is red brick; the capacity per 24 hr. is 30 tons dolomite; the coke consumption 20 per cent. on dolomite. In starting, the cupola is half filled with coke, then follow alternate layers of dolomite (first-size) and coke. The calcined material is sorted to remove imperfectly calcined parts which are returned to the furnace. The well calcined part is crushed in a dry-pan to pass a 1/4-in. sieve and is then ready to be burnt in for a furnace-bottom, or it is mixed with 7 to 10 per cent. dehydrated tar in a steam-heated pan by hand or preferably mechanically to a mass which is plastic while warm and hardens upon cooling. The mixture is ready to be rammed in or it is formed into bricks which are kiln-burnt and then laid with a mortar consisting of burnt dolomite-dust and dehydrated tar. Dolomite brick are not much used now.[2]

164. Chromite and Chrome-brick.—Chromite,[3] $FeCr_2O_4$, contains Cr_2O_3 68.0 and FeO 32.0 per cent., usually the Cr_2O_3 is replaced in part by Al_2O_3 and

[1] Gerson, Plant of Osnabrück, *Stahl u. Eisen*, 1907, XXVII, 1066.

[2] Examples: Pottstown (Pa.) Basic Bessemer Plant, *Iron Age*, May 28, 1891; *Tr. A. I. M. E.*, 1892–93, XXI, 745; Drawing of Brick-plant, *Stahl u. Eisen*, 1902, XXII, 1201.

[3] Pratt, *Eng. Min. J.*, 1898, LXVI, 696.

Fe_2O_3, and the FeO by MgO. The ore[1] contains Cr_2O_3 38 to 40, Al_2O_3 24.5, Fe_2O_3 17.5, SiO_2 3.25, MgO 15 per cent. The mineral is infusible in C-heated furnaces, melting at 2,180 ° C.[2], and little attacked by acid or basic slags.

Experiments by Simonis[3] in determining the effect of Zettlitz kaolin (Table 142, melting at Seger cone No. 35) upon chromite (Cr_2O_3 52.9, FeO 22.6, Al_2O_3 4.8, SiO_2 9.6, MgO 10.1, melting at Seger cone No. 42) gave a V-shaped curve with the eutectic point at Seger cone No. 15, the mixture containing 35 per cent. kaolin.

Chromite was first used in the shape of blocks to form in the basic open-hearth furnace the neutral joint between the acid (silica brick) roof and the basic bottom (calcined dolomite), the mortar employed being a mixture of chromite fines.

Chromite brick have replaced the blocks; they are also used quite extensively in building up the bottom of copper[4] and copper-nickel blast-furnaces, in the sides of copper refining furnaces, and as shown above in lining the hot-zone of the cupola for calcining dolomite. Chromic iron ore[5] is mined mainly in Greece,[6] Turkey,[7] Asia Minor,[8] Russia, New Caledonia, New South Wales and Canada;[9] some ore is produced in California[10] and Newfoundland. It contains[11] Cr_2O_3 40 to 50 per cent., Fe_2O_3 and Al_2O_3 about 20 per cent., MgO 10 to 15 per cent., SiO_2 3 to 6 per cent.

For the manufacture of chrome brick, crushed chromite or concentrate is mixed with the necessary bond, molded by hand or machinery (similarly to magnesite brick) dried and baked. The bond used is clay, CaO and MgO, as the refractory serpentine rock, in which the mineral usually occurs, combines with these materials and holds firmly together the particles of chromite.[12] Seger-Cramer[13] patented the addition of 2 per cent. $CaSO_4$ and 1 per cent. $Al_2.3SO_4$ (=0.85 per cent. CaO and 0.151 per cent. Al_2O_3) to the crushed chromite, which renders the fine ore sufficiently plastic to permit its being molded. The sulphates are decomposed by heat and the small addition of fluxes does not greatly reduce the refractoriness.

[1] Harbison-Walker Refractories Co., Pittsburgh, 1909. "A Study of the Open Hearth," p. 22.

[2] Kanolt, *Technolog. Paper* 10, U. S. Bureau Standards, 1912.

[3] *Stahl u. Eisen*, 1908, XXVIII, 334.

[4] Glenn, *Tr. A. I. M. E.*, 1901, XXXI, 374.

[5] *Min. World*, 1911, XXXIV, 883.

[6] Habets-Bonanos, *Rev. Un. Min.*, 1908, XXI, 129, 139.

[7] *Eng. Min. J.*, 1902, LXXIV, 275.

[8] *Tr. Instit. Min. Met.*, 1894–95, III, 448.

[9] *J. Can. Min. Inst.*, 1899, II, 25; *Gen. Min. Assoc. Quebec*, 1894–95, II, 108.
F. Crickel, "Chrome Iron Ore Deposits of Quebec," Ottawa, 1909.

[10] *Min. Sc. Press*, 1897, LXXIV, 301; *Cal. State Min. Bureau Rep.*, 1895–96, XIII, 490.

[11] Analyses: Leo, *Stahl u. Eisen*, 1891, XI, 643; Busek, *Thonind. Z.*, 1893, XVII, 372, 397; Steger, *Zt. Berg. Hütten. Sal. Wesen i. Pr.*, 1901, XLIX, 101; *Min. Ind.*, 1905, XIV, 72; 1911, XX, 125.

[12] Lundström, *Berg. Hüttenm. Z.*, 1890, XLIX, 316.

[13] *Thonind. Z.*, 1893, XVII, 1144.

TABLE 150.—ANALYSES OF CHROME-BRICK

Locality	SiO_2	Cr_2O_3	Fe_2O_3	Al_2O_3	CaO	MgO	Authority
?	5 23	35.80	15 26	31.28	0 91	11.43	Seger, "Sprechsaal," 1893, p. 107 5.
?	9.20	29 10	16.14	21.36	0 01	24.13	Bischof, Collected Analyses, p. 143.
Pennsylvania ..	10 93	43 17	28 4	11 60	tr.	21.20	Private notes, 1913.

Chrome brick melts at 2,050° C.,[1] becomes mechanically weak at 1,450°, expands considerably upon heating,[2] and is a good conductor of heat (Table 144).

165. Carbon, Brasque, Carbon Bricks.—The property C possesses of not being attacked by heat,[3] change of temperature and slags, makes it suitable for refractory purposes under reducing conditions. The three available forms of C are charcoal, coke, and graphite.

GRAPHITE[4] is iron-black to dark steel-gray, opaque, soft, has a greasy feel and a metallic luster when scaly, but is dull when earthy; it is a good conductor of heat and electricity, has a high heat capacity, burns very slowly in air, is not attacked by acids or alkali solutions, is oxidized by heating with KNO_3 or boiling with $KCrO_4$ and H_2SO_4. As graphite ores are usually low-grade, they have to be enriched by combined wet and dry concentration processes to furnish a product of the required grade of 60 to 70 per cent. C. The purest graphite mined is that of Ceylon with about 70 per cent. C.; it is especially adapted for the manufacture of refractory articles, as, owing to its irregular fracture, the particles are more strongly bound together by the fire-clay cementing material than is concentrated graphite which has a flaky to earthy character. The artificial graphite of the International Acheson Graphite Co., Niagara Falls, N. Y.,[5] is used in the manufacture of paint, stove polish, lubricant, electrical and chemical carbons. A later product is the "Deflocculated Graphite" of Acheson.[6] The use of charcoal and coke in the preparation of brasque has been referred to in

[1] Kanolt, *loc. cit.*
Brown, *Tr. Am. Ceram. Soc.*, 1912, XIV, 391.
[2] Shelby, *Eng. Min. J.*, 1008, LXXXV, 851.
[3] Borchers, "Fusion (?) of Carbon," *Metallurgie*, 1912, IX, 230.
[4] Weinschenk, E., "Der Graphit," Druckerei, A. G., Hamburg, 1898.
Donath, E., "Der Graphit," Deuticke, Leipsic-Hanover, 1904.
Escard, J., "Le Carbon," Dunod, Paris, 1906.
Cirkel, F., "Graphite, Its Properties, Occurrence, Refining and Uses," Department of Mines, Ottawa, 1907.
Mène, *Compt. Rend.*, 1867, LXIV, 1091.
Downes, W. F., *Iron Age*, 1900, April 19, May 3, 10, 24, June 14.
Donath, *Stahl u. Eisen*, 1906, XXVI, 1249.
[5] Fitzgerald, F. A. J., "Künstlicher Graphit," Knapp, Halle, 1904; *Eng. Min. J.*, 1903, LXXV, 484.
Donath, *Stahl u. Eisen*, 1906, XXVI, 1249.
[6] Acheson, *J. Frankl. Inst.*, 1907, CLXIV, 375; *Electrochem. Met. Ind.*, 1902, I, 52; 1909, VII, 187.

§159. Carbon brick[1] have been made of graphite and clay, coke and clay, and coke and pitch.[2]

TABLE 151.—ANALYSES OF CARBON BRICK

Kind	C	SiO_2	Fe_2O_3	Al_2O_3	CaO	MgO	Ash	Reference
Coke-clay.......	64 23	21.51	1.41	12.05	0.67	0 29	....	Gayley, *Tr. A. I. M. E.*, 1892–93, XXI, 116.
Coke-pitch	87.26	.....	.....	.....	.....	.....	12.74	

At present coke-pitch brick are being mainly manufactured. The coke, as free from ash as possible, is ground in a ball-mill to a limiting size of 2 mm. and pugged with about 20 per cent. tar, freed from water and light oils, in a horizontal open-mouth mill heated from below. The mixture is rammed hot with heated tools into wooden molds painted with residuum oil. The bricks are removed at once from the molds, the tiles after half a day, when they have come to set; they are then kept for 2 or 3 weeks in a cool chamber and baked in a built-up muffle at the temperature of Seger cone No. 10. The bottom of the muffle receives a layer of 4 to 8 in. coke crushed to 1/16 in. size.

The bricks are placed on this 2 in. apart and from the side-wall; the open spaces are filled with coke which is tamped down, as in heating the bricks soften and do not retain their shape unless supported. A good coke brick should show a crushing strength of 4,600 lb. per sq. in., have a specific gravity of 1.2 to 1.4 and give a clear ring when struck; the fracture should be fine-grained and free from pores. The mortar used in laying the brick consists of two parts coke dust and 1 part clay.

GRAPHITE[3] CLAY MIXTURES are used in the manufacture of crucibles,[4] retorts, nozzles for steel ladles, etc. Their heat conductivity (Table 144) is twelve times that of a fire-clay mixture.[5] The Joseph Dixon Crucible Co.[6] uses for crucibles[7] a mixture of 50 per cent. graphite, 33 per cent. air-dried fire-clay and 17 per cent. fire-sand. The graphite is ground to pass screens ranging from 40- to 100-mesh, and the sand through a 40-mesh sieve. Coarse materials

[1] Raymond, *Tr. A. I. M. E.*, 1896, XXVI, 185.

[2] Hofman, *Min Ind.*, 1893, II, 435.

[3] Haenig, A., "Der Graphit," Hartleben, Vienna, 1910.

Ihne, Graphite in the U. S., *Min. Sc.*, 1909, LX, 297, 316, 343.

Roush, Microscopical Examination, *J. Ind. Eng. Chem.*, 1912, III, 368; *Metallurgie*, 1912, IX, 166.

Le Chatelier-Wolgodine, "Specific Gravity," *Rev. Mét.*, 1908, V, 140.

Bastin, "Ceylon Graphite," *Econ. Geol.*, 1912, VII, 419.

[4] *Brass World*, 1911, VII, 307.

[5] Pourcel, *Stahl u. Eisen*, 1885, V, 484, col. 1.

[6] *Min. Ind.*, 1893, II, 341.

[7] Walker, Manufacture and Care of Crucibles, *J. Am. Found. Assoc.*, through *Iron Age*, May 20, 1897, p. 16.

make a crucible too porous; fine materials too dense and likely to crack. The clay is mixed with water to form a fine paste, sand and graphite are stirred in; the whole is pugged and then tempered for several weeks. The crucibles are molded on a wheel which places the graphite flakes tangentially and causes the mass to bind better than when crucibles are made in a press. They are then placed in a plaster mold for 3 hr. in which time part of the water is absorbed, dried for 1 week at 30° C., then for another week or longer at 60° C. and burnt slowly in a pottery kiln reaching Seger cone 011 to 07. In crucibles used for making crucible steel, which under any circumstances last only a few charges,[1] earthy graphite is used more extensively than in crucibles used for melting non-ferrous metals and alloys, as these last for 40 to 60 fusions.[2] Retorts and nozzles are made of mixtures similar to those used for crucibles.

TABLE 152.—ANALYSES OF GRAPHITE CRUCIBLES (a)

Kind	English	Morgan	German	Hessian	Austrian
SiO_2	50.0	37.09	33	25.91	13.04
Al_2O_3	20.0	14.58	12–13	11.26	6.12
Fe_2O_3	1.5			0.51	0.44
P_2O_5	......	....	....	....	0.01
C	25.5	44 40	48–50	58.24	77.80
H_2O	3.0	2.92	....	2 77	1.95
CaO	0.5	....	....	tr.	....
K_2O	....	....	....	....	0.43
Loss	0.5	....	....	....	....
Reference	(3)	(4)	(5)	(4)	(6)

(a) *Tonind. Z.*, 1908, XXXII, 225, 498.
(3) Mène, *Compt. rend.*, 1867, LXIV, 1091. (4) Kerl-Cramer-Hecht, *op. cit.*, p. 954.
(5) *J. Am. Found. Assoc.*, 1901–02, X, 3, 53. (6) *Electrochem. Met. Ind.*, 1909, VII, 82.

SILOXICON[3] (Si_2C_2O ?) of Acheson, produced in an electric furnace at about 2,500° C., practically imperfect carborundum, is a greenish substance which is very refractory, insoluble in molten metal and not attacked by slags; it is decomposed in the dry way by alkali, oxidized in O at 1,500° C., and dissociated at 3,000° C. An analysis gave Si 57.7, C 25.9, Fe 2.1, Al 0.4, Ca and Mg tr., O by diff. 13.9. It can be shaped by using 2 per cent. Al_2O_3 as a bonding material and baked below Seger cone No. 19; an addition of 5 per cent. clay gives additional strength to the molded articles. An analysis of siloxicon brick gave: SiO_2 46.26, Al_2O_3 8.42, Fe_2O_3 1.20, C 10.68, Si and other elements in combination 21.36.

[1] Sperry, Life of Crucibles, *Brass World*, 1908, IV, 75.
[2] Annealing Crucibles, *Metal Ind.*, 1907, IV, 293.
[3] *Electrochem. Ind.*. 1903, I, 287, 373; 1904, III, 442; 1909, VII, 191.

Carborundum,[1] SiC, is produced by the reduction of SiO_2 by means of C at 1,820° C.[2] in an electric resistance furnace, and dissociated at 2,220 to 2,245° C.[3] It is a crystallized substance ranging in color from pale yellow-green to emerald-green; has a specific gravity 3.12 to 3.22, hardness 9 to 10 (used mainly as an abrasive material); is infusible in metallurgical furnaces, not attacked by acids (even HF) by heating in O, slightly attacked by caustic alkali, decomposed by fusion with Na_2CO_3. It has been used as a paint to protect fire-brick.

Siloxicon,[4] Silundum,[5] Crystolon[6] resemble carborundum in their properties.

Kryptol[7] is a fire- and electric-current resisting material made up by mixing graphite, carborundum (?) and clay in proportions to form a granular mass which yields upon combustion 0.24 to 0.60 per cent. ash.

166. Metal.—The best manner of protecting a refractory material against heat and pyrochemical corrosion is to keep it cool. This is done by backing it with metal which in its turn is prevented from becoming heated by means of a stream of water. The metals employed are wrought iron, low-C steel, cast steel, cast-iron, bronze and Cu. Sometimes a water-cooled pipe is placed in the brickwork (cupelling-furnaces) or a water-box (bosh-plate in the iron blast-furnace); the outside of brickwork is enclosed by a steel plate over which water flows (spray-jacket); part of the brick is replaced by a water-box (water-jackets of lead blast-furnaces); in most copper blast-furnaces the whole shaft is made up of water-jackets; the material which is chilled on the water-cooled metal forms a natural refractory lining.

[1] Acheson, *J. Frankl. Inst.*, 1893, CXXXVI, 194.
Fitzgerald, *op. cit.*, 1897, CXLIII, 81.
Blake, *Eng. Min. J.*, 1893, LVI, 270, 320.
Fitzgerald, *Cass. Mag.*, 1895–96, IX, 386.
Editor, *Electrochem. Met. Ind.*, 1909, VII, 189.

[2] *Met. Chem. Eng.*, 1911, IX, 613; 1912, X, 289.

[3] Saunders, *Met. Chem. Eng.*, 1912, X, 289.

[4] *Met. Chem. Eng.*, 1911, IX, 613.

[5] *Electrochem. Met. Ind.*, 1909, VII, 24; *Zt. Electrochem.*, 1909, XV, 725.

[6] *Met. Chem. Eng.*, 1912, X, 128.

[7] Fitzgerald, *Electrochem. Met. Ind.*, 1905, III, 5; 1906, IV, 210.
Voelker, *Verh. Ver. Bef. Gew.*, 1904, LXXXIII, 102.
Bronn, *Electrotech. Ztschft.*, 1906, XXVII, 213.
Rodgers, *Iron Age*, 1909, LXXXIV, 1482 (Foundry).
Baraduc-Müller, *Rev. Mét.*, 1909, VI, 700; *Tonind. Z.*, 1910, XXXIV, 890, 905.

CHAPTER VIII

PYROMETALLURGICAL PROCESSES AND APPARATUS

167. Processes and Apparatus in General.—All processes outlined under this head are carried on at elevated temperatures. The heat may be raised only sufficiently to drive off hygroscopic water and thus *dry* a substance; it may be high enough to remove combined water and other volatile components by *calcining;* a chemical change may be effected below fusion by *roasting,* or at fusion by *smelting,* or separations may be accomplished by *liquation, crystallization, distillation* and *sublimation;* lastly metals or alloys may be subjected to various *heat treatments* in order to influence their structure and physical properties.

These operations are carried on in different kinds of furnaces which have been classified according to the manner of heating and the kind of fuel used, according to the form, and the metallurgical processes carried on. In the following general statement the manner of heating has been given the first consideration; the form of the furnace and the process carried on have been made subsidiary, as the various forms merge into one another so as to make it often difficult to determine where one stops and the other begins, and the process, although influenced by the manner of heating, is not always determined either by it or by the form of the furnace.

Classification of Furnaces

I. Fuel Furnaces:[1]
- A. The charge in contact with fuel is heated by the combustion of the fuel.
  - 1. Hearth Furnaces.
    - (a) Without blast: heap and stall for roasting and coking, trough for liquating, etc.
    - (b) With blast: smith's forge, lead ore-hearth, Catalan forge, etc.
  - 2. Shaft Furnaces.
    - (a) With natural draft: calcining and roasting furnaces.
    - (b) With induced draft: some cupolas.
    - (c) With blast: smelting blast-furnaces, most cupolas.
- B. The charge separate from fuel is heated by a flame.
  - 1. Reverberatory Furnaces.
    - (a) With natural draft: most roasting furnaces.
    - (b) With forced draft: most smelting furnaces.

[1] Ledebur, A., "Die Oefen für Metallurgische Zwecke," Craz and Gerlach, Freiberg, 1878. Havrez, *Rev. Un. Min.*, 1862, XI, 383, XII, 1.

C. The charge enclosed in a vessel is heated by conduction.
   1. Closed-vessel (crucible-, retort-, tube-, muffle-) furnaces heated by contact with
      (a) Solid fuel.
      (b) Flame.
      (c) Electric current.

D. The charge containing its own fuel is oxidized.
   1. In a preheated space.
      (a) The charge is solid: coarse- and fine-ore roasting kiln; beehives for coking.
      (b) The charge is liquid: Bessemer, Manhès, David, etc., converters.
   2. In a non-preheated space: up- and down-draft blast-roasting apparatus.

II. Electric Furnaces:[1]

A. Resistance-heating Furnaces.
   1. Direct resistance-heating: the charge is itself included in the electric circuit, *e.g.*, Héroult aluminum-alloy furnace, Hall aluminum furnace. Kjellin induction furnace.
   2. Indirect resistance-heating: the charge to be heated is in contact with an electrically heated conductor; it is piled around a carbon core heated by the current (Acheson carborundum furnace) or is enclosed in a vessel (crucible, muffle, tube) wound with a wire or ribbon (Fe, Ni, Pt) heated by the resistance of the wire to the current.

B. Arc-heating Furnaces.
   1. Direct arc-heating: the charge to be heated forms one or both poles of an electric arc: Siemens original crucible furnace.
   2. Combined resistance- and arc-heating, *e.g.*, the Héroult, Girod, etc., steel furnaces.
   3. Indirect arc-heating: the charge to be heated is in a space heated by an electric arc: the Stassano furnace.

HEARTH FURNACE.—A hearth furnace is a structure with plan area greater than elevation area, in which the charge is heated by the direct action of the solid fuel contained in it or added to it. The simplest form of hearth worked without blast is the pyramidal or conical *heap* used in charring wood (Figs. 116 to 120), coking coal and roasting sulphide copper ore. (Figs. 240 and 241.) When three of the movable sides of a pyramidal heap are replaced by permanent walls, there is formed the oblong *stall*, the fourth side of which is closed by a temporary wall when the enclosed space is being filled; and taken down again when the charge has been treated. (Figs. 242 and 243.)

A LIQUATION TROUGH was formerly used in the desilverization of argenti-

[1] Minet, A., "Le Four Électrique," Hermann-Desforges, Paris, 1905–1908.
Borchers, W., and Solomon, "Electric Furnaces," Green & Co., London, New York, 1908.
Stansfield, A., "The Electric Furnace," McGraw Hill Book Co., New York, 1914.
Richards, *Aluminum World*, 1899, v, 105, 126, 149.

ferous copper by means of lead.[1] In a modified form the trough is still employed in southeast Missouri for drossing lead.[2]

Of the hearths worked with blast, the simplest form is the smith's forge, a back-wall with tuyere-pipe blowing into the coal heaped in front of it. Placing a wall on either side of the back-wall and having a depression to hold the liberated

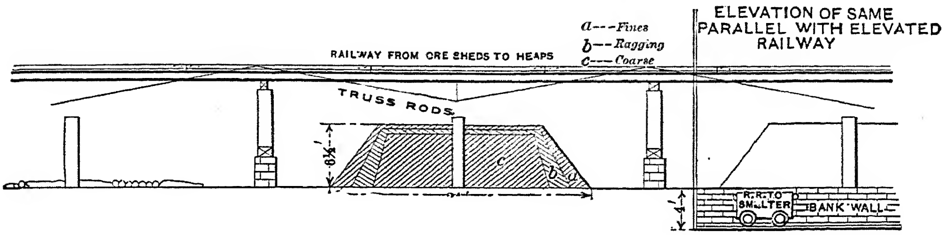


FIG. 240.—Cross-section through roast heaps for sulphide ore parallel with elevated railway.

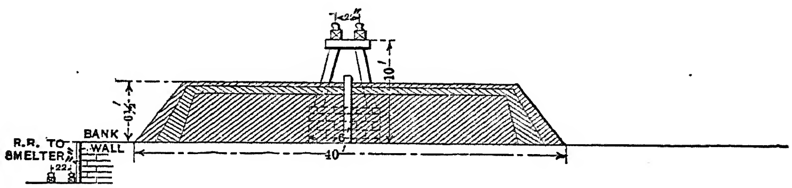


FIG. 241.—Longitudinal section through heap across railway.

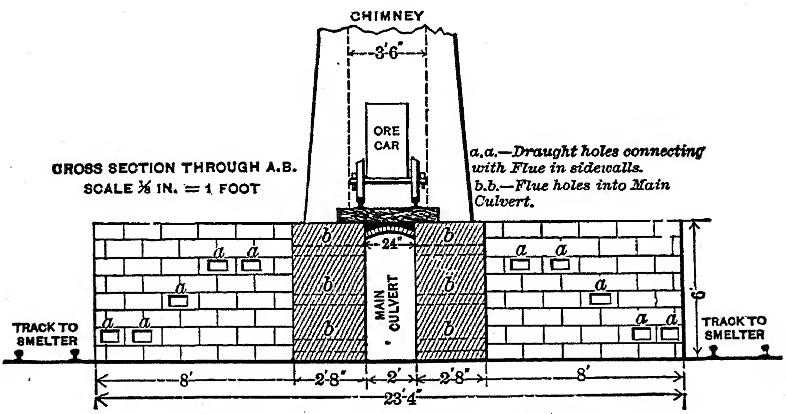


FIG. 242.—Roast stalls for sulphide ore.

metal gives the typical forms of a lead ore-hearth, Figs. 244 and 245, or a Catalan forge. These hearths are small.

Another type of blowing ore-hearth which has a larger capacity is represented by the Wetherill grate (Fig. 208), and Bartlett furnaces for producing zinc white and white zinc-lead pigment; a third type is represented by the up-draft

[1] Percy, "Silver and Gold," Murray, London, 1880, I, 303.

[2] Brinsmade, *Mines and Minerals*, 1901–02, XXII, 300.

blowing kettles employed in blast-roasting (§188). They are larger than ore-hearths, but still work intermittently as do all ore-hearths. The thermal efficiency, §189, of an ore-hearth is low, 4 to 7 per cent.

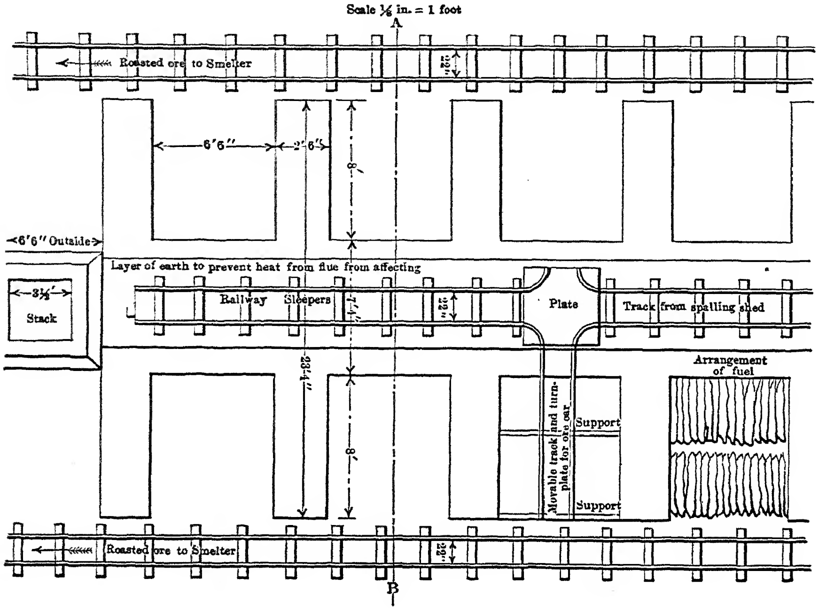
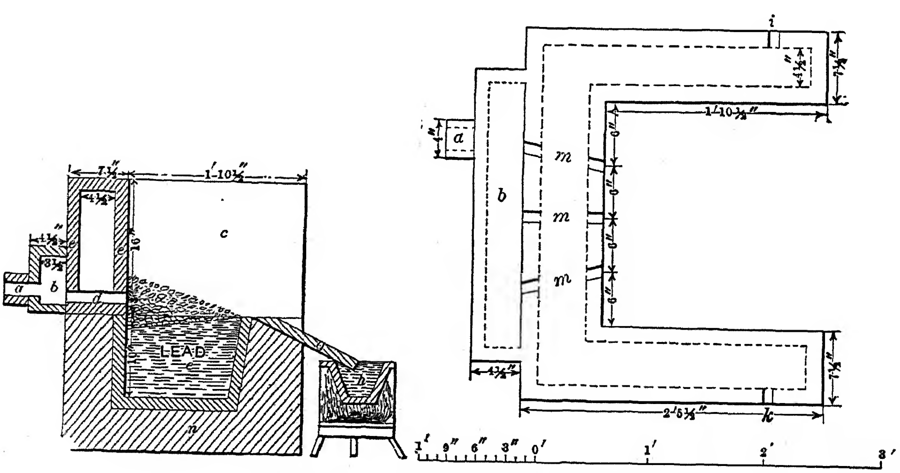


FIG. 243.—Roast stalls for sulphide ore.



FIGS. 244 and 245.—Lead ore-hearth.

SHAFT FURNACES.—A shaft furnace differs in construction from a hearth furnace in that the elevation area is at least as large and usually larger than the plan area, and that the furnace is entirely enclosed by permanent walls. Shaft furnaces have a larger cubical capacity than hearth furnaces and work

continuously. They are kept filled with charge which consists of ore, flux and fuel; the material to be treated is fed at intervals at the top and a corresponding amount of product, which has undergone metallurgical changes during its descent through the shaft, is withdrawn at the bottom. There is a column of solid charge passing slowly downward through the shaft; at the same time there is a quickly ascending gas current, because the fuel charged with the ore and flux burns at or near the bottom of the furnace where the necessary air is admitted, and the products of combustion rise in the furnace. In doing this, they give up their heat to the descending charge and act chemically upon it before leaving the furnace at the top. Hence the thermal efficiency of a shaft furnace is large, reaching 30 to 50 per cent.

Shaft furnaces working with natural draft burn little fuel per unit of time; they are mainly low-temperature furnaces suited for burning limestone, calcining oxide (zinc, iron) and roasting some sulphide (quicksilver) ores, Fig. 246. In Spain[1] they used to be operated for smelting readily fusible oxide-lead ores. On account of the prevailing low temperature, the furnaces are generally cylindrical, as this is the most desirable form as regards uniform combustion of fuel and descent-of charge; further, offering the smallest surface for a given area, the loss of heat by radiation is reduced to the lowest figure. Shaft furnaces with induced draft have found occasional application in foundry work (Herbertz cupola) where only a low-pressure blast is required to obtain the moderate temperatures necessary to melt pig-iron. Draft-cupolas generally are also cylindrical.

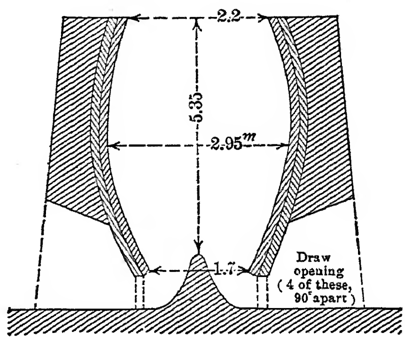


FIG. 246.—Shaft furnace for calcining oxide zinc ore.

Shaft furnaces with blast, blast-furnaces, Fig. 247, are used almost exclusively for smelting, as natural or induced draft does not furnish sufficient air per unit time; the necessary volume of air is forced under pressure into the furnace through tuyeres, placed in the side near the bottom, and burns a large amount of solid fuel per unit of time and area, and thus generates the high temperature required for smelting. The larger the ratio of fuel burnt per unit time and unit area the higher will be the temperature. The tuyere section of the furnace is circular, as with shaft furnaces run with natural or induced draft, unless the pressure required for the blast to penetrate an area of given diameter is too great to be suited for the ore under treatment; in such cases the section should be oblong; the circular iron blast-furnace and the oblong lead or copper blast-furnace furnish characteristic examples.

The cross-section area of the furnace at the tuyere- or smelting-zone, should be smaller than any other higher up, as the combustion has to be rapid, and as with the disappearance of the solid fuel and the shrinkage of the rest of the

[1] Thum, *Berg. Hüttenm. Z.*, 1875, xxxiv, 389.

charge, the volume required is smaller. The enlarged cross-section area above the tuyere level causes a retardation of the ascending gas current and with this a better cooling of the gases accompanied by a more effective chemical action upon the descending charge-column. Blast-furnaces, as a rule, are higher than shaft furnaces run by natural draft, as it takes a longer charge-column to absorb the greater heat, and as the chemical changes in the charge, usually of a heat-absorbing character, require considerable time.

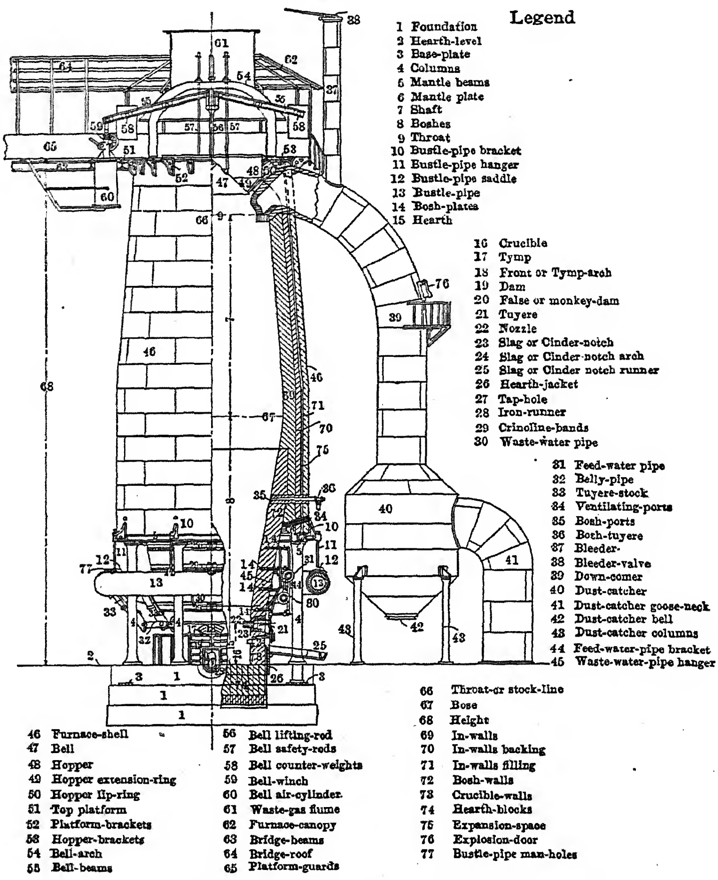


FIG. 247.—Blast-furnace for smelting iron ore.

REVERBERATORY FURNACES.—In the reverberatory furnace, Fig. 248, a flame, produced by the combustion of solid, liquid or gaseous fuel, enters the laboratory or hearth holding the charge; it strikes the roof (arch), is more or less deflected (reverberated) downward and thus heats the charge either directly by contact or indirectly by radiation. The flame and its products of combustion heat also the fire-walls which exert their effect upon the charge by radiation. In direct-firing, the solid fuel is usually burnt in a fireplace separated from the hearth by a low wall, the fire-bridge. Fuel dust (§ 107) forms the only exception

to this general statement, as it is burnt in the laboratory proper as is the case with liquid and gaseous fuels. The gases from the laboratory pass off either directly into a chimney or after they have given up some of their heat to apparatus for raising steam, preheating air and producer gas, drying and warming refractory vessels, etc. The forms of the fireplaces for solid fuels and of the hearths, as well as the ratios of the respective areas differ with the work that is to be done, and this embraces practically all pyrometallurgical operations.

In direct-firing, the fireplace is supplied with the necessary air either by natural or by forced draft; the former method is better suited for low-, the latter for high-temperature furnaces. The manner of admitting air is governed, however, not alone by the character of the work, but depends to a large extent upon the character of the fuel. As a rule, a finely divided charge is treated more successfully in a reverberatory furnace than in a shaft furnace, as the latter requires a coarse charge to do its best work. The thermal efficiency of a reverberatory furnace will depend, other things being equal, upon the temperature at which the products of combustion pass off into the open. If they do this immediately upon leaving the hearth, the efficiency is 10 to 15 per cent.; if they first give up the heat, in excess over that necessary to furnish effective draft, to regenerators or recuperators, this figure rises to 20 or 30 per cent.

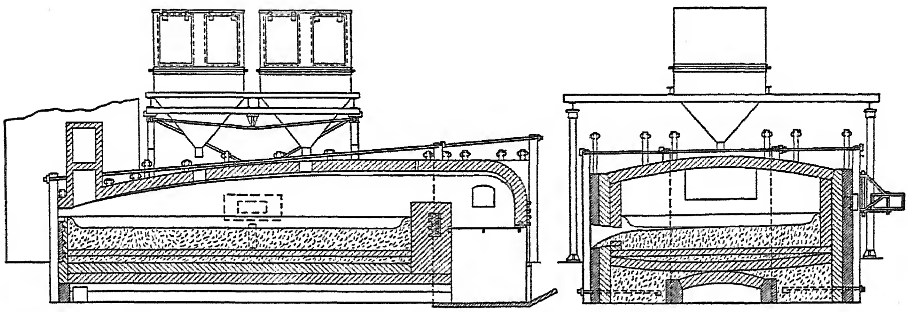


FIG. 248.—Reverberatory matting furnace.

CLOSED-VESSEL FURNACES.—Closed-vessel furnaces are used only when it is essential that the charge and its products are to be protected from the contact with solid fuel or flame, or when vapors are to be condensed and collected. The vessels are made of clay or iron. The kind of furnace chosen depends upon the form of the closed vessel and the manner of heating best suited. Thus there are crucible-, tube-, muffle-furnaces, etc. The ordinary crucible- (wind- or pot-) furnace,[1] Fig. 249, fired with charcoal, coke or anthracite and run with natural draft, is common in the laboratory for making assays, in silver and gold mills for making fusions, in the brass foundry, etc. In some works the natural draft is replaced by forced draft.

[1] Schmatolla, E., "Die Tiegeloefen," Seydel, Berlin, 1901.
Irresberger, *Stahl u. Eisen*, 1904, XXIV, 169, 253.

Gas- and oil-fired crucible furnaces are not uncommon in laboratories and industrial establishments; they are used almost solely in making crucible-steel. Coal- and gas-fired retort-furnaces are characteristic for the Belgian process of smelting zinc ores (Fig. 706, Converse-de Saulles) or for distilling and sublim-

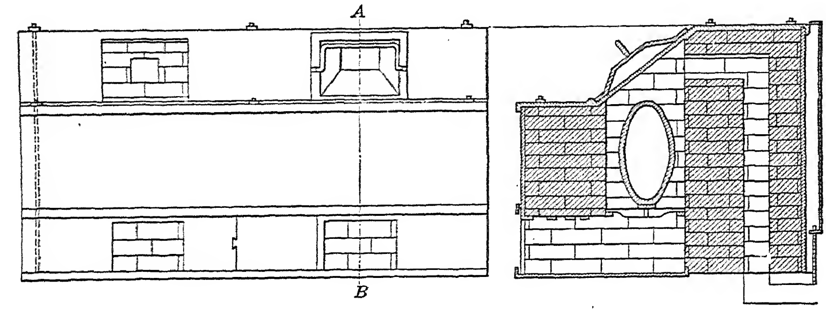


FIG. 249.—Crucible furnace.

ing; tube-furnaces serve for liquating antimony sulphide; muffle furnaces for coking, roasting, for making cement steel, malleable iron, etc. From the examples quoted it is seen that a closed-vessel furnace can be heated with any kind of fuel, but as the heat has to penetrate the wall of the closed vessel before

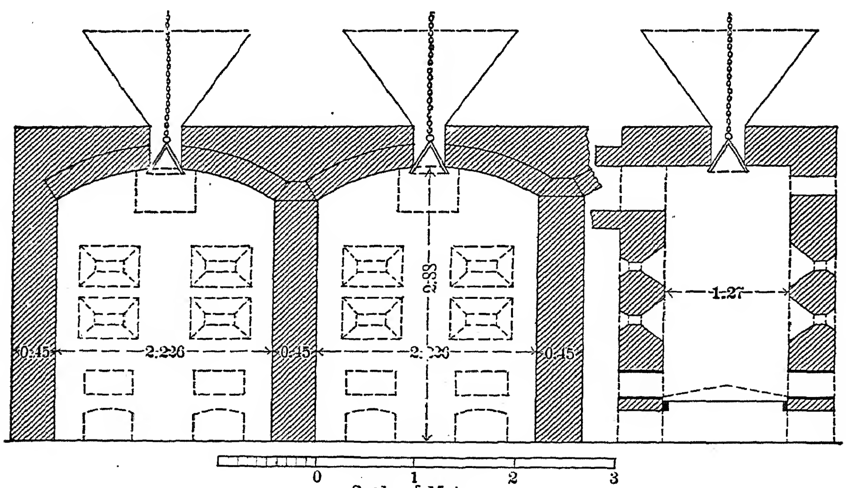


FIG. 250.—Roasting kiln.

it can act upon the charge, the thermal efficiency will always be low, viz., 2 to 5 per cent. The hot waste gases of the closed-vessel furnace can be and are utilized for superheating the air necessary for combustion, but the efficiency in heating will always remain low.

KILNS AND CONVERTERS.—In the operations carried on in the preceding furnace, the charge was of such a character that the changes affected con-

sumed more heat than they brought so that the deficiency had to be made up by burning carbonaceous fuel. The heat set free in the oxidation of some charges is sufficient to carry through the process without outside assistance as

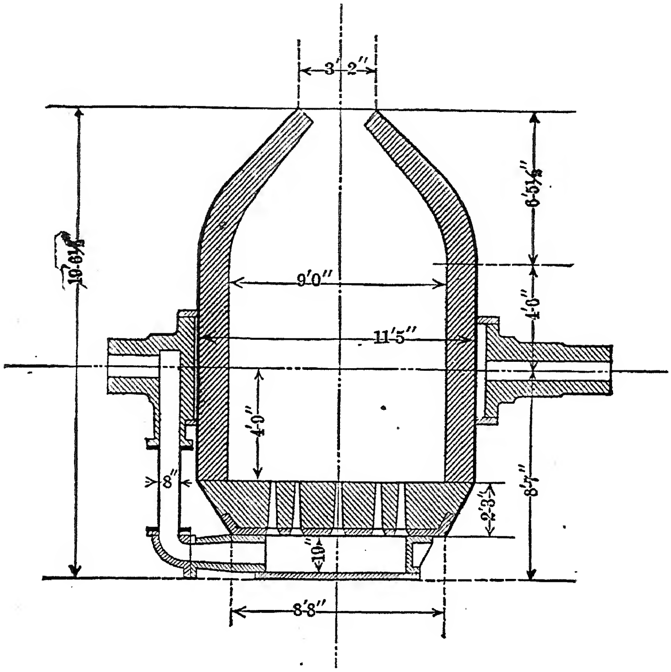


FIG. 251.

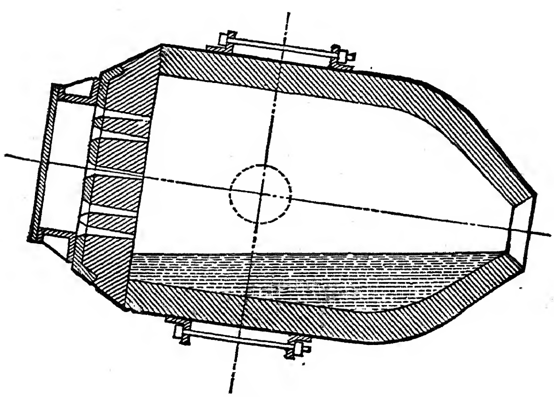


FIG. 252.

FIGS. 251 and 252.—Concentric bessemer converter.

long as the furnace is sufficiently preheated to start the process and an excessive loss of heat by waste gases and radiation is avoided. Such a charge can be solid or liquid. Furnaces treating solid charges are the roasting kilns, working continuously, and the beehive coking oven, working intermittently.

Roasting kilns, Fig. 250, are low shaft-like furnaces which are worked in a manner similar to that of shaft furnaces having natural draft. Their efficiency is lower than that of a shaft furnace, as the gases pass off at a comparatively high temperature. The beehive coking oven, Figs. 136 and 139, is a peculiar

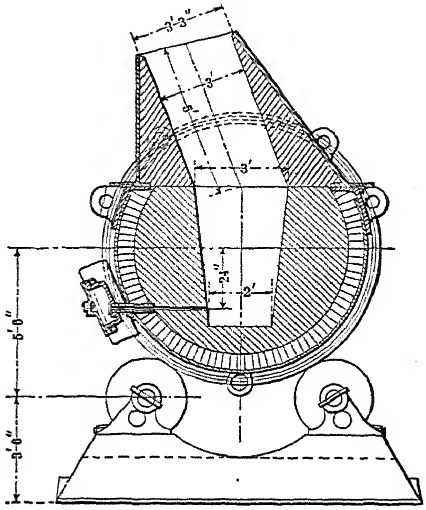


FIG. 253.—Shelby barrel converter.

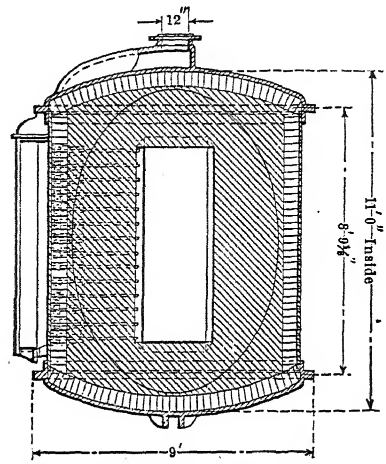


FIG. 254.—Shelby barrel converter.

furnace still used in the United States and England. Its thermal efficiency is lower than that of the kiln.

Furnaces treating liquid charges are called *converters*. They are upright or horizontal cylindrical vessels used in the conversion of pig iron into steel

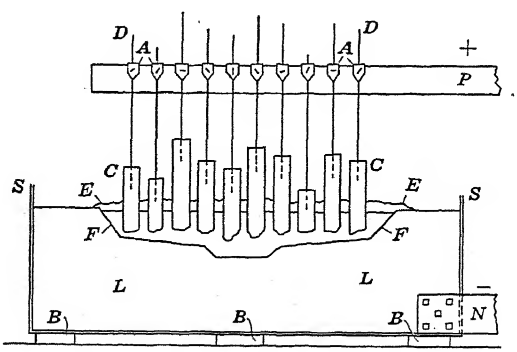


FIG. 255.—Hall aluminum furnace.

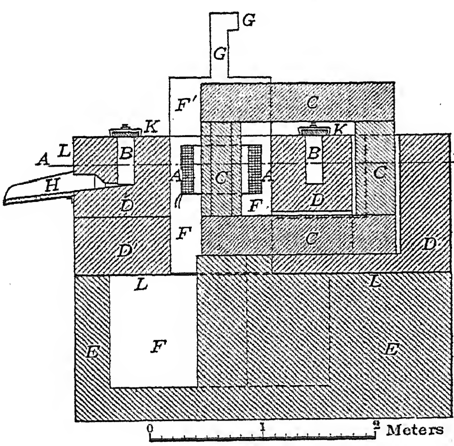
(Figs. 251 and 252), copper matte into metallic copper (Figs. 253 and 254), and low-grade nickel-copper matte into a product of higher grade. Air under pressure is forced through the liquid charge and oxidizes part of the constituents with such quickness that the temperatures attained are sufficiently high to keep the steel, or the metallic copper, or the concentrated nickel-

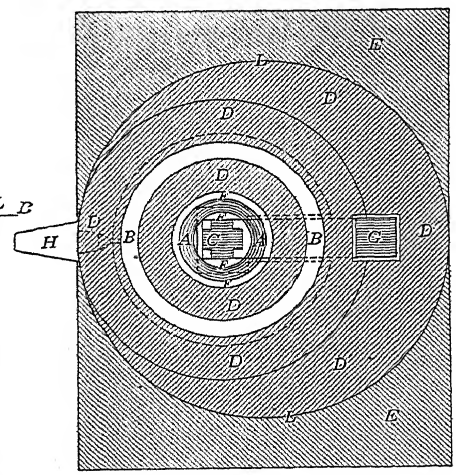
copper matte liquefied so that they can be poured. The thermal efficiency is 50 to 60 per cent.

Blast-roasting apparatus are taken up in §188.

ELECTRIC FURNACES.[1]—These have many advantages over furnaces heated with carbon, viz.: (1) a higher temperature can be obtained (3,600 *vs.* 2,000° C.); (2) the heat can be developed in a small space just where it is wanted, which is equivalent to a small furnace and a correspondingly small loss of heat by radiation; (3) the charge need not be exposed to the influences of the gaseous products of combustion; (4) the temperature, in fact the operation as a whole, is under perfect control; (5) the expense of running lasts only as long as the current is doing active work.

The disadvantages are few: the substance to be heated cannot always be kept out of contact with C; the cost of the current makes electric heating often more expensive than ordinary firing. On account of the many advantages, especially of those given under headings 2 and 4, the thermal efficiency of electric furnaces[2] is high, ranging in melting from 60 to 80 per cent.





Section A-B

FIGS. 256 and 257.—Kjellin induction furnace.

Fig. 255 gives a longitudinal section of the Hall aluminum furnace as a representative of direct resistance-heating; in it S is the iron shell; N, negative conductor; L, carbon lining and negative electrode; P, positive conductor; A, screw clamps; D, copper rods; C, carbon cylinders and positive electrode; F, fused bath; E, cover of alumina and carbon dust.

Figs. 256 and 257 represent the Kjellin induction furnace. In this A is the primary coil of insulated copper wire wound about one leg of the magnetic circuit C, made up of thin insulated sheets of soft iron; B, the secondary formed

[1] Richards, Aluminum World, 1899, V, 105.

[2] Richards, *Tr. Am. Electrochem. Soc.*, 1902, II, 51; *Electrochem. Met. Ind.*, 1905, III, 303.

by the metal charge contained in the annular groove B; L the sheet-iron casing lined with fire-brick D and magnesite or silica brick; E, the brick foundation; F, an air space provided with hood F' and chimney G; K, covers; H, tapping

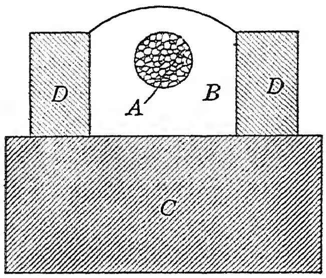


FIG. 258.—Acheson carborundum furnace.

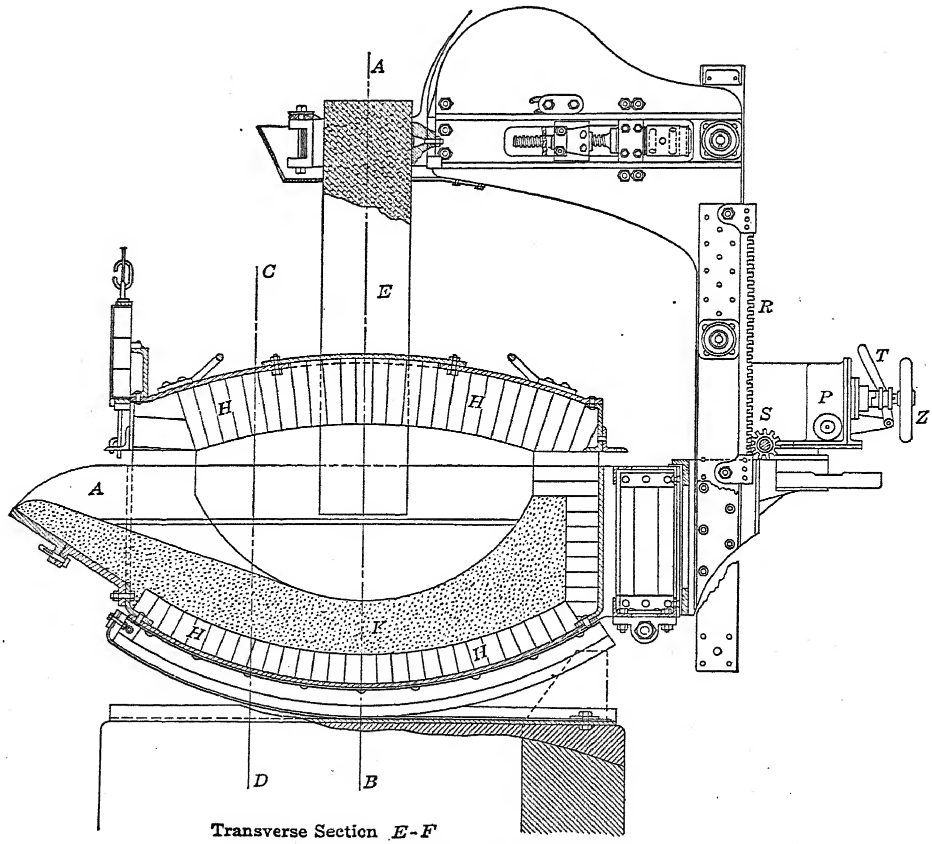


FIG. 259.—Héroult electric furnace.

spout. The current passes through coils A, excites a varying magnetic flux in the circuit C, and this induces an alternating current in the content of the an-

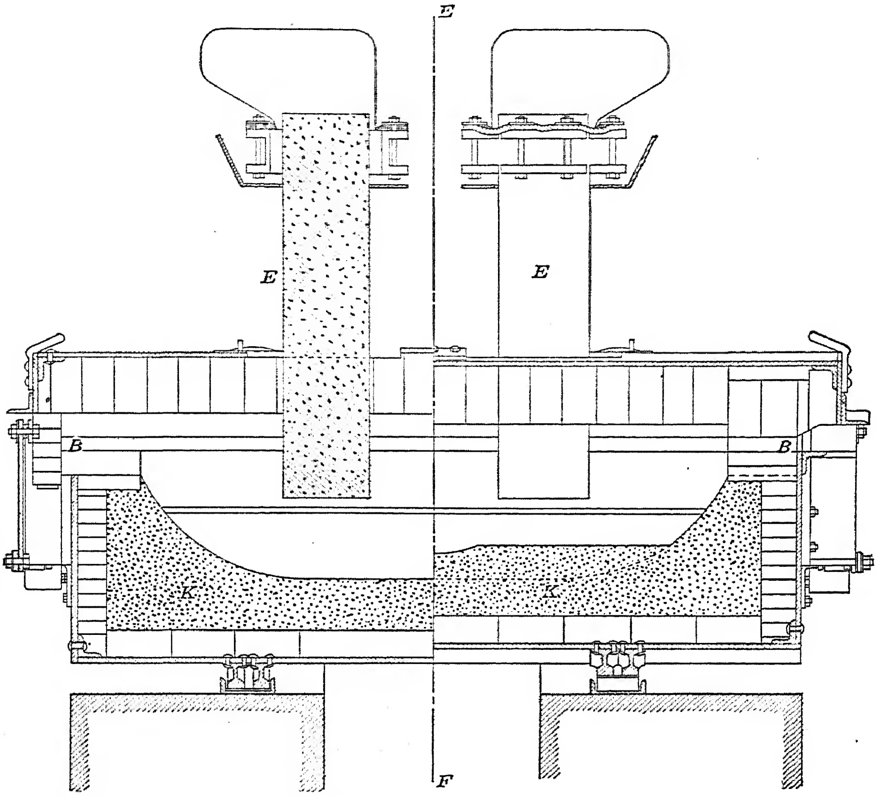


FIG. 260.—Héroult electric furnace.

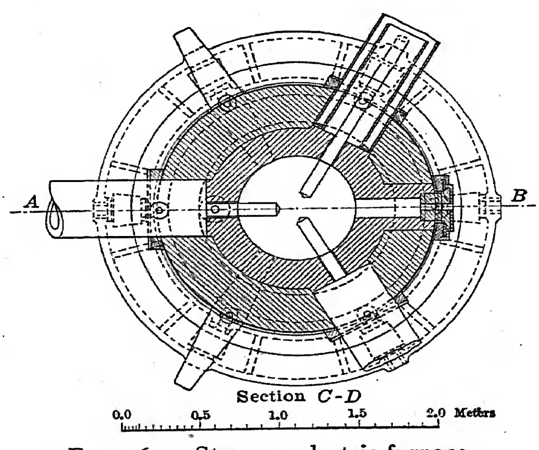


FIG. 261.—Stassano electric furnace.

nular groove B and melts it. The arrangement is similar to that of a stepdown transformer having a large number of primary turns (A) and a single secondary turn (B).

Indirect resistance-heating is illustrated by the cross-section of the Acheson carborundum furnace, Fig. 258, in which A is the core of powdered coke forming the heating resistance, B the charge, C the foundation, D the side-walls laid dry.

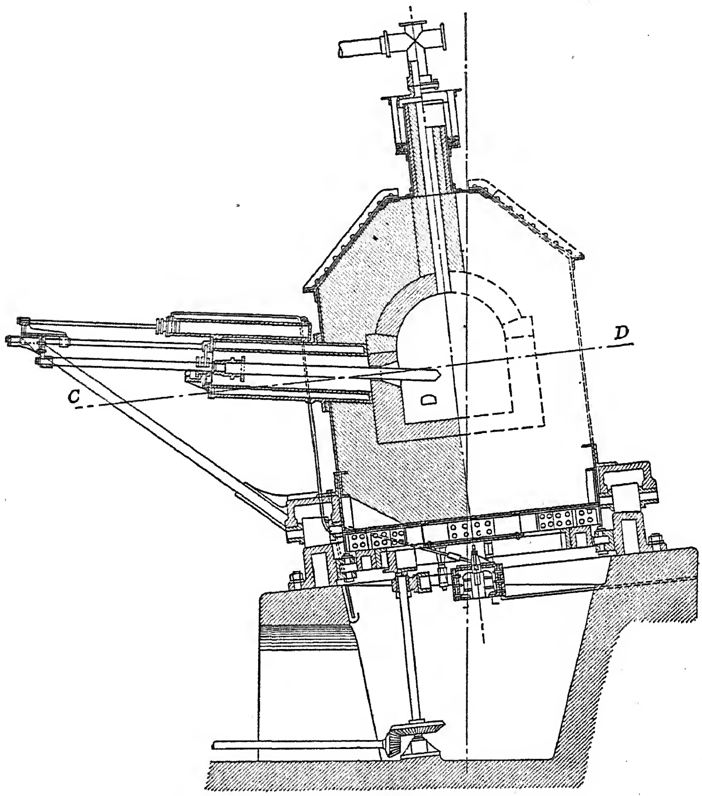


FIG. 261*a*.—Stassano electric furnace.

Direct arc-heating, the original form of the Siemens crucible furnace, Fig. 323, may serve as example.

Combined resistance- and arc-heating is best characterized by the Héroult steel furnace, Figs. 259 and 260, in which H is magnesite brick built into a steel-plate shell carrying the basic lining K; A, pouring spout; E, suspended electrode, of which there are two; H', silica brick roof; P, motor for driving automatic regulator; T, lever for throwing motor P out of action; Z, hand-wheel operating pinion S' for regulating electrodes by hand; R, rack with gearing S for raising and lowering electrode.

Indirect arc-heating has found its chief application in the Stassano steel furnace, Figs. 261 and 261*a*. This is inclined to the vertical and rotated from

below; the lining is of magnesite brick; there are three carbon electrodes entering the furnace through water-jacketed openings; the electrodes are regulated through hydraulic cylinders and receive the current through cables and brushes from a stationary ring. The charge is fed from an opening in the top (not shown), slag and metal are tapped through openings near the bottom, the forms of which are shown.

With the exception of the induction furnace all electrical furnaces must have carbon electrodes[1]. Carbon is chosen, as it fills best the requirements of being refractory, a good transmitter of heat and a poor conductor of electricity, does not contaminate, and acts chemically only in its reducing capacity.

The raw materials used for furnishing the C have been charcoal, coke, anthracite, soot, gas-carbon and petroleum-coke. The petroleum-coke (ash 3.09, V.M. 6.55, S 1.44, Cl 2.36), used in the early stages of manufacture, has to be heated in a closed vessel to drive off the 5 to 8 per cent. V.M. it contains. Used alone, it did not work well, as the electrodes crumbled. It was replaced by gas-carbon (ash 1.77, V.M. 0.46, S 0.73 per cent.) which furnished better electrodes, but suffered from the difficulty that the material collected from numerous localities was irregular in composition. Only when anthracite came into use was there made real progress in their manufacture, so that to-day electrodes 24 in. sq. or round 24 in. diam., and 7 ft. long are made, while a few years ago 15 in. sq. and 5 ft. long were deemed satisfactory.

At present, mixtures of the raw materials mentioned are used, of which anthracite forms by far the largest part. Anthracite mixed with other forms of C is crushed to from 0.5 to 3 mm. size; from 15 to 25 per cent. of a mixture (ash 0.15, V.M. 43.09, F.C, 56.85 per cent.) of pitch and dehydrated tar is added, and the whole worked in a steam-heated (70 to 90° C.) pug-mill. The prepared mass is rammed into blocks weighing as much as 25,000 lb. in a single-stamp hammer machine. These blocks are transferred to a horizontal hydraulic compression machine and subjected to a pressure of from 35 to 40 tons per sq. ft. The electrodes are now baked in a gas-fired ring-oven (30 chambers each holding six electrodes). The electrodes are lowered in fire-clay vessels, the unoccupied spaces filled with C, and the batch baked

[1] Zellner, J., "Die Künstlichen Kohlen für electrische und electrochemische Zwecke; ihre Prüfung und Herstellung," Springer, Berlin, 1903.

Louis, *Rev. Mét. Extr.*, 1911, VIII, 320.

Anon, *Stahl u. Eisen*, 1912, XXXII, 1857, incl. bibliography.

Stedman, *Canad. Min. J.*, 1912, XXXIII, 339.

Collins, Graphite Electrodes, *Electrochem. Ind.*, 1904, II, 227.

Fitzgerald-Forssell, "*Carbons for Electrometallurgy*," *Tr. Am. Electrochem. Soc.* 1907, XI, 37.

Hansen, *Met. Chem. Eng.*, 1909, VII, 389.

Hering, *op. cit.*, 1909, VII, 401, 442, 473; 1910, VIII, 129, 188, 238, 391.

Turnbull, *op. cit.*, 1912, X, 289.

Rodenhauser-Schoenawa -vom Baur, "*Electric Furnaces in the Iron and Steel Industry*," Philadelphia Book Co., Philadelphia, 1913.

6 to 10 days for small sizes, and from 10 to 16 days with large sizes, the chamber being brought gradually to 1,200° C. and then cooled slowly. A good electrode has the following properties: apparent specific gravity[1] 1.50 to 1.55; specific electric resistance (range 4 to 500 sq. in.) 40 to 300 ohm.; temperature coefficient of resistance (range 25 to 900° C.) 0.000318; specific heat at 100° C., 0.18 to 0.22; compressive strength 3,300 to 6,000 lb. per sq. in.; transverse strength, 730 to 1,100 lb. per sq. in.; expansion (range 0 to 700° C.), 0.26 per cent. at 0° C.; heat conductivity of 1 c.c. with fall of temperature of 130 to 20° C., 0.24 cal. per hr.; ash 2.5 to 3.0 per cent.; P 0.45 to 0.53 per cent.; S 0.90 to 1.10 per cent.[2]

If the electrode is to be graphitized and its electric conductivity increased from three to four times, it is placed in a closed vessel, and heated electrically with a current of from 200 to 250 amp. per sq. in. cross-section. At the resulting high temperature, oxides (Al_2O_3, Fe_2O_3, Mn_2O_3, SiO_2) are converted into carbides which are dissolved in the electrode and dissociated, setting free graphite.

The electric connection is never fixed with large electrodes, as the loss in energy (8 to 12 per cent. with 50 volt) is too great; with small electrodes, a head socket is firmly attached. Large electrodes always have a movable connection in the form of a copper or bronze clamp, generally water-cooled, a few layers of copper gauze being placed between the clamp and smooth hard surface of the electrode to form a good contact. A bad contact will cause the carbon to glow at the contact; the evil grows worse, as the electric conductivity of C increases with the temperature. The contact surface is made sufficiently large so as not to exceed a current density of 130 amp. per sq. in.; the cross-section of the clamp can have 1,000 amp. per sq. in. if water-cooled, 600 if not.[3] The clamp is placed as close to the roof of the furnace as the heat will permit. The electrode is raised and lowered mechanically (Thury regulator); new electrodes are attached to old ones by nipples of electrode-material molded with a screw-thread; with a current density of 30 amp. per sq. in. there is a loss of 2 to 2.5 volts.

At present electrodes are made of such a cross-section that they take a current of 30 to 35 amp. per sq. in. section when they remain cool. Hansen[4] advises 50 amp. for carbon and 150 amp. for graphite electrodes. His experiments have shown that the restivity C : Gr. = 2.25 : 1, and the heat conductivity C : Gr. = 1 : 10. The life of an electrode is shortened through disintegration by the current, chemical action of C upon charge, solution of C by charge, oxidation by air. In steel refining with a current of 40 volts, the wear is 1/4 to 1/2 in. per hr., in the production of pig iron 1/16 in. per hr., and in the production of Al, 1/8 oz. per lb. Al produced.

[1] Fitzgerald, *Tr. Am. Electrochem. Soc.*, 1902, II, 43.
[2] *Stahl u. Eisen, loc. cit.*
[3] Louis, *loc. cit.*
[4] *Loc. cit.*

168. The Efficiency of Furnaces.[1]—In general, efficiency is the ratio of useful work done to energy expended. With furnaces, thermal efficiency, which can be either absolute or relative, is of leading importance. Absolute thermal efficiency (E_a) is the ratio of the net thermal effect produced (E_n) to the calorific power of the fuel (P) or $E_a = E_n : P$. It is of general application, as it gives absolute data. Relative efficiency (E_r) is the ratio of the net thermal effect produced (E_n) to the number of calories actually realized (P_a) or $E_r = E_n : P_a$. Its application is limited, as the factor a varies with each individual case.

In order to obtain the data necessary for determining the thermal efficiency, it is often convenient to balance the heat-producing with the heat-absorbing factors that enter into an operation. In the accompanying heat balance the main factors are arranged in debit and credit columns; each main factor may be the sum of several minor ones.

HEAT BALANCE

Debit	Credit
Heat-producing Factors per Unit of A	Heat-absorbing Factors per Unit of A
1. Heat generated by combustion of fuel.	1. Heat carried off in gases and fumes.
2. Sensible heat entering the furnace.	2. Heat carried off in solid or liquid products.
3. Exothermic reactions of charge.	3. Endothermic reactions of charge.
	4. Heat lost by radiation and conduction.

EXAMPLES.—I. In an annealing furnace with cast-iron chamber the fuel consumption per 24 hr. was at the rate of 27 kg. bituminous coal (calorific power 7,000 Cal.) for bringing 100 kg. German silver (spec. heat, 0.096) from 15 to 600° C.

Debit: 27 kg. coal $\times$ 7,000 = 189,000 Cal.

Credit: 100 kg. German silver $\times$ rise in temperature (600 − 15) $\times$ spec. heat of G. S. 0.096 = 5,616 Cal.

Absolute efficiency: $\frac{5,616}{189,000} = 0.02972 = 2.97$ per cent.

If the charge is to be kept at 600° C. for an indefinite time in order to cause certain physical changes to take place, no absolute thermal efficiency can be calculated, as the alloy heated to 600° C. does not absorb any more heat. Here the relative efficiency of two or more furnaces comes into play. The comparison of these can be based on the number of calories expended to keep either a given space (cubic content of furnace) or a given weight of material at a given temperature for unit time.

II. In melting 100 kg. silver (heat in 1 kg. melted silver at melting-point =

[1] Grüner, *Ann. Min.*, 1875, VIII, 173.
Dürre, *Oest. Zt. Berg. Hüttenw.*, 1879, XXVII, 176.
Ledebur, *op. cit.*, p. 44.
Richards, "Metallurgical Calculations," 1906, I, 76; *J. Frankl. Inst.*, 1907, CLXIII, 129.
Pomerantzeff, *Rev. Mét.*, 1911, VIII, 127; *Stahl u. Eisen*, 1912, XXXII, 197.
Landis, *Met. Chem. Eng.*, 1910, VIII, 520 (Rotary Cement Kiln).

89.15 Cal.) in a crucible there were required by the pot-furnace 30 kg. coke (calorific power 6,500 Cal.).

Debit: $30 \times 6{,}500 = 195{,}000$ Cal.

Credit: $100 \times 89.15 = 8{,}915$ Cal.

Absolute efficiency: $\frac{8{,}915}{195{,}000} = 0.04572 = 4.57$ per cent.

III. Gold-bearing pyrite with 10 per cent. gangue and 48 per cent. S is roasted in a hand reverberatory furnace at the rate of 9 metric tons in 24 hr. with a consumption of 1 metric ton of coal (C 73.57, O 10.14, H 5.14, N 1.24, S 1.86, H_2O 2.55, ash 8.05; calorific power 7,300 Cal.). The gases, escaping from the furnace at a temperature of 200° C., contain SO_2 1.5 and SO_3 0.5 per cent. vol.; the roasted ore, with 98 per cent. of its S eliminated, is withdrawn from the furnace near the fire-bridge at a temperature of 900° C.

Debit:[1]

1. Heat evolved in burning coal: 1,000 kg. $\times 7{,}300 = 7{,}300{,}000$, is the total calorific power. The ashes formed weighed 89.4 kg., and contained 10 per cent. or 8.94 kg. C. The calorific power lost in ashes is $8.94 \times 8{,}100 = 72{,}414$ Cal. Therefore the actual heat generated by the combustion of the coal is $7{,}300{,}000 - 72{,}414 = 7{,}227{,}586$ Cal.

2. Heat evolved in burning sulphur: the raw ore contains $9{,}000 \times 0.48 = 4{,}320$ kg. S; of this 98 per cent. or $4{,}320 \times 0.98 = 4{,}234$ kg. have been burnt; 1/4 of 1,059 kg. was burnt to SO_3 and 3/4 or 3,175 kg. to SO_2. The heats evolved are $1{,}059 \times 2{,}872 = 3{,}041{,}448$ Cal. and $3{,}175 \times 2{,}164 = 6{,}870{,}700$ Cal., or a total of 9,915,000 Cal.

3. Heat evolved in oxidizing iron: the S remaining in the ore will be assumed to be in combination with Fe as FeS. There was driven off 98 per cent. of the total S in the ore, therefore there remained behind 2 per cent. or $4{,}320 \times 0.02 = 86.4$ kg. Combining these with iron, $S : Fe = 32 : 56 = 86.4 : x$, gives $x = 151.2$ kg. Fe. The iron in the ore, present as FeS_2, weighs $S_2 : Fe = 64 : 56 = 4{,}320 : y$, $y = 3{,}780$ kg.; hence $3{,}780 - 151.2 = 3{,}628.8$ kg. Fe were oxidized. The heat of formation of Fe_2O_3 per kg. Fe $= 1{,}746$ Cal., hence the heat generated is $3{,}628.8 \times 1{,}746 = 6{,}335{,}885$ Cal.

The total heat generated from the three sources is $7{,}227{,}586 + 9{,}915{,}000 + 6{,}335{,}885 = 23{,}478{,}461$ Cal.

Before taking up the credit column, it will be necessary to calculate the total volume of gas. There were oxidized from the pyrite 4,234 kg. S and from the coal 18.6 kg., making a total of 4,252.6 kg. As 1 cbm. SO_2 or SO_3 contains 1.44 kg. S, the weight of S in 1 cbm. flue gas $= (0.015 + 0.005) \times 1.44$ kg., and the total volume of gas $= \frac{\text{wt. of S burned}}{\text{wt. of S in 1 cbm. flue gas}} = \frac{4{,}252.6}{(0.015 \times 0.005)1.44} =$ 147,639 cbm. Of this total, $148{,}700 \times 0.015 = 2{,}215$ cbm. is SO_2 and $148{,}700 \times 0.005 = 738$ cbm. is SO_3. There were oxidized to CO_2 from 1,000 kg. coal,

[1] While it is not necessary to cast a heat balance in order to obtain the data required to calculate the efficiency, it has, nevertheless, been carried through, as it brings out many interesting facts.

735.7−8.9 (in ashes) = 726.8 kg. C, which gives 1,346 cbm. CO_2, as 1 cbm. contains 0.54 kg. C.

The 51.4 kg. H in 1,000 kg. coal required $H_2 : O = 2 : 16 = 51.4 : x$, $x = 411.2$ kg. O. There are present in the coal 101.4 kg. O, hence 411.2−101.4 = 309.8 kg. O have to be furnished by the air. The 51.4 kg. H form 462.6 kg. H_2O which, added to the 25.5 kg. H_2O in 1,000 kg. coal, gives 488.1 kg. H_2O existing as water vapor in the gas, or 602.6 cbm.

Nitrogen in 1,000 kg. coal: The coal contains $1{,}000 \times 0.0124 = 12.4$ kg. N = $\frac{12.4}{0.09 \times 14} = 9.8$ cbm. The N accompanying the O is arrived at as follows:

The oxidation of FeS_2 takes place according to: $2FeS_2 + 12O = Fe_2O_3 + 3SO_2 + 1SO_3$, hence $4S : 6O_2 = 4 \times 32 : 12 \times 16 = 2 : 3$, or 4,234 kg. total $S : O = 2 : 3$, gives $O = 6{,}351$ kg.

The oxidation of the 729 kg. C in the coal to CO_2 according to $C : O_2 = 12 : 32 = 726.8 : x$, requires $O = 1{,}938$ kg. The oxidation of the 18.6 kg. S in the coal to SO_2 according to $S : O_2 = 32 : 32 = 18.6 : y$, requires $O = 18.6$ kg. The total $O = 309.8 + 6{,}351 + 1{,}938 + 18.6 = 8{,}617.4$ kg., and this is accompanied by $8{,}617.4 \times \frac{10}{3} = 28{,}725$ kg. N; $\frac{28{,}725}{1.26} = 22{,}798$ cbm. N. Adding the items gives H_2O 602.6 + N (from air) 22,798 + CO_2 1,346 + SO_2 2,215 + SO_3 738 + N 9.8 (from coal) = 27,709.4 cbm. gas. The total volume of gas formed was found, however, to be 147,639 cbm. showing that 147,639 − 27,709 = 119,930 cbm. air or 4.2 times the theoretical amount of air entered as excess air. This figure is reasonable considering an oxidizing roast with the aim to eliminate all the S.

Credit:

1. Heat in waste gases:

In H_2O vapor...............	$602.6(0.34+0.00015\times200)200 =$	38,650 Cal.
In N and excess air $(22{,}798+9.8+119{,}930)$	$(0.303+0.000027\times200)200 =$	8,804,068 Cal.
In CO_2....	$1{,}346(0.37+0.00022\times200)200 =$	111,449 Cal.
In SO_2......................	$2{,}231(0.36+0.0003\times200)200 =$	186,060 Cal.
In SO_3......................	$743(0.405+0.00017\times200)200 =$	64,796 Cal.
	Total =	9,210,965 Cal.

2. Heat in roasted ore:

3,628.8 kg. Fe = 5,184 kg. Fe_2O_3 with	$5{,}184(0.1456+0.000188\times900)900 =$	1,468,731 Cal.
151.2 kg. Fe = 237.6 kg. FeS with	$237.6\times0.1357\times900 =$	29,018 Cal.
900 kg. gangue (SiO_2) with	$900(0.1833+0.000077\times900)900 =$	204,606 Cal.
	Total =	1,702,355 Cal.

3. Heat absorbed in decomposing FeS:[1]

This amounts to 429 Cal. per kg. Fe. As there were decomposed 3,628.8 kg. Fe in the form of pyrite, this calls for

$3{,}628.8\times429$	1,556,755 Cal.

Adding the three items gives:

Heat accounted for	12,470,075 Cal.
Heat lost by radiation $23{,}478{,}461-12{,}470{,}075$	11,008,386 Cal.

$$\text{The absolute thermal efficiency}=\frac{\text{heat in ore}+\text{heat absorbed in decomposing } FeS_2}{\text{total heat generated}}$$

$$=\frac{1{,}702{,}355+1{,}556{,}755}{23{,}478{,}461}=13.88 \text{ per cent.}$$

IV. An iron blast-furnace produces mottled pig iron with C 3.5, Si 1.5 and Mn 0.5 per cent. from hematite using 1 metric ton coke per metric ton of pig; the coke contains C 90 per cent. and has a calorific power of 7,000 Cal.

Debit:

C in coke: $1{,}000\times0.90=900$ kg.; C in pig iron: $1{,}000\times0.035=35$ kg., hence the C actually burnt $900-35=865$ kg. developed $7{,}000\times\frac{865}{900}=6{,}728$ Cal., and the heat from 1,000 kg. amounted to 6,728,000 Cal.

Credit:

1. Reduction of Si: There were reduced $1{,}000\times0.015=15$ kg. Si. With a heat of reduction from SiO_2 per kg. S of 7,600 Cal. the heat absorbed is $15\times7{,}600=114{,}000$ Cal.

2. Reduction of Mn: There were reduced $1{,}000\times0.005=50$ kg. Mn. With a heat of reduction from MnO per kg. Mn of 1,653 Cal., the heat absorbed is $5\times1{,}653=8{,}265$ Cal.

3. Reduction of Fe: There were reduced $1{,}000-(35C+15Si+5Mn)=900$ kg. Fe. With a heat of reduction from Fe_2O_3 per kg. Fe of 1,746 Cal., the heat absorbed is $945\times1{,}746=1{,}649{,}970$ Cal.

The heat evolved in forming slag is neglected, as it is approximately balanced by the heat absorbed in decomposing fluxes.

The absolute thermal efficiency=

$$\frac{\text{heat usually applied}}{\text{total heat generated}}=\frac{114{,}000+8{,}265+1{,}649{,}970}{6{,}728{,}000}=0.2633=26.33 \text{ per cent.}$$

V. In distilling silver amalgam in an iron retort, the charge of 500 kg. with 20 per cent. Ag is retorted with 2/3 cord =1,250 kg. seasoned wood having a calorific power of 3,200 Cal. The Hg vapors pass off at 400° C., and the finishing temperature is a full cherry red, or 700° C.

Debit:

Heat evolved in the combustion of 1,250 kg. wood is $1{,}250\times3{,}200=3{,}999{,}000$ Cal.

[1] FeS_2 has been replaced for the present by FeS, as the amount of heat required to split FeS_2 into FeS and S has not been determined; all that is known is that it is small. See Richards in Peters "Principles of Copper Smelting," New York, 1907, p. 541.

Credit:

1. Decomposition of amalgam. The heat absorbed per 108 kg. Ag is 2,470 Cal, hence 2,280 Cal. per 100 kg. Ag.

2. Heat in 100 kg. Ag at 700° C. is according to Pionchon's formula $700[0.05758+0.0000044\times700+0.000000006\times700^2]100=4,248$ Cal.

3. Heat required for bringing 400 kg. Hg to boiling-point according to Naccari's formula is

$356[0.03337-0.00000275\times356-0.0000000667\times356^2]400=7,350$ Cal.

4. Heat of vaporization for 400 kg. Hg is $400\times77.5=31,000$ Cal.

5. Heat in 400 kg. Hg vapor at 400° C. is $400\times[0.025\times(400-356)]=440$ Cal.

The absolute thermal efficiency$=\dfrac{2,280+4,248+(7,350+31,000+440)}{3,999,000}=$ 1.11 per cent.

VI. With electrically heated furnaces used for electro-thermic processes the calculations are the same as with carbon-heated furnaces excepting that the current furnishes the necessary heat energy; and 1 kw. sec. $=0.239$ Cal., or 1 kw. day$=0.239\times60\times60\times24=20,650$ Cal.

Beside the thermal efficiency of a metallurgical operation there are other factors which go to make up the total or economic efficiency. Some of these are: the costs of installation including the ground occupied and the necessary housing, of power, of repair, of depreciation, of fuel, of labor, of superintendence, of time required to treat a unit.

169. Construction of Furnaces.—While the forms of furnaces and the materials of which they are built vary more or less with the processes that are to be carried out in them; every furnace must meet some general requirements, if it is to do its work satisfactorily. A few of these are:

1. Support.—Parts of a furnace exposed to different physical and chemical influences ought to be independent of one another. In a modern blast-furnace for smelting iron ore or lead ore, the bosh-walls do not carry the shaft-walls as of old, but they are supported through a mantle- or carrier-plate by hollow casttion columns. In a reverberatory smelting furnace, *e.g.*, for copper, the fireplace is always independent of the hearth.

2. Refractory Material.—Refractory materials ought to be adapted to the physical and chemical wear they are to resist. A basic open-hearth furnace has a hearth of burnt dolomite, a roof of silica brick, and the joint where the two meet is made of chrome brick, each suited for its special purpose. In a copper reverberatory smelting furnace the hearth is made of siliceous sand, the roof of silica brick, and the fireplace is lined with fire-brick. The inner parts of a furnace only are built of refractory materials, as erecting an entire furnace in this way would be too expensive. The inner fire-brick wall is therefore usually enclosed by red brick or first by a second-grade fire-brick. Generally an expansion space filled with sand, ashes, gravel, loam, is left between the inner and outer walls. Different grades of fire-brick will have to be used in parts exposed mainly to mechanical wear or to slagging or to corrosive gases in addition to high temperatures.

3. Water Cooling.—Parts of furnace exposed to high temperatures and slagging ought to be made thin and be water-cooled; parts exposed to low temperatures ought to be thick so as to diminish the loss of heat by radiation. Thus in a blast-furnace, the bosh walls are thin and water-cooled or water-jacketed, while the shaft walls are thick.

4. Reinforcement.—A furnace ought to be well ironed to counteract the expansive force of the walls when heated. With some parts iron sheathing (shaft of iron blast-furnace) is sufficient; others require encasing with heavy cast-iron or steel plates (crucible of iron blast-furnace). Oblong furnaces usually are strengthened by vertical iron or steel posts (buckstays) held by iron tie-rods.

5. Radiation.—The different parts of a furnace ought to be assembled in such a way as to reduce the loss of heat by radiation to a minimum. Thus shaft furnaces are made circular whenever this is permissible, as this form furnishes the largest cross-section for the smallest perimeter.

6. Foundation.—A solid foundation is a prime requisite for furnaces as for other structures. The requirements for a blast-furnace or a chimney are much more rigid than those for a reverberatory furnace which covers a larger area. Solid rock is, of course, the best foundation and next best is hard-pan (a mixture of gravel, clay and sand). An artificial foundation may be made of cross-spiked planks or of cement concrete depending upon the weight of the superstructure. In all cases of a built-up foundation, it will be necessary to make provision for drainage and to start masonry work well below the frost-line. Foundations are usually built of undressed rock, the crevices are filled with spalls, and the whole is grouted with lime-mortar reinforced by cement.

7. Electric Furnaces.—The general requirements electric furnaces have to fulfil are simple as compared with those of carbon-heated furnaces, as the charge is usually small and the source of heat is imbedded in the charge itself. Hence only a comparatively simple encasing wall has to be provided to hold the charge together. Special attention has to be given to the carbon electrodes and their electric contacts.

170. Drying.[1]—Drying or the driving off of water is not common in metallurgical operations as a separate process even though the heat absorbed by the evaporation of the moisture may retard to some extent the process that is being carried on. Thus ores with perhaps an average of 5 per cent. H_2O are treated without having been dried unless, *e.g.*, they have to be dry-crushed and then

[1] Hausbrand, E., "Das Trocknen mit Luft und Dampf," Springer, Berlin, 4th ed., 1911, transl. by A. C. Wright, Scott, London, 1912.

Davis, G. E., "A Handbook of Chemical Engineering," Davis Bros., Manchester, 2d ed., 1904, vol. II, pp. 253–266.

Masson, E., "Le séchage," *Rev. Un. Min.*, 1903, IV, 283, 313.

Marr, O., "Das Trocknen und die Trockner," Oldenbourg, Munich, Berlin, 1914.

Grovenor, Calculations for Dryer Design, *Tr. Am. Inst. Chem. Eng.*, 1908, I, 134; *Electrochem. Met. Ind.*, 1910, VIII, 35.

Meade, *op. cit.*, p. 104.

Nagel, Drying Appliances, *Electrochem. Met. Ind.*, 1908, VI, 147.

Porter, *op. cit.*, 1909, VII, 480.

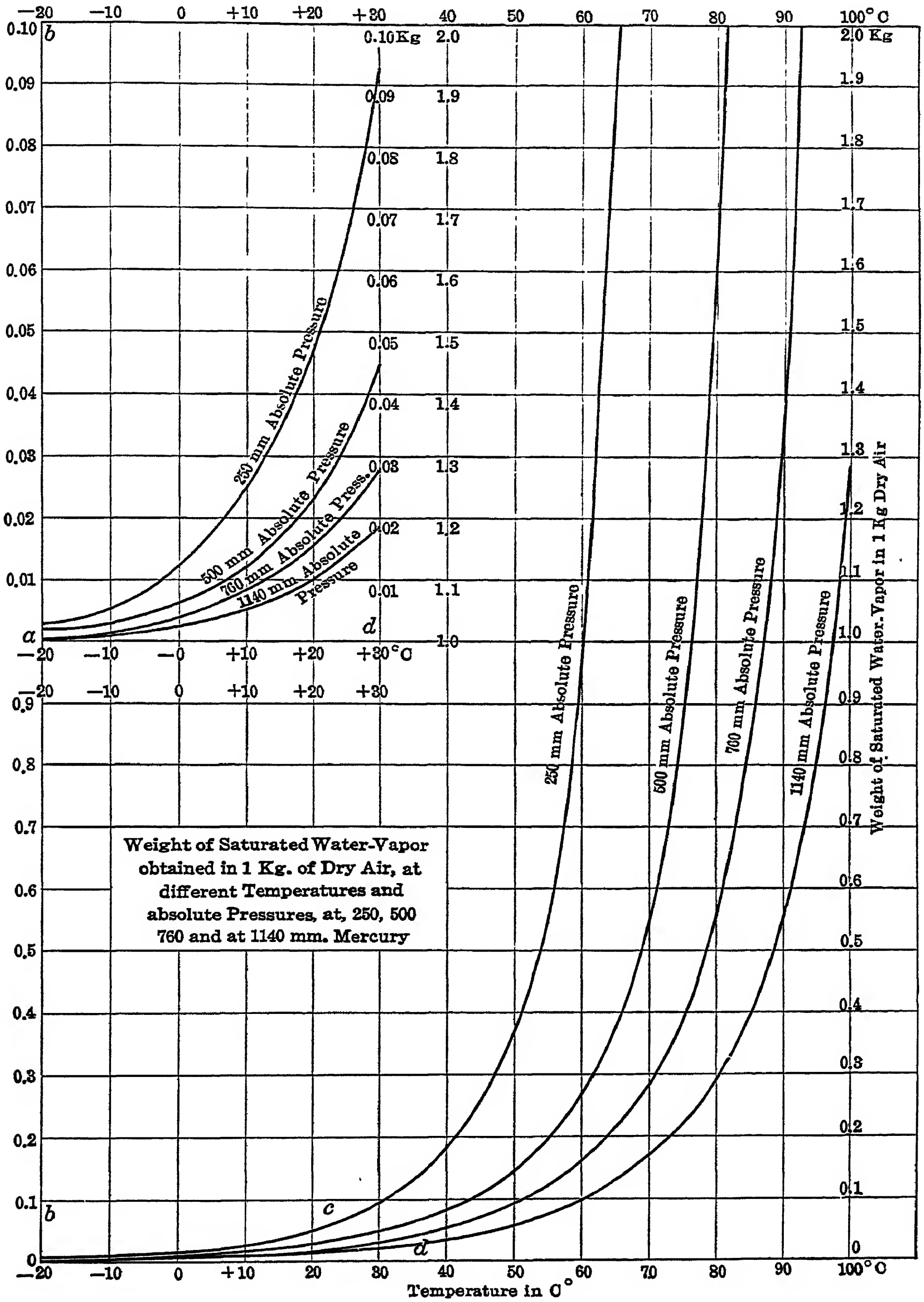


FIG. 262.—Humidity table.

mixed with dried salt to be chloridized; sulphide concentrates with say 10 to 12 per cent. H_2O are often dried before they are roasted; washed coal is usually freed from part of its water by draining before it is burnt under boilers; it has to be dried, however, before it can be briquetted (§111); raw peat (§109) and freshly-cut wood (§100) have to be air-dried before they can serve as metallurgical fuels.

Materials will dry at the ordinary temperature of the air, as this is never completely saturated with water; more often it is half-saturated, and the drier the air, the more moisture it can absorb.[1] The amount of water air or any other gas can take up increases as the temperature rises and the pressure falls. This is clearly shown by the curves in Fig. 262 in which the abscissa represent ° C., the ordinate kg. water in 1 kg. dry air and the curves the absolute pressures in mm. The amounts of water absorbed are seen to rise very quickly with the temperature and to be indirectly proportional to the pressure. In practice it is essential that the drying be carried on at as high a temperature as is practicable, that the air come into intimate and prolonged contact with the ore, and that the air leave the drying apparatus at a temperature sufficiently high to hold as vapor all the water that has been taken up from the ore. Intimate contact between air and ore will be favored by small sizes (not larger than 2 in.), and by stirring the ore.

The common method for removing part of the water is that of air-drying. The raw material is spread on covered ground, bricks and briquettes are distributed on pallets in sheds through which air can circulate freely. With heated low-temperature drying chambers, *e.g.*, for zinc retorts, superheated steam often furnishes the necessary heat; the rooms are sometimes connected with exhaust-fans. Forcing superheated air through drying chambers, common in chemical manufacture, is exceptional in metallurgical work. With high-temperature dryers, the waste heat from roasting and smelting furnaces is often conducted under drying floors or through drying tunnels in which cars charged with material are moved slowly in a direction opposite to that of the gas current. Floors and tunnels often have their separate fireplaces. With plants of large capacity, the ores are dried in special reverberatory and shaft-furnaces similar to those used in calcining and roasting.[2]

171. Calcining (Burning).[3]—Calcining is the process of expelling a volatile constituent of a compound at a temperature below fusion without otherwise affecting chemically the resulting product. Most hydrates give up their water below 300° C.; carbonates (§76) require 600 to 800° C., sulphates (Table 52) and clays (§158) 600 to 1,000° C. to be decomposed; bituminous matter is in part driven off from shales at a low temperature; rocks may have

[1] "Temperature and Relative Humidity Data" for different parts of the U. S. are published by the Weather Bureau.

[2] The Atlas Dryer used by M. A. Hanna Co., Ashtabula, O., for Mesabi iron ore, *Met. Chem. Eng.*, 1912, x, 798.

[3] Schmatolla, E., "Die Brennöfen," Jänecke, Hanover, 1903.
Heinrich, Calcining Kilns, *Tonind. Z.*, 1909, xxxiii, 1388.

to be burnt at high temperatures to loosen their texture. Calcining is confined mainly to burning limestone, dolomite and magnesite (§163); to drive off CO_2 and H_2O from oxide zinc ores and to the decomposition of siderites and clays and clayey ores. In calcining siderite, the iron and, in calcining shales, the metallic sulphides are incidentally oxidized; thus these operations form the transition toward roasting; calcining is sometimes used, though wrongly, as synonymous with roasting. The apparatus and mode of operating are similar to those employed in roasting.

172. Roasting in General.[1]—Roasting may be defined as heating to an elevated temperature, without fusion or with at most incipient fusion, a metal or metallic compound in contact with O, H_2O-vapor, C, S or Cl (HCl) in order to effect a chemical change and to eliminate a component by volatilization. Thus there are distinguished oxidizing, reducing, and chloridizing roasts. Metals and metallic oxides are rarely roasted; but metallic sulphides, arsenides, antimonides, etc. are commonly treated so. The behavior of these compounds, other conditions being the same, varies somewhat with the sizes to which they have been crushed, *i.e.*, whether they are pulverulent, say 8-mesh and finer, or in lump form, say 3 to 4 in. diam.

Friedrich[2] carried on some investigations as to ignition temperature of some of the leading metallic sulphides; these are assembled in Table 153. Of the sulphides, blende decrepitates at 40° C., pyrite at 60°, pyrrhotite at 80°, galena and millerite at 90°, the gas given off reddening litmus paper; the other sulphides show higher decrepitation temperatures.

A more recent investigation[3] correlates the behavior of a matte and speise in roasting with the constitutional diagram.

173. Oxidizing Roast of Metallic Sulphides.—In heating a metallic sulphide (MS) in pulverulent form with access of air to a temperature at which the affinity of the S for O becomes stronger than for the metal, usually a dark red, there is formed metallic oxide and sulphur dioxide ($MS+3O=MO+SO_2+$cal.). The SO_2 will pass off unchanged or in the presence of air it may be converted in part into SO_3 by catalysis, thus:

$2SO_2+3O_2+SiO_2 \rightleftarrows 2SO_3+2O_2+22{,}600$ cal.; in the absence of air the reaction,

$3SO_2 \rightleftarrows 2SO_3+S$ may take place.

($Pd+3SO_2=PdS+2SO_3$; $3Cu+2SO_2=CuSO_4+Cu_2S$).[4]

By coming in contact with a metallic oxide which can form a metallous oxide the SO_2 may be peroxidized $SO_2+2MO \rightleftarrows SO_3+M_2O$; $SO_2+3Fe_2O_3 \rightleftarrows SO_3+2Fe_3O_4$. The conditions necessary for effective catalysis are an excess

[1] Plattner, C. F., "Die Metallurgischen Röstprocesse," Engelhardt, Freiberg, 1856 (tr. by A. Fétis, "Traité théorique des procédés métallurgiques de grillage," E. Noblet, Paris, 1860.

Balling, C. A. M., "Metallurgische Chemie," Strauss, Bonn, 1882, p. 33.

Janda, *Oest. Zt. Berg. Hüttenw.*, 1905, LIII, 223, 234.

[2] *Metallurgie*, 1909, VI, 170.

[3] Friedrich, *Metall-Erz.*, 1912, I, 97.

[4] Uhl, *Berg. Hüttenm. Z.*, 1890, XLIX, 400.

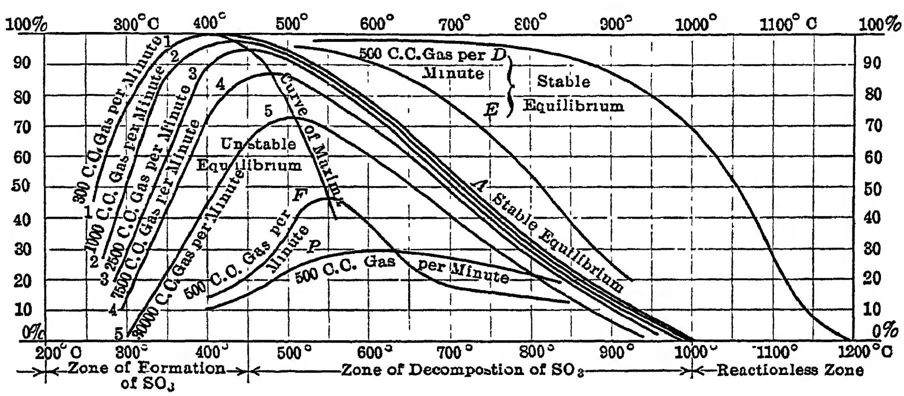
TABLE 153.—IGNITION AND INCANDESCENCE TEMPERATURES, DEG. C., OF SOME METALLIC SULPHIDES, WHEN HEATED IN AIR

Material	Pyrite			Pyrrhotite			FeS	Ni 73.3, S 26.7			Co 66.37, S 33.63			Co 70.20, S 29.80		
Size of grain (a)	I	II	III	I	II	III	II	I	II	III	I	II	III	I	II	III
First notice of SO_2	325	405	472	430	525	590	535	700	802	886	574	684	859	514	751	1,019
Incandescence	.....	533	.....	.....	595	.....	626	...	...	...	...	...	...	...	850	....

Material	Stibnite		Molybdenite		Cinnabar		Chalcocite		Bi 83.3 S 16.7		Mn 61 01 S 33 98 Fe 2.02		Argentite		Blende		Galena (b)		Millerite	
Size of grain (a)	I	III	I	III	I	III	I	III	I	III	I	III	I	III	I	III	I	III	I	III
First notice of SO_2	290	340	240	508	338	420	430	679	500	626	355	700	605	875	647	810	554	847	573	616

(a) I=0.1 mm., II=0.1–0.2 mm., III= over 0.2 mm.
(b) In oxygen.

of air, a suitable finely divided substance and a correct temperature. Fig. 263, representing some of the curves of Knietsch,[1] shows the effects of indifferent catalyzers and temperatures upon a gas of the composition SO_2 7, O 10, N 83 per cent. vol., which corresponds to the equation $2SO_2+3O_2=2SO_3+2O_2$. In curves 1 to 5 platinized asbestos is the catalyzer; from 300 to 30,000 c.c. gas were passed per min. over a given amount of contact mass. Curve 1 shows that with 300 c.c. per min. all the SO_2 is converted into SO_3 at a temperature of 400 to 430° C., and that the amount of SO_3 formed decreases as the temperature rises, the conversion stopping at 1,000° C. Curves 2 to 5 show that with an increased velocity of gas the amount of conversion decreases as is to be expected,



Composition of gas	Curves	Contact
SO_2 7, O 10, N 83%	1–5	Platinum.
" " "	*F*	Roasted Fe_2O_3, CuO, etc.
" " "	*P*	Broken porcelain.
SO_3	*D*	Empty porcelain tube.
SO_3	*E*	Broken porcelain contact.

FIG. 263.—Formation and decomposition of SO_3.

and that the curves converge and unite between 950 and 1,000° C. The contact substance is thus seen to assist conversion up to certain temperatures and to hinder it beyond these. This fact is proved by curve *D* representing the decomposition of SO_3 when conducted through a heated porcelain tube. The SO_3 is stable up to 800° C. and then begins to be decomposed at an accelerated rate with rise of temperature. Curve *E*, representing the decomposition in a tube, filled with fragments of broken porcelain, brings out clearly their effect when compared with curve *D*. That the catalyzing effect of broken porcelain is weak is shown by curve *P* with only 30 per cent. SO_2 converted into SO_3 as a

[1] *Ber. deutsch. chem. Ges.*, 1901, xxxiv, 4069; *Min. Ind.*, 1901, x, 622; *V Internat. Kongress für angewandte Chemie.*, 1903, I, 614.

maximum. Roasted Fe_2O_3, CuO, etc., as contact mass acts more energetically. Its maximum conversion of 45 per cent., as seen in curve *F*, is obtained at 550° C. This then is the amount of conversion that may be obtained in roasting sulphide ores.

The SO_3 formed may pass off as did the SO_2; it may combine with metallic oxide to sulphate, or it may act oxidizingly upon metals and metallic compounds. The formation of metallic sulphate is expressed by $MO+SO_3 \rightleftarrows MSO_4 \pm$ cal. In some instances the whole aim of a roast is to produce sulphate, *e.g.*, in the first stage of chloridizing roasting (§177), in leaching copper ore, zinc ore, nickel ore, copper matte, etc. The operation then goes by name *Sulphatizing Roast.*[1] This is assisted by an admixture of pyrite (FeS_2) or a sulphate ($FeSO_4$) which is decomposed by heat before the sulphate that is to be formed. It is improbable that metallic sulphate is formed by a primary reaction, similarly to the slow oxidation of a wet sulphide, as metallic sulphate upon heating is more or less decomposed into MO and SO_3, or, SO_2+O, but not into MS and O_4; precious metal sulphates form $M+SO_3+O$.

The decomposition temperatures of hydrous and anhydrous sulphates are given in §77.

It is generally held that in decomposing MSO_4 by heat, the SO_3 will pass off as such in the presence of a large excess of air, and that it will be more or less dissociated into SO_2 and O in the absence of it. Laboratory tests[2] made by heating $ZnSO_4$ in air and in a current of CO_2 appear to substantiate this general belief. The occasional opinion[3] that SO_3 passes off as such with sulphates decomposed at low temperatures ($FeSO_4$) and is split into SO_2+O with high temperatures seems to be a half-truth. It appears reasonable from Fig. 263, but improbable, if the sulphatization of Ag by the decomposition of $CuSO_4$ in the Ziervogel process be considered. Some leading facts regarding this subject were given in §77.

The gangue accompanying the sulphide is more or less affected in roasting by heat alone or by the SO_3 set free. This may be the case with some silicates; $CaCO_3$ and $MgCO_3$ may be converted into oxides and sulphates. It has been held, *e.g.*, that in McDougall furnace, treating Butte concentrates, the $CaCO_3$ present was converted into $CaSO_4$; observations made at Anaconda[4] show that the $CaCO_3$ is not decomposed by the sulphurous gases.

The gangue may affect the desulphatization favorably or be indifferent. Taking SiO_2 and Fe_2O_3 as the leading minerals, they will assist the process if the formation of the silicate or ferrite, is lower than the dissociation temperature of the sulphate; they will have no effect if it is higher.[5]

The oxidizing effect of SO_3 may be expressed by the following equations:

[1] Vondracek, *Oest. Zt. Berg. Hüttenw.*, 1906, LIV, 437; 1907, LV, 95.

[2] Hofman, *Tr. A. I. M. E.*, 1905, XXXV, 813.

[3] Janda, *Oest. Zt. Berg. Hüttenw.*, 1905, LIII, 224.

[4] Laist, *Internat. Congr. Appl. Chem.*, New York, 1912, vol. III, p. 97; *Tr. A. I. M. E.*, 1912, XLIV, 806.

[5] Mostowitsch, *Metallurgie*, 1911, VIII, 771.

with a metal, $Ag_2+SO_3=Ag_2O+SO_2$; with a metallic sulphide, $FeS+3SO_3=FeO+4SO_2$, $Cu_2S+3SO_3=Cu_2O+4SO_2$, $Cu_2S+4SO_3=2CuO+5SO_2$; with a metallic oxide, $Cu_2O+SO_3=2CuO+SO_2$.

SO_3 may, however, be deoxidized to SO_2 by heat alone, $SO_3 \rightleftarrows SO_2+O$, or by coming in contact with reducing agents, $2SO_3+S=3SO_2$, $2SO_3+C=2SO_2+CO_2$; $SO_3+CO=SO_2+CO_2$.

The resulting SO_2 may also give up its O to reducing agents, $SO_2+C=S+CO_2$; $SO_2+2C=S+2CO$; $SO_2+2CO=S+2CO_2$; $SO_2+4H=S+2H_2O$; $3SO_2+2CH_4=3S+2CO+4H_2O$; $2SO_2+C_2H_4=2S+2CO+2H_2O$. The occurrence of these reactions will depend largely upon the amount of heat that has to be supplied to bring them about; little, if anything, is known about their reversibility.

Roasting is essentially a slow process carried on at a low temperature. The cold ore will be heated gradually, stirred to bring fresh particles into contact with the air, and the temperature raised to decompose the sulphates that have been formed. Pulverulent roasted ore is porous and is of much greater volume than the raw ore; its color varies according to the percentage of iron present. Sintering during a roast is caused by the fusion of a eutectic mixture which sticks together unfused particles. Sintering or fusion of the charge retards the roast, and may stop it altogether. The result of a roasting operation will depend mainly upon the chemical composition and size of the ore, the admission of air, the thickness of the charge, the amount of stirring of the ore-bed, and the time of exposure and temperature of the furnace. An ore may be purposely given only a partial roast, when some of the S will remain behind as S-ide and S-ate sulphur, or it may be dead-roasted (roasted sweet) when all the S will have been expelled, an operation requiring prolonged heating and stirring. If at the end of the roast the ore has retained its pulverulent form, it is not well suited for smelting in the blast-furnace; the temperature of the hottest part of the furnace is therefore sometimes raised to sinter the particles together (sinter-roasting) or even to fuse them (slag-roasting). The last procedure used to be common with lead ores free from or low in Ag, as undecomposed $PbSO_4$ is thereby converted into silicate: $2PbSO_4+SiO_2=Pb_2SiO_4+2SO_3$, beside furnishing lump ore well suited for blast-furnace work. This process requires a high temperature even with a well-composed charge and is accompanied by considerable losses of Pb by volatilization.

THE INJECTION OF STEAM[1] in an oxidizing roast has been repeatedly suggested, as steam has a decomposing effect upon metallic sulphides at an elevated temperature. It is, however, little used if at all, as it is expensive and as its oxidizing effect is unsatisfactory. The reactions taking place may be expressed by the following examples: $3Ag_2S+2H_2O=6Ag+2H_2S+SO_2$; $Ag_2S+H_2O=Ag_2O+H_2S$; $2FeS+4H_2O=2FeO+SO_2+H_2S+6H$; $3FeS_2+4H_2O=Fe_3O_4+4H_2S+2S$. If H_2S and SO_2 are set free together, they will react upon one another ($2H_2S+SO_2=2H_2O+3S$); if air and steam have access together, any

[1] Plattner, *op. cit.*, p. 238.
Thomas, *Metallurgie*, 1910, VII, 610, 637 (for treating blende).

S or H_2S set free by the steam will burn ($H_2S+3O=H_2O+SO_2$), and SO_2 may form more or less SO_3.

With the exception of some iron ores, ores are roasted in lump form only, if they are rich enough in S to roast of their own accord after the surfaces or the lower parts of the lumps have been brought to the kindling temperature. Oxidation, starting at the surface, generates enough heat to bring the underlying adjacent layer to the ignition point. In this way oxidation penetrates toward the interior; the lump itself swells and becomes porous and fissured, thus furnishing the air new avenues of access. The SO_2 formed is in part converted into SO_3 by the catalyzing effect of the oxidized rind and exerts an oxidizing effect of its own. With lump ore the temperature has to be carefully regulated, as if too high there is danger of sintering and fusing which retards or stops the roast, and if too low the elimination of S is also imperfect. It is evident that the roasting cannot be so complete as when the ore is finely divided. Sulphide copper and copper-nickel ores are frequently roasted in lump form, mattes less often at present than formerly. A special form of this method of roasting goes by the name of *kernel-roasting* practised with copper-bearing pyrite; the formation of enriched kernels with sulphide nickel ores has also been observed.

Losses in metal that are encountered in roasting are due to dusting and volatilization.

174. Oxidizing Roast of Metallic Arsenides, Antimonides, Selenides and Tellurides.—Metallic arsenic heated with exclusion of air is volatilized at 450°. C. without becoming liquefied; brought to an elevated temperature with free access of air it is oxidized to volatile As_2O_3, viz.: $2As+3O=As_2O_3$; at a low temperature with a limited amount of air gaseous As_2O is also formed: $4As+4O=As_2O_3+As_2O$. The As_2O_3 heated in contact with an indifferent substance, or one difficult of reduction, air being excluded, forms non-volatile As_2O_5 and As_2O, viz.: $2As_2O_3+SiO_2=As_2O_5+As_2O+SiO_2$ or $2As_2O_3+3NiO=Ni_3As_2O_8+As_2O$; if air has access it is converted into As_2O_5, viz.: $As_2O_3+SiO_2+O_2=As_2O_5+SiO_2$ or $As_2O_3+3NiO+O_2=Ni_3As_2O_8$.

Heated in contact with readily reducible oxide or sulphate, As_2O_3 is oxidized at the expense of the latter: $As_2O_3+2Fe_2O_3\rightleftarrows 4FeO+As_2O_5$, $As_2O_3+4CuO\rightleftarrows 2Cu_2O+As_2O_5$, $3As_2O_3+6CuSO_4=2(Cu_3As_2O_8)+As_2O_5+6SO_2$.

The process of roasting finely divided metallic arsenide may be summarized by the following example: $FeAs_2+4O=FeO+As_2O_3$; one part As_2O_3 is volatilized, another is peroxidized to As_2O_5 and forms $Fe_3As_2O_8$ (iron arsenate); a third forms $Fe_3As_2O_6$ (iron arsenite) which, however, is readily decomposed into arsenate, magnetic oxide, and arsenic ($2Fe_3As_2O_6=Fe_3As_2O_8+Fe_3O_4+2As$) or into arsenate, magnetic oxide, and arsenious oxide ($Fe_3As_2O_6+3O=Fe_3As_2O_8+Fe_3O_4+As_2O_3$).

Metallic arsenates are not decomposed by heat alone; it requires the presence of fumes of SO_3 or of a powerful reducing agent (C, As) to accomplish it. An arsenical ore can therefore not be dead-roasted by simply heating with access of air. The roasted ore will contain metallic oxide and arsenate. In order to eliminate the last of the As, fritting with alkali and then leaching with water are resorted to to remove the soluble alkali arsenate that has been formed.

Friedrich[1] carried on some experiments upon the ignition and incandescence temperatures of some artificial cobalt and nickel arsenides. He found in the Co series, using material 0.1 to 0.2 mm. in size, that Co arsenides with from 46.5 to 78 per cent. Co begin to roast at temperatures varying between 750 and 822° C., and that the ignition and incandescence temperatures do not rise with the percentage of Co. In the Ni series, the arsenides ranging in composition from 44 to 71 per cent. Ni show, in 0.1 mm. size, incandescence temperatures of from 500 to 1,050° C., which rise with the Ni contents. The same is the case with grains 0.1 to 0.2 mm. diam., excepting that the temperatures are higher, covering a range of from 680 to 1,200° C.

Antimonides show a behavior similar to that of arsenides, excepting that Sb_2O_3 and Sb_2O_5 combine to form the stable SbO_2, and that Sb_2O_3 may be converted into Sb_2O_5 by decomposing sulphates, $Sb_2O_3+2SO_3=Sb_2O_5+2SO_2$. A low temperature and free access of air favor the formation of Sb_2O_5 and SbO_2, a high temperature and a limited air supply that of Sb_2O_3. On the whole little is known in detail about the behavior of antimonides.

The losses in metal by dusting and volatilization are usually higher with As-ides and Sb-ides than with S-ides, as the ores have to be crushed more finely, the temperatures have to be higher,[2] and the vapors of As_2O_3 and Sb_2O_3 are likely to carry along with them finely divided metal, *e.g.*, Ag.

Little can be added to what has already been given in §61, 62, 80 and 81, regarding the behavior of *Selenides* and *Tellurides* in an oxidizing roast, as they have not been much studied.[3] In general the behavior of MSe and MTe in roasting resembles that of MS excepting that both are less readily decomposed.[4] They are converted into MO and volatile SeO_2 (bluish flame and horse-radish odor), or TeO_2 (thick white fumes resembling Sb_2O_3) which have a strong tendency to form $MSeO_3$, or $MTeO_3$ difficult of decomposition by heat alone. The leading ores are those of silver[5] and of gold.[6] The practical points to be considered in roasting are a gradual and even rise of temperature,[7] a prolonged bright-red (900° C.) toward the end, and the danger of loss of precious metal by volatilization. By overheating or by not having sufficient access of air[8] there is danger of fritting fine particles together and imprisoning metal which becomes unamenable to solvents. In roasting, Küstel[9] experienced a loss of 20 per cent. Au, Smith[10] found in 1-hr. tests on Colorado ores losses averaging

[1] *Metallurgie*, 1910, VII, 79.
[2] Howe-Campbell-Knight, *Tr. A. I. M. E.*, 1907, XXXVIII, 162.
[3] Davis, *West. Chem. and Met.*, 1908, IV, 137.
[4] Pearce, *Proc. Col. Sc. Soc.*, 1894–96, V, 147.
[5] Merz, *Oest. Zt. Berg. Hüttenw.*, 1904, LII, 59, 71, 86, 99; *Metallurgie*, 1904, I, 142, 163, 185.
[6] Kemp, *Min. Ind.*, 1897, VI, 295.
[7] Greenawalt, *Eng. Min. J.*, 1905, LXXX, 145, 165; 1906, LXXXII, 193.
[8] Merrill, *Tr. A. I. M. E.*, 1895, XXV, 104.
[9] "Roasting Gold and Silver Ores," Leary, San Francisco, 1880, p. 57.
[10] *Tr. A. I. M. E.*, 1896, XXVI, 494.

11.5 per cent. Au and 8.5 per cent. Ag; Skewes[1] records higher losses. In actual work[2] the losses are very much lower, sometimes practically nil.

Mac and Scibird[3] in roasting telluride gold from Colorado in which calaverite, melting at 365° C., is the leading gold mineral, found that heating gradually to 830° C. with the necessary air was sufficient to eliminate the S, and that the desulphurization was not affected by the size of ore particles as long as they were smaller than 14-mesh. Te began to be driven off at 550 to 575° C.; this continued up to the melting-point of Au (1,064° C.); shotting of Au, *i.e.*, its presence in roasted ore in the form of pellets, was due to fusion of telluride mineral and began at 550° C. The loss in gold was 2.04 per cent.; in practice it would be smaller as part is recovered in the form of flue-dust. Davis[4] has employed the microscope for testing the effect of roasting upon telluride gold ores.

175. Reducing Roast.—This means heating an oxidized metallic compound in contact with C, CO, C_xH_y, (S) in order to effect a deoxidation below the temperature of fusion. Thus hematite ore is heated[5] in a more or less reducing atmosphere in contact with solid fuel to convert Fe_2O_3 wholly or in part into Fe_3O_4 that it may be readily separated from its gangue by magnetic concentration.

Metallic sulphates[6] that are completely decomposed by heat alone are partly reduced below the decomposing temperature when treated in contact with carbonaceous matter, the SO_3 oxidizing the C, viz.: $2MSO_4+C=2MO+2SO_2+CO_2$ or $4MSO_4+3C=2M_2O+4SO_2+3CO_2$. Thus C begins to act upon $ZnSO_4$[7] at 409° C., while the salt begins to be decomposed by heat alone at 702° C. Schorr[8] found that upon addition of charcoal at 560° C. to partly roasted matte the 5 per cent. soluble $CuSO_4$ was quickly reduced, and $CuSO_4$ is decomposed by heat alone at 653° C. The decomposition of a MSO_4 by means of a reducing roast is, however, imperfect, as the reaction $MSO_4+2C=MS+2CO_2$ may take place at the temperature at which the partial reduction becomes decidedly active; thus $Fe_2SO_6+2C=2FeO+2CO+SO_2$ and $Fe_2SO_6+5C=FeS+FeO+5CO$ may take place at the same time, one overlapping the other.

The partial reduction of arsenates (and antimonates) is less complete than that of the corresponding sulphates. Plattner[9] found that ferric arsenate was thus readily decomposed ($Fe_2As_2O_8+2C=Fe_2O_3+As_2O_3+2CO$ or $Fe_2As_2O_8+4C=Fe_2O_3+As_2O+4CO$), $Cu_3As_2O_8$, less readily but still satisfactorily and $Ni_3As_2O_8$ and $Co_3As_2O_8$ very imperfectly. Pearce[10] experimenting with an ore

[1] *Eng. Min. J.*, 1898, LXV, 488.
[2] Argall, *Min. Ind.*, 1897, VI, 372.
Rothwell, *op. cit.*, 1900, IX, 364.
[3] *Min. Sc. Press*, 1907, XCV, 751, 777.
[4] *West. Chem. Met.*, 1908, IV, 137.
[5] Phillips, *Tr. A. I. M. E.*, 1895, XXV, 399.
[6] Gay-Lussac, *Erdmann's J. prakt. Chem.*, 1837, XI, 65.
[7] Hofman, *Tr. A. I. M. E.*, 1905, XXV, 834.
[8] *School Min. Quart.*, 1900, XXI, 66.
[9] *Op. cit.*, p. 252.
[10] *Tr. A. I. M. E.*, 1889-90, XVIII, 62, 457.

consisting of chalcopyrite, bornite and enargite, reduced the original 3.26 per cent. As in a dead-roast to 2.50 per cent. and succeeded in lowering this figure by a reducing roast to only 2.19 per cent. Plattner further found that $Ni(Co)As_2O_8$ could be converted into $Ni(Co)_9As_2O_{14}$ as long as Ni prevailed over Co, and that indefinite basic salts were formed if the reverse was true. Howe-Campbell-Knight[1] found that the addition of charcoal to smaltite ($CoAs_2$) at the beginning or the end of an oxidizing roast failed to increase the expulsion of As. Little is known about the behavior of Sb-, Se- and Te-ides.

A special form of reducing roast of metallic sulphides in which S acting as a deoxidizing agent liberates the metal component in the metallic state, goes by the name of *reaction-roast*. It is practised in some lead ($PbS+PbSO_4=2Pb+2SO_2$ and $PbS+2PbO=3Pb+SO_2$) and copper ($Cu_2S+2Cu_2O=6Cu+SO_2$ and $Cu_2S+2CuO=4Cu+SO_2$) smelting processes.

176. Blast-roast.—Blast-roasting is a generic term given by A. S. Dwight[2] to the process of forcing air through finely divided metallic sulphide with the object of roasting and agglomerating in a single operation. The process which originated with Huntington and Heberlein[3] in 1889 was confined to a galena concentrate, limestone being added to serve both as a diluent to keep separate the particles of galena that they might be thoroughly oxidized, and as a flux that the partly roasted ore might be agglomerated by the formation of a sinter. Later the process was extended to treat other sulphides, as well as arsenides;[4] it has more recently been adopted to the agglomerating of finely divided iron ore by the addition of pulverized coal which furnishes all the heat that is required.[5] As Huntington and Heberlein used limestone with their rough-roasted ore, the term "lime-roasting" was proposed by Ingalls.[6] In the development of the process it was found that lime although desirable with lead ores was not necessary and was even harmful with some other sulphide ores. Austin[7] proposed the term "pot-roasting" as the operations were carried out in spherical kettles, but since the advent of the Dwight-Lloyd roasting machines this second term has lost its original generic meaning.

The last two expressions are still used while "bessemer-roasting" and "blast- and down-draft sintering"[8] and others have remained suggestions. In the original Huntington and Heberlein process the galena concentrate mixed with limestone is given a preliminary rough-roast, in order to oxidize some of the sulphide and thus reduce its calorific power, before it is moistened and charged into the converting pot. In the later Savelsberg process[9] the moistened

[1] *Tr. A. I. M. E.*, 1907, XXXVIII, 162.
[2] *Eng. Min. J.*, 1908, LXXXV, 649.
[3] U. S. Patent No. 600347, March 8, 1898; *Eng. Min. J.*, 1906, LXXXI, 1005.
[4] Guillemain, *Metallurgie*, 1910, VII, 595.
Hahn, *Eng. Min. J.*, 1911, XLI, 858.
[5] *Min. Ind.*, 1911, XX, 481.
[6] *Eng. Min. J.*, 1905, LXXX, 402.
[7] *Min. Sc. Press*, 1906, XCIII, 511.
[8] *Met. Chem. Eng.*, 1910, VIII, 137, 222.
[9] U. S. Patent No. 755598, March 22, 1904; *Eng. Min. J.*, 1905, LXXX, 1067; 1906, LXXXI, 9.

galena-limestone mixture is blown direct without having been subjected to a rough-roast.

In the third modification, the Carmichael-Bradford process,[1] the mode of operating is the same as with the Savelsberg, only limestone is replaced by dehydrated gypsum. These three established processes, as well as some other modifications, are characterized as the *up-draft operations*[2] and are usually intermittent; the *Dwight-Lloyd* process[3] is the leading representative of the down-draft operation which is usually continuous.

Many theories have been proposed regarding the chemical reactions that occur in blast-roasting. The general conclusion[4] is that blast-roasting is nothing more or less than an enforced ordinary roast, in which a large volume of air, forced or drawn into the ore-charge, surrounds the single particles of sulphide, drives off the SO_2 as soon as formed, and thus counteracts to some extent the formation of MSO_4; at the same time it causes the oxidation to proceed at such a speed that the heat generated causes the resulting oxide to form a sintered mass with the accompanying gangue or added flux.

For the success of a blast-roast, it is essential that the sulphurous gases formed be withdrawn as quickly as possible; that the heat furnished by the oxidation be sufficient for agglomeration and not in any great excess over the amount required; that the quality and quantity of diluent flux, or the gangue, be such that it form with the oxide a sintered mass; that the size and form of ore as well as of diluent flux, be correct; and that the operation be carried out in a suitable manner.

The quick withdrawal of SO_2 as soon as formed prevents its being converted into SO_3 by catalysis and acting upon oxide or sulphide. As regards the heat set free in roasting, it is well to recall that the amounts liberated by the oxidation of metals, such as Zn, Fe, Ni, Pb, Cu, per molecule O, are much greater than is that by the burning of S to SO_2, viz., (Zn, O) = 84,800 cal., (Fe, O) = 65,700, (Ni, O) = 61,500, (Pb, O) = 50,800, (Cu_2O) = 40,800 (S, O_2) = 34,630 cal. Thus, a low-grade copper matte, or one that is rich in Fe, will behave differently from one that is high-grade or poor in Fe, and a lead matte differently from a galena concentrate, on account of the high temperatures developed by the first, as compared with the second. On account of the large amount of heat developed by the oxidation of As per molecule oxygen, viz., (As_2,O_3) = 52,100 cal., an arsenide will not be so readily blast-roasted as a corresponding sulphide although, the heat of vaporizaton of arsenious oxide has to be deducted from the above high value.

There are five ways open for correcting the heating effect of an excess of metallic sulphide: One is to add an extra amount of water to the charge, *e.g.*, 15 per cent. as against the usual 5; another to rough-roast the ore in a fine-ore kiln or reverberatory furnace, and thus convert part of the sulphide into sulphate and oxide; the third is to dilute the ore with sufficient flux to reduce the

[1] U. S. Patent No. 705904, July 29, 1902; *Tr. A. I. M. E.*, 1908, xxxix, 629.

[2] Hofman, *Tr. A. I. M. E.*, 1910, xli, 740.

[3] Hofman, *op. cit.*, p. 755.

[4] Hofman, *Eng. Min. J.*, 1911, xci, 48.

calorific power of the ore mixture; the fourth is to choose a diluent of high specific heat; the fifth, a suggestion of Savelsberg, to blow in air containing less than the normal amount of oxygen.

Wetting down the charge is the simplest expedient, but water can be used only in limited amounts. Rough-roasting not only decomposes part of the metallic sulphide, but changing this into oxide contributes thereby a certain amount of diluent and correspondingly diminishes the quantity of extraneous material that would otherwise have to be added. As a rough-roast in a mechanical furnace is a cheap operation, this method offers many advantages over the third, the addition of flux to the raw ores. With this method an economic limit is quickly reached, as it decreases the smelting power of the blast-furnace for ore, the capacity of the shaft being taken up by flux. Dilution of air with fuel gases has not been tried.

The effect of the diluent upon the roast is two-fold. It holds apart the sulphide particles and thereby favors the speed of the roast. As far as this mechanical effect is concerned the character of the diluent has no influence upon the result, but with the thermal and chemical behavior the case is different, as it is essential for a blast-roast that complete oxidation of the sulphide be accompanied or closely followed by sintering, or fusing. The latter takes place at a slightly higher temperature than the former; with silicate mixtures, the two curves usually run parallel. The specific heat of the flux or the gangue must be such as to absorb any excess heat generated by the forced roast, acting as a thermal balance-wheel which equalizes the temperature and holds it at a point at which roasting and sintering can proceed at the desired rate and in the required degree. If the specific heat is too low, the temperature rises too quickly and the charge fuses before it is sufficiently roasted; if it is too high, too much heat is absorbed, the temperature of the sulphide is too low for a perfect roast and especially for the desired sintering, with the result that part of the charge is imperfectly roasted and remains pulverulent. The following substances are arranged approximately in the order of their specific heats, the first having the lowest: Lead oxide, cerussite, anglesite, barite, copper oxide, witherite, manganese oxide, iron oxide, iron silicate, calcium sulphate, basic copper carbonate, alumina, silicate low in iron, hornblende, limestone, sandstone, ground brick, dolomite, clay, copper and lead blast-furnace slag, gypsum.

As regards the chemical effect of the flux, it is essential that the mixtures have a composition which sinters at a low temperature, forming silicate. For example, with lead ores the composition is usually one which will result in a singulo-silicate of low formation temperature formed from the remaining components after the amount of lead and matte expected has been deducted. Of course, if the percentage of metallic sulphide is too low, with galena ore 7 to 10 per cent. sulphur or less, raw sulphide will have to be added to the charge.

The size of grain of the ore will be the same as that which has been found to be best for an ordinary roast, *i.e.*, for galena about 8-mesh; the grain of the flux ought to be coarser with a galena charge, about 4-mesh; further, the grains ought to be angular and not rounded, in order that they may interlock, resist the

pressure of the blast and keep open the spaces necessary to furnish free access to the air. The flux will therefore be crushed fine in rolls and not in ball mills or similar apparatus furnishing rounded grains.

As regards the mode of operating, it is essential that the components of a charge be intimately mixed; further, it has been found necessary to moisten the charge. The water acts in two ways, in that it prevents dusting and keeps down the temperature, excess heat being absorbed by vaporization of the water.

177. **Chloridizing Roast.**—The aim of a chloridizing roast is to convert metals into chlorides and thus render them amenable to solvents; incidentally volatile chlorides are formed which may or may not be recovered. The chloridizing agents are Cl, MCl_x, and HCl-gas. The raw materials for producing them are NaCl ($MgCl_2$), MSO_4, H_2O-vapor and occasionally SiO_2; MS and SiO_2 are either already present in the ore oı, if absent, have to be added.

Cl is produced: (1) By the interaction of NaCl and SO_3 at an elevated temperature: $2NaCl+2SO_3=Na_2SO_4+SO_2+Cl_2$, the SO_3-vapor resulting from the oxidation of the S of the charge; (2) by the combined action of SiO_2 and O upon NaCl at a comparatively high temperature: $2NaCl+SiO_2+O=Na_2SiO_3+Cl_2$; this reaction is insignificant at ordinary roasting temperatures; (3) by the heating of MCl_x without or with access of air: $2CuCl_2 \leftrightarrows Cu_2Cl_2+Cl_2$; $4CuCl_2+O_2=Cu_4O_2Cl_4+2Cl_2$; $3MnCl_2+2O_2=Mn_3O_4+3Cl_2$; $3FeCl_2+2O_2=Fe_3O_4+3Cl_2$. The resulting nascent Cl acts little upon metallic oxides, more so upon metals ($Ag+Cl=AgCl$; $Au+3Cl=AuCl_3$) and their sulphides ($4Cl+Cu_2S+3O=2CuCl_2+SO_3$), arsenides and antimonides.

Metallic chlorides are produced by the action of NaCl or higher MCl_x upon M, MS and MSO_4, viz.: $Fe_2Cl_6+2Ag=2AgCl+2FeCl_2$; $2NaCl+2Ag+CO_2+O=Na_2CO_3+2AgCl$; $Fe_2Cl_6+CuS=CuCl_2+2FeCl_2+S$; $Fe_2Cl_6+Cu_2S=2CuCl+2FeCl_2+S$; $CuCl_2+Ag_2SO_4=2AgCl+CuSO_4$; and $2NaCl+PbSO_4=PbCl_2+Na_2SO_4$. They act in a manner similar to Cl.

Hydrochloric acid is formed: (1) By SO_3, S-ate or SiO_2 acting upon NaCl in the presence of H_2O-vapor, $2NaCl+SO_3+H_2O=Na_2SO_4+2HCl$; $2NaCl+MgSO_4+H_2O=Na_2SO_4+MgO+2HCl$; $2NaCl+SiO_2+H_2O=Na_2SiO_3+2HCl$; (2) by the action of H_2O-vapor upon MCl_x, viz.: $H_2O+MCl_2=MO+2HCl$; (3) by the action of Cl upon C_xH_y, viz.: $C_xH_y+Cl_y=C_x+yHCl$ and $C_x+O_{2x}=xCO_2$. In order to obtain more water-vapor than enters a furnace with the air and the fuel, the ash-pit of the fireplace is sometimes filled with water in order that water-vapor may be generated by the ashes dropping into the water. Nascent HCl acts to some extent upon metallic oxides ($Ag_2O+2HCl=2AgCl+H_2O$) and has a strong decomposing effect upon metallic sulphates ($Ag_2SO_4+8HCl=2AgCl+SCl_2+4H_2O+4Cl$), arsenates ($2Ag_3AsO_4+16HCl=6AgCl+2AsCl_3+8H_2O+4Cl$), and antimonates ($2AgSbO_3+12HCl=2AgCl+2SbCl_3+6H_2O+4Cl$).

Chloridizing roasting finds its main application in the leaching of some copper ores (Longmaid-Henderson process), and nickel ores (Stahl process), and in the leaching and amalgamating of real silver ores. Conducting Cl or HCl-gas through suitable prepared ores has little effect on Cu and Ag, as the gases act mainly in the nascent state. As Cl has hardly any effect upon MO, and HCl

very little, and as both act upon metals, MS, MAs, MSb, and especially upon MSO_4, $MAsO_x$, $MSbO_y$, it is evident that if a metal is to escape chloridation it ought to be present as an oxide. If FeS is to be chloridized, the temperature of the charge will have to be maintained at about 600° C. when the $FeSO_4$, formed during the first stage of the roast, is decomposed. In chloridizing sulphide copper ores, the FeS_x will have been mostly converted into Fe_2O_3 by a preliminary oxidizing roast before any salt is added to the charge.

The temperature for chloridizing will be held at about 670° C. to 750° C. when $CuSO_4$ loses part of its SO_3 and $CuCl_x$ is formed, but only very little $FeCl_x$, the latter being due mainly to the decomposition of $Fe_2SO_6 = Fe_2O_3 + 2SO_3$ at a temperature below that of $CuSO_4$. With silver ores the chloridizing temperature lies above that best suited for copper ores; with Ni ores[1] the original ore is roasted nearly dead and the necessary sulphide added later in the form or FeS_2 with the NaCl in order that Cl, or HCl, may act upon the remaining $NiSO_4$ or $CoSO_4$. These sulphates will have to be more or less decomposed before Mn can be satisfactorily chloridized.

In a chloridizing roast there are generally distinguished two stages, a preparatory roast which usually is more or less sulphatizing, and a finishing roast in which the desired metal is converted into chloride. The gangue of an ore is likely to be affected by the operation. While quartz remains practically unchanged, porphyry and other alumina-bearing rocks may be slightly decomposed; $Ca(Mg)CO_3$ is converted into Ca(Mg)O, $Ca(Mg)SO_4$ and $Ca(Mg)Cl_2$; CaO has a decomposing effect upon base-metal chlorides; $CaSO_4$ is indifferent. All decomposition of country-rock increases, of course, the amount of MS and NaCl necessary for carrying on the process. Losses in metal are likely to behigh. Base-metal chlorides are volatile at low temperatures; AgCl, while non-volatile at roasting temperature, is carried away more or less by the base-metal chlorides and thus volatilized for practical purposes. Losses in Au are likely to be very high.

178. Roasting Apparatus in General.—The apparatus used for roasting ores show a great diversity of form which is caused by the character of the ore, the kind of roast, the tonnage required, and the cost of plant and of operating. The various forms may be classified as follows:

I. Heaps.
II. Stalls.
III. Shaft furnaces.
IV. Reverberatory furnaces.
- 1. Raked by hand:
  - Furnaces with one hearth.
  - Furnaces with two or more superimposed hearths.
- 2. Raked mechanically:
  - Furnaces with stationary hearth.
  - Furnaces with rotating circular hearth.
  - Furnaces with revolving cylindrical hearth.
- 3. Automatic for:
  - Coarse ore.
  - Fine ore.

[1] Stahl, *Berg. u. Huttenm. Z.*, 1894, LIII, 105.

V. Muffle furnaces.
- 1. Raked by hand:
  - Furnaces with one muffle.
  - Furnaces with two or more superimposed muffles.
- 2. Raked mechanically:
  - Furnaces with stationary muffle.
  - Furnaces with rotating circular muffle.
  - Furnaces with revolving cylindrical muffle.
- 3. Automatic.

VI. Kilns.........
- 1. Raked by hand.
- 2. Raked mechanically.
- 3. Automatic for:
  - Coarse ore.
  - Fine ore.

VII. Blast-roasting apparatus. .
- Up-draft. . .
  - Intermittent.
  - Continuous.
- Down-draft
  - Intermittent.
  - Continuous.

The following discussion is general in character, as it is only to bring out the main points of the leading types; the details of the apparatus and their workings belong to the metallurgy of the several metals with which the furnaces find application.

179. **Heaps.**—Roasting in heaps, Figs. 240 and 241, requires that the ore be a metallic sulphide and coarse, *i.e.*, in lump form. Lump ore may be in part replaced by briquettes. The operation consists in piling up the ore to the form of a truncated pyramid on a bed of wood on a suitably prepared ground, and igniting the fuel which heats the superincumbent charge and starts the oxidation. If the charge contains sufficient metallic sulphide to keep up active combustion, the process will proceed of its own accord; if not, the lack has to be made up by mixing in fine fuel. In some instances black-band iron ores or copper-bearing bituminous shales are roasted in heaps. These may not require a fuel bed to start the process, the oxidation of the bitumen furnishing the necessary heat instead of the S.

The most important dimension of a heap is the height; the length and width have little influence upon the result of the roast. The height varies inversely as the percentage of S; an ore with, say, 10 to 12 per cent. S can stand 7 ft. height, one with 35 per cent. S only 5 ft.; the greater the height the stronger the draft and hence the more energetic oxidation; if the roasting proceeds too quickly the temperature rises sufficiently to melt the sulphide ore, and, with fusion, roasting stops. Heaps usually are 6 ft. high. It is more advantageous to have one large heap than several small ones of the same combined capacity, as it is cheaper to build, run and pull down, and as it furnishes less fines to be re-roasted.

The location of the roast yard is usually so chosen that the heap may be protected from strong winds, and so that the prevailing wind may blow the fumes away from the works. The area of the roast yard must be large, as heap-roasting is a very slow process and locks up much ore; the arrangements for handling raw and roasted ore must be efficient, as excepting that it furnishes

lump ore for the blast-furnace, the process has little in its favor if it is not cheap.

Ore heaps are rarely covered with sheds as is sometimes the case with matte heaps. The ground on which the heap is to be built must be dry and solid, similar to a macadamized road. In most instances it is necessary to prepare the ground. A ditch is dug at the upper side to prevent water from running on to the yard which has a gentle slope toward the discharge-track or slopes toward the sides to drain off water. Drainage is often assisted by underground tile drains. Lack of drainage has been the cause of considerable loss of metal in the form of soluble sulphate.[1] In order to make the ground hard, the surface soil is removed and the excavated space filled with rock or coarse slag; the interstices are then filled with coarse to fine gravel (concentrator tailing or granulated slag), and this surface is covered with loam and rolled down. The finished yard should be a little higher than the surrounding ground. The ore for the heap has to be crushed and is usually sized into three grades: Coarse, 1.75 to 3 in.; medium (ragging), 1.75 in. to 3-mesh; and fine, <3-mesh. The coarse forms the body of the heap, the medium the main cover, and the fine a layer varying in thickness to govern the progress of the roast. Heap-roasting is practised mainly in connection with low-grade sulphide copper ores and sulphide copper-nickel ores; formerly matte was treated in this manner, but the process is too slow for rich ore or for an enriched smelter product.

The main advantages of heap-roasting are cheapness of plant and of operation, small consumption of fuel with ores rich in S, and a coarse product for the blast-furnace. The disadvantages are that the ore must be coarse, that the elimination of S is incomplete, that the operation is slow, intermittent and dependent upon the weather, that there is danger of loss by dusting, tramping under foot, and by leaching, and that the vegetation of the surrounding country is killed.

180. **Stalls.**—Roasting in stalls, Figs. 242 and 243, also requires that the ore be in lump form. A stall is an oblong enclosure surrounded on three sides by permanent walls, usually slag brick; the fourth, the front, is wholly or partly closed by a temporary wall of clay brick laid dry or an iron plate which has to be removed every time the stall is to be emptied. A number of stalls are built either in a single row against a main wall, or preferably in a double row as in Figs. 242 and 243, back to back, with a main flue between, as this effects a saving in cost of plant, reduces waste of heat and facilitates the charging of ore and the removal of gases. The top of the stall is usually open; formerly it frequently was covered by a brick arch or an iron plate when it was necessary to carry off all the fumes (Hg) or it was desired to confine the heat in the charge. A good draft and a careful regulation of the admission of air are essential for good work. The height of a stall is governed by the same considerations as that of a heap; as regards the length and width the conditions are just the reverse. Since the larger the stall, the more difficult the regulation of the air current, hence small sizes are the rule, say 8×6 ft. and 6 ft. high. Usually the coarse ore is placed on a

[1] Wendt, *School Min. Quart.*, 1885–86, VII, 173, 180.

bed of wood; it is followed by some medium-size ore and this by a small amount of fines acting as a cover. In some instances a bed of wood for starting the roast has been dispensed with by having several small grates in an end-wall, as in the Wellner stall[1] to kindle the ore, or by having a heavy grate 18 to 24 in. above the floor, as in Vershire,[2] Vermont, or Boston,[3] Massachusetts, to carry a matte charge and fire it from below. These and other modifications are obsolete and deservedly so; anything that complicates the construction and operation of a stall increases the cost and thus makes its existence, already precarious, impossible under present industrial conditions.

Roasting in stalls has cèrtain advantages over roasting in heaps:[4] There is a better distribution and utilization of heat, hence a smaller consumption of fuel; there is less loss of ore by dusting, tramping and leaching; the fumes can be more easily disposed of. The disadvantages are cost of plant and of repairs, and greater expense in charging and discharging. The questions of size of charge for a heap and a stall, and of handling of roasted ore appear to have been the leading factors to put the stall in the back-ground. A heap $20 \times 40 \times 6$ ft. holds 240 tons of pyritic ore, and takes 80 days for building, roasting and tearing down, *i.e.*, it furnishes 3 tons roasted ore per day. A stall $8 \times 6.5 \times 6$ ft. holds 20 tons ore, requires 10 days for filling, roasting and emptying, *i.e.*, it furnishes 2 tons roasted ore per day.

A plant smelting say 500 tons of ore per day cannot undertake the laborious work of charging and emptying the necessary number of small stalls. It has to work with large units, going as far as the use of a steam-shovel to handle roasted ore (Copper Cliff, Ont.). Stalls in this country are, therefore, confined to the roasting of small amounts of coarse ore (East Helena, Mont.; Butte, Mont.) which are to be worked in with other smelting material; they have proved themselves to be unsuited for large-scale work[5] which requires large units.

181. Shaft Furnaces.—In shaft furnaces, Fig. 246, ore and fuel are always charged in horizontal layers; the ore has to be coarse, as the furnaces are worked with natural draft; the furnaces aré automatic, *i.e.*, the passage of the ore is effected by gravity. The charge moves slowly downward through the shaft at the same rate as roasted ore is withdrawn at the base. Here the air enters, takes up the heat of the roasted ore, rises warmed to the roasting zone higher up, and oxidizes the ore. The roast gases ascend through the shaft, gradually give up their heat to the descending charge, and thus prepare it for the oxidizing zone below.

The charge is replenished at intervals at the top, and thus the shaft is kept filled. The fuel commonly used, 5 to 10 per cent. of the weight of the charge, is the fines of the blast-furnace fuel; sometimes special fuel, such as non-coking

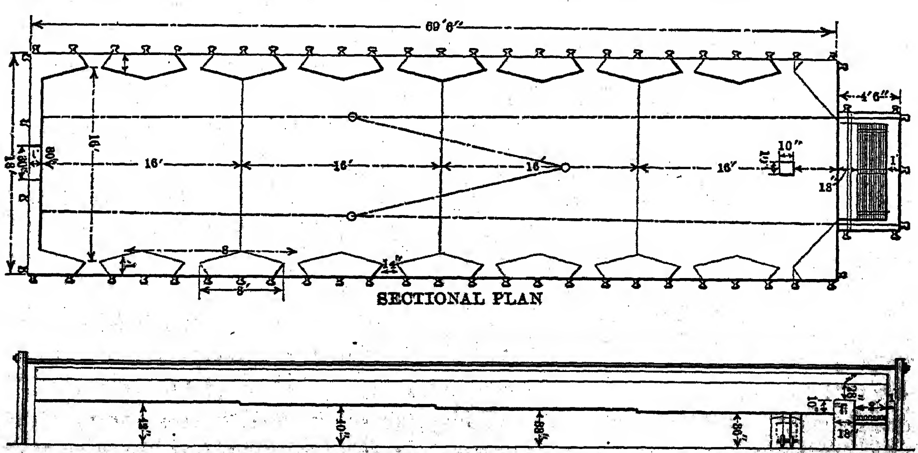
[1] Plattner, "Röstprocesse," pp. 52, 326.
[2] Wendt, *School Min. Quart.*, 1885–86, VII, 306.
[3] Egleston, *op. cit.*, p. 374.
[4] Henrich, *Tr. A. I. M. E.*, 1895, XXV, 224.
[5] Keswick, Cal., *Bull.* 23, State Min. Cal., 1902, pp. 2 and 7.

bituminous coal, has to be provided. The form and height of a shaft vary somewhat with the character of the ore to be treated. Usually a shaft is circular in horizontal section, as this causes an even descent of charge, an even distribution of heat inside and a minimum loss of heat by radiation. The vertical section may show a taper toward the throat, if the ore is to move quickly; one toward the discharge, if slowly; frequently the furnace has parallel or slightly tapering walls which narrow suddenly toward the bottom, *i.e.*, have a bosh. Here the ore will lie compactly and move slowly, thus being exposed for a long time to oxidizing influences.

The ore-column in some furnaces simply rests on the floor, in others on a roof- or cone-shaped center or on a grate to facilitate discharging. The height of the shaft, 8 to 30 ft., varies with the size and fusibility of the ore. Up to a certain limit furnaces of large capacity are cheaper in every respect than small ones; they must, however, not be so large that the outside air cannot penetrate to the center. This may be corrected in part by a central flue, but complications in construction have to be avoided, since the process must be cheap.

Examples are furnished in the calcining of oxide zinc ore, roasting oxide iron ore and sulphide quicksilver ore. From these it is seen that shaft furnaces are used for calcining and for roasting ores that run very low in S; they have, therefore, a field of their own which could hardly be covered by heaps and stalls; they compete, however, with coarse-ore reverberatory furnaces having a vertical hearth (see §182).

182. Reverberatory Furnaces.—Excluding the coarse-ore automatic furnaces which have a shaft-like hearth and are discussed toward the close of this



FIGS. 264 and 265.—One-hearth hand reverberatory roasting furnace.

section, reverberatory furnaces (Figs. 264 and 265) treat finely divided ore, with an upper limit in size of perhaps 4-mesh, whatever may be its mineralogical character; calcination and all kinds of roasts are carried on in it. This type of furnace is in more general use than any other. It shares with heaps,

stalls and shaft furnaces the disadvantage that the gases run very low in S, about 0.5 per cent. SO_2, and are contaminated with the products of combustion of carbonaceous fuel, both of which reasons prevent their being utilized for the manufacture of liquid sulphurous or sulphuric acids, and may necessitate their being rendered harmless (§339). Further, the ore being finely divided, there is always formed more or less flue dust which has to be recovered by some method of condensation. The air necessary for roasting commonly enters the hearth at three places: (1) Over the fire-bridge as excess air in the products of combustion; it is hot, but insufficient in quantity. While superheated air is more effective than cold air, it is not advantageous to admit all the air that is necessary through the grate-bars, as the temperature would be too high for an effective roast and the preheating too costly. (2) Through ports in the fire-bridge, whereby it is slightly warmed, but the volume is obviously too small. (3) Through the working doors;[1] the air is cold, but is supplied to the hearth just where it is wanted, and if preheated would cause a considerable saving in fuel and an accelerated roast.[2] Different attempts have been made to accomplish this, (the latest which promises success is the combined recuperative and regenerative system of Friedrich[3]), but in practically all cases the bulk of the air necessary for oxidation enters the hearth at atmospheric temperature. The ratio of hearth to grate area varies from 15 : 1, in furnaces requiring little extraneous heat, to 10 : 1, in furnaces which require much firing is necessary to expel the last of the S. The dimensions of the hearth vary with the manner of raking the ore. Hand-raked furnaces are always shorter and as a rule wider than those in which the ore is rabbled mechanically. The thickness to which the ore is spread on the hearth is about 4 in.; it may reach 6 and 8 in. with sulphuides that are easily roasted or require only an imperfect roast, or it may be as small as 2 in. with ores that give up their S with difficulty and have to be dead-roasted. The principles of operating are to bring the ore quickly to the ignition-point in intimate contact with air preferably preheated; to keep the temperature approximately constant for a given period of time; to raise it subsequently to the permissible maximum and hold it there for the required time. While heating, the ore is stirred at longer or shorter intervals depending upon the mineralogical character and the kind of roast intended.

The time demanded for roasting varies with the character of the operation; usually the ore remains 24 hr. in the furnace; this time may be reduced to 6 hr. when the ore is to be only rough-roasted, or increased to 48 hr. with a difficult dead-roast. Thus, the roasting capacity per sq. ft. hearth area shows a range of 10 to 70 lb. and usually lies between 15 and 30. The fuel consumption is 200 to 600 lb. coal per ton of ore.

Roasting furnaces have been observed to consume more fuel in summer than in winter, and to eliminate the S less satisfactorily. The reason for this is

[1] In the Greenawalt furnace a large part of the air enters through a porous hearth.

[2] Hommel, *Metallurgie*, 1912, IX, 281.

[3] *Metallurgie*, 1906, III, 749.

to be found, in part at least, in the weaker draft and the smaller efficiency of the hand-labor.

183. Hand-raked Reverberatory Furnaces. THE SINGLE-HEARTH HAND-REVERBERATORY FURNACE.—According to the size of the hearth and the manner of operating, such furnaces may be classed as intermittent and continuous. The intermittent furnace has a small hearth, say 6×8 ft. and not over 10×10 ft., with a working door on one side; it is charged with a given amount of cold ore which is roasted and then withdrawn. For this reason it is very wasteful in heat and labor, and is used only when a small batch of exceptional material, *e.g.*, precipitated Ag_2S, is to be treated or when the quality of work outweighs any other consideration.

The continuous single-hearth furnace (Figs. 264 and 265) has a hearth 40 to 80, usually 60 ft. long. The length is determined by two factors, the capacity of the ore to develop heat while it is being oxidized and the conservation of the heat generated by both the fuel and the ore. With pyritous ore the fire on the grate will be able to maintain the requisite temperature for a longer distance than with ores which run low in S; hence the furnace will have to be longer with ores of the former class than of the latter, if the heat is to be well utilized. The width, governed by the ability of the workman to handle the charge, ranges from 8 ft., with furnaces worked from one side, to 16 or 17 ft., with furnaces worked from both sides (Figs. 264 and 265). The distance between hearth and roof near the fire-bridge is always greater than near the flue-bridge in order to make it possible to carry the heat into the region of the feed opening without unduly overheating the ore near the discharge-end. The change in height is obtained by giving the hearth a slope from flue-bridge to fire-bridge, or preferably, as shown in Figs. 264 and 265, by building the hearth in horizontal planes separated from one another by offsets about 3 in. high. Every hearth, about 16 ft. long, will be served by two working doors on each side with centers 8 ft. apart, or a 64-ft. furnace will have 8 doors on each side. The hearth is built of red brick excepting the parts near the discharge which with the fire-bridge and fire-box is built of fire-brick. The fire-bridge is frequently cooled by a horizontal air canal from which flues may lead on to the hearth and supply it with fresh warmed air. The ore, charged at the flue end, is raked and turned over every 15 to 30 min. and gradually moved toward the bridge end to be discharged as a pulverulent, a sintered or a fused mass. The tools required are rabbles, paddles, and slice-bars to break up accretions.

THE TWO-HEARTH HAND-REVERBERATORY FURNACE.—The object in having superimposed hearths instead of a single hearth is to economize floor-space and heat. The ore fed at one end of the upper hearth travels over it to the other, drops through a slot on to the lower hearth and is then moved in the opposite direction to the discharge near the fire-bridge; the flame issuing from the fire-place at one end of the lower hearth travels in a direction opposite to that of the ore and leaves the furnace near the feed end. Furnaces of this type are permissible only with ores that do not become sticky in roasting or attack the hearth, as repairs on the lower hotter hearth are expensive, because the upper part has

to be removed to get at the lower. Double-hearth furnaces are used in the oxidizing roast of blende and the chloridizing roast of silver and copper ores. Besides economizing floor-space, these furnaces have the advantage that the combined length of the two hearths can often conveniently be made greater than if the total length is confined to a single hearth. It is doubtful whether the better conservation of heat is not more than balanced by the great inconvenience in rabbling and moving the ore. With the lower hearth at the usual height of 30 to 36 in. above the main floor, the workman has to stand on a movable platform to get at the upper hearth; this is conducive to bad work and makes inspection difficult. Nevertheless in Missouri and Kansas four-hearth reverberatory furnaces are in common use for roasting pure blende. Multiple-hearth furnaces make more flue dust than the single-hearth, as the ore dropping from an upper on to a lower hearth makes dust.

184. **Mechanically-raked Reverberatory Furnaces.**[1]—For rough-roasting mechanically-raked reverberatory furnaces have several advantages over long-hearth hand-reverberatory furnaces although the latter give most satisfactory results as to elimination of S, As, etc., and though the consumption of fuel is low, they have these disadvantages that the labor cost is high, that the admission of air is likely to be excessive, and that the result obtained depends largely upon the skill and attention of the operator. Mechanically-raked furnaces do away with these drawbacks. In these the stirring is more frequent and regular, and since the ore can pass through the furnace at a greater speed, the roasting capacity is increased. With this increase there is a corresponding decrease in the amount of fuel burned per ton of ore, as the fuel consumed does not grow proportionally with the amount of ore put through. Further, the possibility of having several fireplaces on the side, Fig. 266, instead of a single one at the end, in as the hand-reverberatory furnace, permits making the hearth as long as is desired and heating as conditions may require. On the other hand mechanically-raked furnaces have certain disadvantages (1) there is considerable cost of installation, of maintenance, of power, of repair of moving parts which are exposed to elevated temperatures and acid fumes, and the necessity of extensive dust chambers; moreover (2) they are unsuited for ores that become sticky, as this by interfering with the mechanism and causing the ore to ball together stops the roast; lastly (3) they are suited only for rough-roasting, being too costly for a finishing-roast. For a dead-roast it is necessary (1) that during the first stage the ore be moved slowly and evenly over the hearth and rabbled at certain intervals of time when the sulphide on the surface has become oxidized; and (2) that during the second stage it remain nearly stationary in the hottest part of the furnace and be rabbled almost continuously to bring sulphate to the surface that it may be decomposed. In a mechanical furnace the moving parts have to travel at a uniform speed; if this is adapted to the first stage of the roast, it is not suited to the second and *vice versa*. Mechanical furnaces will be used mainly with large tonnages of sulphide ore which need not be dead-

[1] Hofman, "Metallurgy of Copper," Detail, p. 116.

roasted, and of siliceous ore containing little S, As, Te, etc., that are to be dead-roasted.

MECHANICALLY-RAKED FURNACES WITH STATIONARY HEARTHS are employed more generally than those in which the hearth is moving. The ore fed at one

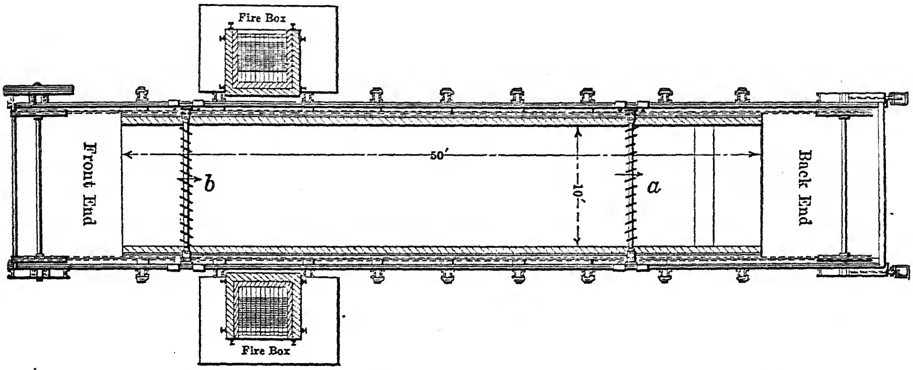


FIG. 266.—Plan Wethey mechanical straight-line reverberatory roasting furnace.

end is moved over the hearth in three ways. The common one is by means of blades, *a b*, Fig. 266, set at an oblique angle to a rabble arm which, in nearly all cases, reaches across the hearth, is supported at the ends or the center, and connected with the motive power. The blades *a*, Fig. 267, of one-half of an arm *A* are set in the opposite direction to that of the other, *b*, in order that the lateral pressure of the ore on the blades of one side may be balanced by that on the other. The rabble-arms are always in pairs, and the blades of one arm are set in a direction opposite to that of the other, *a* and *b* *vs.* *c* and *d*, as when, *e.g.*, the blades on the arm *A*, in making their furrows, push the ore outwardly, those of arm *B* will push it inward, and the combined effect will be to move it onward parallel with the sides of the furnace. The blades may be replaced by plows, the plows of one arm being so set as to cover the furrows made by the other. The second method, used in the Edwards,[1] Fig. 268, and Merton[2] furnaces, is to have radial arms with blades, driven by vertical shafts, which move the ore in zig-zag from feed to discharge. In the third method, represented by the Wedge furnace,[3] the ore fed at the center of a circular

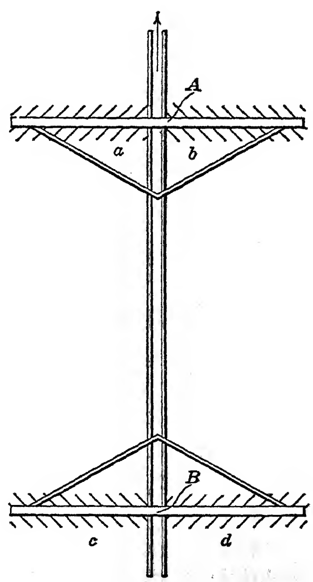


FIG. 267.—Pair of rabble-arms with blades for straight-line mechanical roasting furnace.

[1] Simpson, *Tr. Inst. Min. Met.*, 1903–04, XIII, 27; *Eng. Min. J.*, 1903, LXXVI, 894.
[2] Power, *Eng. Min. J.*, 1903, LXXVI, 775.
Simpson, *Tr. Inst. Min. Met.*, 1903–04, XIII, 30; U. S. Patent No. 1022961, April 9, 1912.
[3] Hofman, "Metallurgy of Copper."

hearth is moved toward the periphery by a pair of radial rabble-arms with blades placed at an oblique angle, but parallel to one another.

In passing over a hearth, the blades or plows may not reach down to the brick bottom, as this is likely to be uneven at the start and, if not, will surely become so after a time. It is customary therefore to have a false bottom, 2 or 3 in. deep, of some loose inert material which does not readily combine with the roasting charge. This also enables the operator to break up with plows or some other means any crust that may form through overheating the ore. In some cases roasted ore forms this false bottom, in others roasted pyrite, crushed limestone, coal ashes, etc.

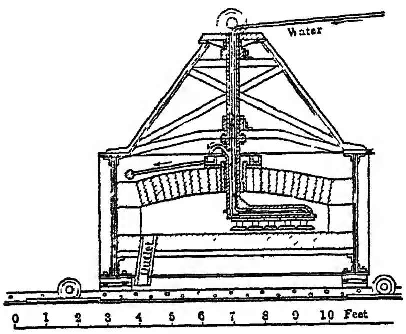


FIG. 268.—Discharge-end of Edwards mechanical roasting furnace.

The hearth of a mechanically-raked furnace may be straight-line, annular or circular. and there may be only a single hearth or two or more superimposed hearths. The straight-line single-hearth furnace, which has the rectangular form of the long-hearth hand-reverberatory furnace, is the one most common; it is represented mainly by the furnaces of Ropp,[1] Cappeau,[2] Brown,[1] Wethey,[1] Edwards. The double-hearth furnaces originated about 1862 with the furnace of O'Hara[3] improved by Brown and by Allen;[3] these were followed by the constructions of Wethey,[3] Keller[3] and Merton. Only one annular furnace (the Brown Elliptical[4]), has come into prominence. Of the circular single-hearth furnaces, the Pearce Turret,[3] and the Brown Horseshoe,[3] are used quite extensively; the Wedge furnace[5] serves a special purpose; of the double-hearth furnaces the Parkes[6] and Ross-Welter[7] are of general interest, while the Pearce is not uncommon. Pearce erected at Butte City[8] a six-hearth furnace.

FURNACES HAVING MOVING HEARTHS.—With these the rakes must be stationary. The hearth is nearly always circular, but two exceptions may be mentioned: the furnace of Argall,[9] which has a rectangular hearth with a reciprocating movement, and the Ems furnace[10] which consists of a train of closely fitting cars moving in a tunnel, the space beneath and above the cars being separated by a curved iron attached to the cars and moving in a trough filled

[1] Hofman, "Lead."
[2] Ingalls, "Zinc."
[3] Hofman, "Lead."
[4] Ingalls, "Zinc."
[5] Hofman, "Copper."
[6] Plattner, "Röstprocesse," p. 22.
[7] *Ann. Min.*, 1885, VII, 512.
[8] *Tr. A. I. M. E.*, 1904, XXIV, 260, 274, 282.
[9] U. S. Patent No. 653202, July 10, 1900.
[10] *Min. Ind.*, 1902, XI, 439.

with sand. The oldest furnace, antedating 1857, is that of Brunton;[1] it was followed about 1870 by that of Gibb and Gelstarp,[2] in 1892 by that of Blake,[3] and about 1899 by that of Godfrey or Heberlein-Hommel.[4] All these furnaces are wasteful of fuel because the flame from the fireplace has only a comparatively short distance to travel before it enters the flue leading to the chimney, and because they draw in a large amount of false air on account of the imperfect joint of the moving hearth with the stationary roof and sides. They, however, accomplish something that other furnaces do not, in that the ore, fed at the center of the hearth, in its spiral passage toward the discharge at the periphery can be heated pretty uniformly. The type is therefore especially adapted to the requirement of heating an ore to a uniform temperature for a considerable period, and of moving and raking it mechanically at the same time.

FURNACES WITH REVOLVING CYLINDERS are iron cylinders, horizontal or inclined, lined wholly or in part with brick. Furnaces with a horizontal axis, of which that of Brückner[5] (1864) is the leading type, work intermittently; furnaces with an inclined axis, such as the White (1870), Howell-White[6] (1872), Hocking-Oxland (1875) and the Argall (1897), have a continuous feed and discharge. The advantages claimed for the type are that the cost of repairs is small and that the ore is stirred continuously. One of the leading disadvantages is that they make a great deal of flue dust, especially when the raising and showering of the ore through the flame is assisted by longitudinal ribs; this dust requires an extended system of dust chambers and a re-treatment of the dust. Another disadvantage is the loss of heat by radiation.

Their use is limited as the layer of ore along the axis of the furnace is rather deep; this is better suited for a chloridizing than an oxidizing roast. However, the Brückner furnace was for a time the leading mechanical furnace in Montana for treating sulphide copper ores, and had replaced most other mechanical roasters for the treatment of mixed lead ores in lead smelteries until it had to give way to blast-roasting. The capacity of the intermittent cylindrical furnace depends mainly upon its length, diameter and speed of rotation; that of the continuous furnace upon the inclination of the axis and speed of rotation. The short intermittent furnace is more wasteful of fuel than the longer continuous form.

185. Automatic Furnaces.—As was stated in §181, a furnace may be called automatic when the passage of the ore through the roasting chamber is effected by gravity. According to the size of the ore that is to be roasted, they are classed as coarse-ore and fine-ore furnaces.

The *coarse-ore furnaces* are similar to the shaft furnaces discussed in §181, excepting that they are heated from external fireplaces at or near the base

[1] *Berg. Hüttenm. Z.*, 1859, XVIII, 355.
[2] *Berg. Hüttenm. Z.*, 1872, XXXI, 309.
[3] *Tr. A. I. M. E.*, 1892, 93, XXI, 943.
[4] *Metallurgie*, 1905, II, 448.
[5] *Tr. A. I. M. E.*, 1873, II, 295.
[6] Ingalls, "Zinc."

where the flame ascends in the furnace filled with ore as in the continuous coarse-ore quicksilver roasting furnace of New Almaden, Cal.[1] The advantage over the shaft furnace, in which ore and fuel are mixed, is that the fuel ashes do not come in contact with the ore, but the operation is expensive as it requires a long-flame fuel, extra labor, and the heat is likely to be excessive. For this reason gaseous fuel (producer gas, tunnel-head gas from the iron blast-furnace) is sometimes used as it is more easily controlled. The gas is introduced at or near the base of the furnace through special openings or nozzles; the air necessary for combustion is admitted either through the ports for the withdrawal of the ore, or higher up through stoking holes or between the two through special air flues. The furnaces are used for the same purposes as the shaft furnaces, *i.e.*, calcining oxide zinc ores, roasting oxide iron ores or roasting sulphide quicksilver ores.

The shaft-like hearth of the *fine-ore furnaces* is either vertical or inclined. In a vertical shaft the ore drops through an empty vertical shaft heated from a fireplace near the top (Whelpely-Storer, 1866, Wilfley, 1908[2]) or near the bottom (Stetefeldt, about 1869); or it glides over shelves placed in the shaft to hold it in its downward passage (Hüttner-Scott, about 1876; Cermak-Spirek, 1891); in an inclined it glides over the floor (Livermore, 1879). The furnaces are used with ores running low in S. Thus, leaving out the Whelpely-Storer furnace which has only an historical interest, the Stetefeldt furnace has become a standard for the chloridizing roasting of silver ores, and the others are characteristic for the oxidizing roasting of quicksilver ores. All these furnaces make a great deal of flue dust, the fine particles of ore being carried off by the large volume of products of combustion traveling in an opposite direction.

186. Muffle Furnaces.—Muffle furnaces are fine-ore furnaces in which only ores that do not become sticky in roasting may be treated. They are suitable for oxidizing roasts of sulphide ore when the S does not generate enough heat to decompose the sulphate formed, and when it is essential that the roast gases be not contaminated with the products of combustion of carbonaceous fuel. The air necessary for the oxidation of the ore being admitted independently from that of the fireplace, the furnace allows a careful regulation of the amount of air admitted and thereby of the volume and concentration of the roast gases. As these are free from harmful carbon compounds, the gases from roasting sulphide ores can be utilized for the manufacture of liquid H_2SO_3 and H_2SO_4 as long as they contain not less than 4 per cent. by vol. SO_2; the gases from arsenical minerals will furnish pure As_2O_3; the gases from chloridizing roasts are easily condensed as their volume is small in comparison with those from reverberatory furnaces. The progress of a roast in a muffle is slow though very uniform, because the heat from the fireplace has to penetrate the brick bottom on which the ore rests. The fire in all cases will have to be hot, as it must balance the cooling effect of the air passing over the ore, unless the air is sufficiently preheated, which is rarely the case.

The general remarks about the different types of reverberatory furnaces

[1] *Tr. A. I. M. E.*, 1884–85, XIII, 561.

[2] McClave, *Eng. Min. J.*, 1908, LXXXV, 453.

(§182) hold good for muffle furnaces, as most reverberatory furnaces can be considered as muffle furnaces if the working hearth of the former be changed into a muffle. The same holds true with the fine-ore kilns discussed in §181.

The hand-raked single-muffle furnace is used mainly in the chloridizing roast of cupriferous burnt pyrite. Superimposed muffles raked by hand are common in Belgium and Germany for the roasting of blende; typical examples are the furnaces of Liebig-Eichhorn, Grillo, Rhenania and Guido Works. While mechanically-raked muffles for roasting blende are little used in Europe, in the United States the tendency is to do away with hand rabbling. The Brown muffle furnace is an early example of a single-muffle furnace rabbled mechanically, and the Pearce of the two-muffle furnaces.

The earliest example of the multiple muffle is probably the rectangular furnace of Mathiesen and Hegeler of 1881; it is closely followed (1884) by the circular furnace of Haas and the more recent modifications of Sjöstedt, Meyer and others. The Falding furnace represents the Edwards and Merton methods of rabbling the ore. Furnaces with a moving muffle have not been constructed, and the only example of a muffle resembling the type of a revolving cylinder is the Douglas furnace, but this has been abandoned.[1] It would appear that a muffle through which the ore passes automatically would be difficult to manage. The Hasenclever furnace of 1872, now abandoned, was for years the leading example; it combined in a unit an inclined automatic muffle ending in a horizontal hand-raked muffle for removing the bulk of the S, with a hand-raked reverberatory furnace to effect a dead-roast.

187. Kilns.[2]—The term kiln is here restricted to the definition given in §167. The heat set free in the roast is sufficient to keep the process going and to decompose the sulphates formed without requiring any extraneous fuel, presupposing that the furnace has been heated at the start to the kindling temperature of the ore. The process, of course, must be continuous, and losses of heat by radiation have to be reduced to a minimum. Kilns are suited only for ores rich in S (28+ per cent.), and pyritic ore forms the usual raw material. It was held that pyrrhotite could not be roasted in a kiln until Sjöstedt[3] proved the contrary. The fact, however, remains that pyritic ore is the most suitable material and that pyrite can carry about 35 per cent. $CuFeS_2$, 18 to 20 per cent. PbS or 1.4 times its weight of ZnS and still work satisfactorily. Besides ore, matte is sometimes roasted in kilns, but imperfectly, giving up about 50 per cent. of its S.

According to the size of the ore that is to be treated, kilns are classed as *coarse-ore* and *fine-ore* furnaces. Coarse-ore furnaces are always automatic; fine-ore furnaces may be automatic, but usually they are rabbled either by hand or mechanically.

The coarse-ore furnaces, Fig. 250, are low rectangular shafts; they are suited

[1] Illustrations of these furnaces are given by W. R. Ingalls in "Metallurgy of Zinc and Cadmium," New York, 1903.

[2] Lunge, G., "Sulphuric Acid and Alkali," Gurney, London, 1913.

[3] *J. Canad. Min. Inst.*, 1904, VII, 489.

to ores that do not carry over 8 to 10 per cent. fines, that do not decrepitate upon heating, and for matte. The walls of the shaft are usually vertical; with very pure pyrite they taper slightly toward the discharge. The cross-sectional area of a furnace is small; the height varies with the percentage of S. Several furnaces are united to a block as are the shelf-burners discussed below. The ore is fed and discharged by hand; it rests on a saddle-shaped bottom or on grate-bars. This type of furnace is used almost exclusively for the production of SO_2 to be converted into H_2SO_4.

The leading fine-ore furnace raked by hand is the "shelf-burner" of Maletra, Fig. 269, built in 1867. A shaft *S* contains a number of staggered shelves *t*; *a* feed-opening, *o* working door, *k* discharge-port, *w* receiving vault and *z* discharge door for roasted ore; *m* gas flue leading into main flue *y*. Several of these furnaces (about 16 in two rows of 8) are united to a block to cheapen the

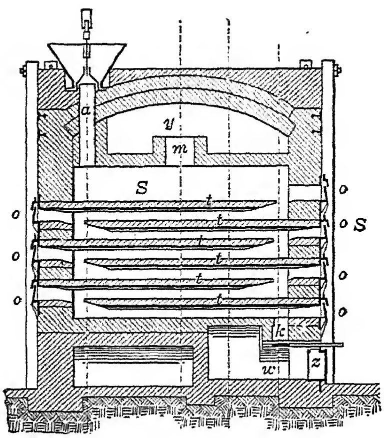


FIG. 269.—Maletra shelf-burner.

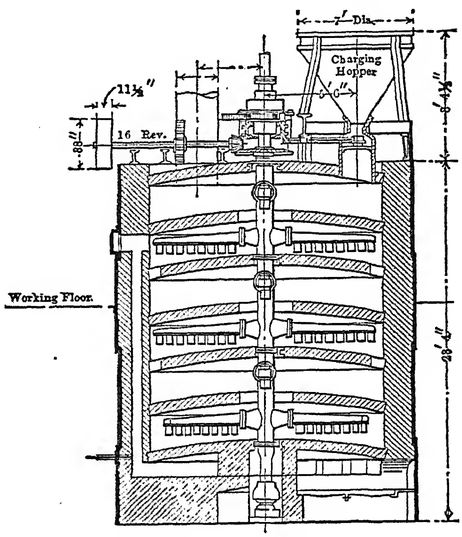


FIG. 270.—Evans-Klepetko-McDougall type of roasting kiln.

construction, to reduce the loss of heat by radiation and to furnish a gas of uniform temperature and composition. While the furnace effects a dead-roast with suitable pyrite, it has some disadvantages; the cost of hand-rabbling is high and air enters the furnace while the ore is being raked from shelf to shelf, and reduces the temperature as well as the S-content of the gas. These drawbacks are removed by mechanical raking first introduced by Spence in 1878; his furnace feeds, moves and discharges the ore by means of machinery. The slow reciprocating movement of the rabbles over the rectangular shelves of the Spence furnace was replaced by McDougall about 1876 by a circular hearth and radial arms attached to a central shaft. While a failure at the start, the furnace has been so much improved that it forms the prototype of the most successful fine-ore burners in this country, those of Herreshoff, Evans-Klepetko (Fig. 270), O'Brien, Wedge and others.

Fig. 270 represents a 6-hearth furnace with fire-box under the lowest to assist in the decomposition of sulphates. The ore, fed mechanically into the furnace, is delivered continuously on the outer edge of the first hearth, where the teeth of the two stirring arms turn it over and work it toward the center; here it drops through an annular opening onto the second; the arms of this turn the ore and convey it slowly to openings near the periphery, through which it drops onto the third hearth and so on until it arrives at the bottom where it falls through a drop-hole into a hopper not shown.

All these furnaces still suffer from the disadvantage of making much flue dust, as the ports through which the ore drops from shelf to shelf form the passages for the ascending gas current. Attempts have been made to separate the two, but not with much success. The furnaces are usually built isolated. Sjöstedt[1] has united four circular furnaces into a rectangular block and has succeeded in this way in making pyrrhotite available as an ore for the manufacture of H_2SO_4.

The fine-ore automatic kilns represented by the constructions of Gerstenhöfer[2] (1863) and Hasenclever-Helbig (1870), have fallen into disuse, the former because the roast is imperfect and the amount of flue dust formed excessive; the latter because it makes the roasting of fine ore dependent upon the working of the coarse-ore division. The furnaces, however, contain constructive features which are of permanent interest and have been adapted to other purposes.

188. Blast-roasting Apparatus.[3]—The apparatus for carrying out blast-roasting operations are quite numerous. They may be divided into two classes: *up-draft* and *down-draft*. In each class some work intermittently, others continuously.

The Huntington-Heberlein blast-roasting pot, Fig. 271, is typical for intermittent up-draft apparatus: *A*, cast-iron kettle; *C*, detached hood; *d*, off-take; *e*, dust flue; *f*, perforated cast-iron plate serving as grate; *g*, baffle-plate for distribution of blast arriving through inlet pipe below. A trough-shaped apparatus has been put into operation by Haas.[4] Continuous up-draft converters have been constructed by Bellinger and Vivian,[5] but details are lacking.

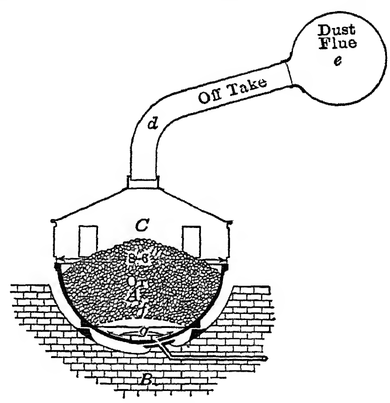


FIG. 271.—Hunting-Heberlein blast-roasting pot.

Intermittent down-draft pots have been constructed by Greenawalt[6] and Dwight-Lloyd;[7] they are, however, of subordinate importance.

[1] *J. Canad. Min. Inst.*, 1904, VII, 489.

[2] Lunge, M., *op. cit.*

[3] Hofman, *Tr. A. I. M. E.*, 1910, XLI, 740.

[4] *Eng. Min J.*, 1910, XC, 814.

[5] *Op. cit.*, 1911, XCI, 49.

[6] *Met. Chem. Eng.* 1912, X, 153, 207.

[7] *Min. Sc. Press*, 1909, LXXXV, 649.

Continuous down-draft apparatus were first brought out by Dwight-Lloyd. There are three types of "sintering machines" the drum-, the horizontal-table, and the straight-line machines; the last is shown in Fig. 272. On account of its importance detailed description is in order. It consists of a frame of structural steel supporting a feeding-hopper, an igniting furnace, a suction-box, and a pair of endless-track circuits to accommodate a train of small truck-like elements called pallets which, in combination, form practically an endless conveyor, with the continuity broken at one place in the circuit. Each pallet is provided with four wheels, which engage with the tracks or guides at all parts of the circuit, except when the pallet is passing over the suction-box, and then the pallet slides on its planed bottom over the planed top of the suction-box, thus making an air-tight joint. A pair of cast-steel sprocket-wheels, turning inside of concentric guide-rails, lift the train of pallets from the lower to the upper track by engaging their teeth with the roller-wheels, and

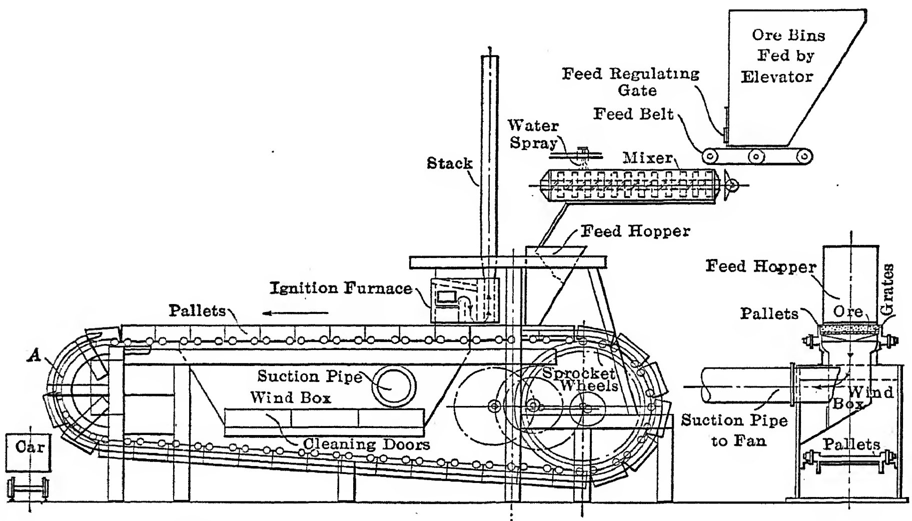


FIG. 272.—Dwight-Lloyd straight-line roasting machine.

launch each pallet in a horizontal path under the feed-hopper and igniting-furnace, and over the suction-box. In a train of pallets in action, all the joints are kept closed, and air-tight, by the pallet being pushed from behind. At the beginning and the end of the track formed by the planed top of the suction-box, there is a planed "dead-plate" over which the pallets must glide; it serves to prevent any leakage of air. After a pallet passes over the suction-box and terminal dead-plate, its wheels engage the ends of the circular discharge-guides. These are adjusted with the view of raising the pallet about 0.5 in. vertically and thus automatically prying up the cake of sinter and freeing it from the grate-slots. A "breaking-roller" prevents the prying action from extending too far back, and tends to form a line of fracture. This roller, however, is not essential in all cases. On reaching the curve of the guides, the pallets one by one drop into

the guides, each strikes the pallet which has preceded it and, at the same time, discharges its load of sinter-cake, and shakes free the slots of the grates. The force of the blow can be regulated by the gap left in the train of pallets at this point. The weight of the train keeps the pallets fed down to the lower teeth of the sprocket-wheels.

The igniter frequently used with this machine is a small coal-burning furnace built of tiles, having a grate-area of 10 by 30 in. and burning 500 lb. of coal in 24 hr. The flame after passing over the fire-bridge is deflected downward upon the ore by a brick curtain that can be raised and lowered, and then is drawn upward by the natural draft of a small stack or bleeder.

The suction-box on top is 12 ft. 6 in. long and 30 in. wide, and gives for the grates an effective hearth-area of 31.25 sq. ft.; this is the true measure of the capacity of the machine. The pallets are each 30 in. wide by 18 in. long and weigh with grates 550 lb.

The power delivered to the machine has its speed-factor reduced by passing through a train of gear-wheels, the last of which engage the internal gear-teeth cast in the large sprocket-wheels, and actuate the train of pallets.

The complete cycle of operations is as follows: a pallet, being pushed onward tangentially from the top of the sprocket-wheels, passes under the feed-hopper, where it takes its load in the form of a continuous even layer of charge, say 4 in. thick, passes next under the ignition-furnace, where the top surface is kindled, and at the same time comes within the influence of the downward-moving currents of air, induced by the suction-draft; these carry the sintering action progressively downward until it reaches the grates. The roast-sintering operation is complete, the cake is discharged by dropping into the discharge-guides, and the pallet crowds its way back to the sprocket-wheels, is slowly raised to the upper tracks, and begins a new cycle.

A straight-line machine of the size described with effective area of 31.25 sq. ft. weighs, without accessories, approximately 16 tons.

A modification of the Dwight-Lloyd Table Machine[1] is found in the von Schippenbach *down-draft* roasting furnace[2] in which the withdrawal of roaster gas is so regulated as to furnish continuously rich gas with from 4 to 5 per cent. vol. SO_2 for the manufacture of H_2SO_4, and poor gas with 0.2 per cent. vol. SO_2 to go to waste.

189. Efficiency of Some Roasting Furnaces.—The efficiency of a roasting furnace can be considered from various points of view. Dwight[3] made the number of pounds of ore treated per day on a square foot of furnace hearth the basis for judging the work of the furnaces given in Table 154. Some of the data cover a considerable range, but this is due to the character of the ore treated and the degree of desulphurization desired. It will be seen that kilns and muffle furnaces are not taken into consideration.

[1] Hofman, *Tr. A. I. M. E.*, 1910, XLI, 761.

[2] Hofman, *Min. Ind.*, 1911, XX, 480.
Kroupa, *Oest. Zt. Berg. Hüttenw.*, 1912, XL, 539.

[3] *School Min. Quart.*, 1911, XXXIII, 1; *Eng Min. J.*, 1911, XCII, 1267.

TABLE 154—EFFICIENCY OF SOME ROASTING APPARATUS

Apparatus	Pounds ore treated in 24 hr. per sq. ft. hearth area	Character of product for blast-furnace smelting
I. Roast heaps and stalls..	5–20	Good.
II. Reverberatory roasting furnaces:		
1. Hand-furnaces....	24–35	Fair.
2. Mechanical furnaces:		
average conditions....	33–75	Too fine.
special conditions....	150	Too fine.
3. Revolving cylinders....	128	Too fine.
III. Blast-roasting pots, range..	500–900	Excellent.
Blast-roasting pots, average..	600	Excellent.
IV. Blast-roasting, thin layers (Dwight-Lloyd system):		
1 Intermittent down-draft pans....	1,000–2,000	Excellent.
2. Continuous sintering-machines......	2,200–3,000	Excellent.

190. Smelting in General.—Smelting is a pyrochemical process for the extraction of a metal or metallic compound from an ore carried on at melting temperature. The difference in specific gravity of the fused metal-bearing part and the fused gangue cause them to separate in layers in the furnace whence they can be withdrawn at different levels. In order to render the gangue fusible at the temperature desired for the separation of the metal-bearing part, it is generally necessary to add certain substances, commonly oxide minerals, called fluxes, with which it forms slags, igneous solutions, the melting-points of which are lower than those of the constituent parts. The leading fluxes are limestone (dolomite), iron ore, manganese ore, silica or aluminous silicates. Their effects are studied in the following sections. In some instances, fluorspar, metallic sulphides, barite, gypsum are added to smelting charges. Fluorspar, which melts at 1,378° C., is very fluid when melted and assists in dissolving refractory slag constituents. With $CaSiO_3$, *e.g.*, melting at 1,501° C., it forms a eutectic with 38.2 per cent. CaF_2 freezing at 1,130° C.[1] The reaction $2CaF_2+2SiO_2=SiF_4+Ca_2SiO_4$ by means of which volatile SiF_4 is to result is of little importance if of any. Metallic sulphides are used for sulphurizing scorified metallic oxides and thus recovering them from the slag. Barite[2] and gypsum[3] furnish both S and bases and act as sulphurizers and fluxes; most of the BaO and CaO will be found in the slag. Slags are formed not only in the treatment of ore, but also in processes carried on to purify, *i.e.*, refine metals, and to work up metallic

[1] Karandief, *Zt. anorg. Chem.*, 1910, LXVIII, 188.
[2] Mostowitsch, *Metallurgie*, 1909, VI, 450.
[3] Hofman-Mostowitsch, *Tr. A. I. M. E*, 1908, XXXIX, 628, 1909 XL, 807; 1910, XLI, 763. Glasenapp, *Tonind. Z.*, 1908, XXXII, 1148, 1197, 1230.

compounds. They are thus essential accompaniments of every smelting operation.

Smelting processes in general are either mainly reducing or mainly oxidizing. The metal-bearing product of a reducing fusion is usually an impure metal, such as pig iron, pig tin; it may be a metallic sulphide called matte, as with sulphide ores; or an arsenide called speise, as with arsenical nickel ores; sometimes all three classes of products are formed in a single fusion, as with mixed Pb, Cu, Ni ores containing S and As. The aim of an oxidizing fusion is (1) to remove impurities contained in metal, as in the conversion of pig iron into steel or wrought iron; the refining of copper, the separation of lead and silver; or (2) to decompose a matte or a speise and thus recover the metals with which the S, or the As, was combined; or (3) to smelt a sulphide ore for matte (pyritic smelting).

191. Slags in General.[1]—Slags were formerly believed to be mixtures of chemical compounds formed in smelting which retained the same chemical composition in the solid state that they had in the liquid. At present slags are held to be igneous solutions of the constituent oxides and of the chemical compounds that may have been formed, the compositions of the latter depending upon the thermal and chemical equilibria of the metallurgical processes in which the slags are produced. Slags, therefore, ought to follow the laws of solutions in a manner similar to that of alloys; there is sufficient experimental evidence to show that in a general way this is the case.

A study of slags covers their constitution, fusibility, fluidity, texture, specific gravity, color, hardness, specific heat, latent heat of fusion, heat of formation, and economic use.

Slags may be classed as *silicate slags* consisting of metallic oxide and SiO_2, or *non-silicate slags*, consisting either of metallic oxide and acid other than SiO_2, *e.g.*, P_2O_5, SO_3, B_2O_3, Sb_2O_5, As_2O_5, TiO_2, etc., or one or more basic oxides, *e.g.*, PbO, CaO, Fe_2O_3, Mn_2O_3, Al_2O_3, etc. Silicate slags are the most important and have received considerable study; comparatively little is known about the non-silicate slags but the principles governing the silicate slags may be applied with suitable modifications to the non-silicate slags.

192. Silicate Slags.—In conformity with the classification of the anhydrous silicate minerals, silicate slags have been classed as sub-, singulo-, bi- and tri-silicates according to the oxygen ratio of bases RO to SiO_2 or the valences of the metal bases to that of Si. This classification, common to-day, is given in Table

[1] Percy, J., "Metallurgy of Fuel," Murray, London, 1875, pp. 46–86; Jüptner, H. v., "Grundzüge der Siderologie," Felix, Leipsic, 1900, Vol. I, p. 304, partial bibliography; Vogt, J. H. L., "Die Silikatschmelzlösungen," Dybwad, Christiania, 1903–04; Babu, L., "Métallurgie Générale," Béranger, Paris, 1904, Vol I, pp. 470–576; Jüptner, "Constitution of Slags," *J. I. and St. I.*, 1900, II, 276; "Kenntniss der Schlacken," *Oest. Zt. Berg. Hüttenw.*, 1902, L, 165, 182; Mathesius, "Die Entstehung der Schlacken," etc., *Berg. Hüttenm. Z.*, 1904, LXIII, 381; *Stahl u. Eisen*, 1904, XXIV, 1000; Vogt, J. H. L., "Die Theorie der Silikatschmelzlösungen," Bericht V. Internat. Kongress Angew. Chemie, Berlin, 1904, Vol. 2, pp. 70–90; Turner, T., Physical and Chemical Properties of Slags, *J. Soc. Chem. Ind.*, 1905, XXIV, 1142; Fulton, C. H., "Principles of Metallurgy," McGraw-Hill Book Co., New York, 1910, pp. 245–288; Doelter, C., "Handbuch der Mineralchemie," Steinkopf, Dresden, 1911, Vol. I, 557.

155. Åkerman and Vogt conforming to modern chemical nomenclature have substituted the name ortho-silicate for singulo-silicate and meta-silicate for bi-silicate, and confined themselves to these two terms, as in their synthetical fusion experiments followed by microscopical examinations, they were enabled (with one exception) to detect only singulo- and bi-silicates; they concluded that the other so-called silicates were simply mixtures of oxides, singulo-silicates, bi-silicates and SiO_2.

Table 155.—Metallurgical Classification of Silicates

Name		Silicate-Degree; O base: O acid; Valency Metal : Valency Si	Formulæ	
Common	Åkerman-Vogt		Bi-valent or monoxide bases	Tri-valent or sesquioxide bases
Sub-silicate....	....	2:1	$4RO.\ SiO_2=R_4SiO_6$	$4R_2O_3.3SiO_2=R_8Si_3O_{18}$
Singulo-silicate	Ortho-silicate	1:1	$2RO.\ SiO_2=R_2SiO_4$	$2R_2O_3.3SiO_2=R_4Si_3O_{12}$
Sesqui-silicate.	....	2:3	$4RO.3SiO_2=R_4Si_3O_{10}$	$4R_2O_3.9SiO_2=R_8Si_9O_{30}$
Bi-silicate......	Meta-silicate	1:2	$RO.\ SiO_2=RSiO_3$	$R_2O_3.3SiO_2=R_2Si_3O_9$
Tri-silicate..	....	1:3	$2RO\ 3SiO_2=R_2Si_3O_8$	$2R_2O_3.9SiO_2=R_4Si_9O_{24}$

Tri-silicates are not produced in metallurgical operations on account of their high degree of viscosity; bi-silicates even are uncommon.

The usual monoxide bases are CaO, MgO, BaO, MnO, FeO ($2K_2O$, $2Na_2O$, Cu_2O, PbO, NiO, CoO, ZnO, and SnO); the only sesquioxide of importance is Al_2O_3, and this under suitable conditions may behave as an acid, forming aluminates (§206). Other sesquioxides are Fe_2O_3, Mn_2O_3.

According to the character of the bases, silicate slags are sometimes classed as non-ferrous and ferrous. The non-ferrous or iron blast-furnace slags (French, laitier) are practically free from FeO and consist mainly of SiO_2, Al_2O_3 and CaO (MgO.BaO). The ferrous slags (French, scorie), rich in FeO, are produced, either in smelting Pb, Cu and similar ores, when they are composed of SiO_2, FeO (MnO), CaO (MgO, BaO), some Al_2O_3 and occasionally ZnO, or in puddling pig iron, when they are made up of SiO_2 and Fe_xO_y.

Table 156 of Balling,[1] to which the data for BaO have been added, simplifies finding the amounts of SiO_2 and base required to form singulo-, sesqui- and bi-silicate slags in calculating furnace charges.

For the computation of the silicate degree of a slag from the chemical analysis, it is necessary to determine the ratio

$$\frac{\text{O in acid}(SiO_2)}{\text{O in bases}(FeO\,.\,MnO.\ CaO\,.\,MgO\,.\,Al_2O_3\,.\,BaO)}$$

[1] "Compendium der Metallurgischen Chemie," Bonn, 1882, p. 98.

TABLE 156.—COMPUTATION OF BASES AND SILICA REQUIRED TO FORM SILICATES

1 lb base requires lb. SiO_2 to form a			Name of base	1 lb. SiO_2 requires lb. base to form a		
Singulo-silicate	Sesqui-silicate	Bi-silicate		Singulo-silicate	Sesqui-silicate	Bi-silicate
0.535	0.803	1.070	CaO	1.86	1.24	0.93
0.196	0.294	0.392	BaO	5.10	3.40	2.55
0.750	1.125	1.500	MgO	1.33	0.88	0.66
0.873	1.310	1.747	Al_2O_3	1.14	0.76	0.57
0.416	0.625	0.883	FeO	2.40	1.60	1.20
0.422	0.633	0.845	MnO	2.36	1.57	1.18

The calculation is much simplified by Table 157 of Mostowitsch[1], slightly changed. Take a slag of the composition SiO_2 40, FeO 36, CaO 20 per cent. The O in SiO_2 is $40 : 30 = 1.33$, the O in bases is (for $FeO = 36 : 72 = 0.50$, and for $CaO = 20 : 56 = 0.35$) $0.50 + 0.35 = 0.85$; hence the silicate degree $1.33 : 0.85 = 1.565$, a sesqui-silicate.

Conversely, to form a singulo-silicate of MgO, it requires 30 lb. SiO_2 and 40 lb. MgO; for a singulo-silicate of CaO and MgO, it requires 30 lb. SiO_2, $\frac{CaO}{2} = 28$ lb. CaO, $\frac{MgO}{2} = 20$ lb. MgO; for a singulo-silicate of FeO, CaO and MgO it requires $\frac{FeO}{3} = 24$ lb. FeO, $\frac{CaO}{3} = 18.6$ lb. CaO, $\frac{MgO}{3} = 13.3$ lb. MgO. Calculations for the chemical formulæ are given by Balling[2] and von Jüptner.[3]

TABLE 157.—EQUIVALENT VALUES OF SLAG-FORMING CONSTITUENTS BASED ON 1 MOL. O

Slag-forming constituent	Mol. weight	Wt. O in 1 mol.	O in 1 mol. / at wt. O (=16)	Equivalent values based on 1 mol. O
SiO_2	60	32	2	$60 : 2 = 30$
FeO	72	16	1	$72 : 1 = 72$
MnO	71	16	1	$71 : 1 = 71$
CaO	56	16	1	$56 : 1 = 56$
MgO	40	16	1	$40 : 1 = 40$
Al_2O_3	160	48	3	$160 : 3 = 53.3$
BaO	153.4	16	1	$153.4 : 1 = 153.4$

193. Constitution.—The constitution of silicate slags is ascertained by the microscopical examination of specimens and by tracing freezing-point curves.

[1] *Metallurgie*, 1912, IX, 566.
[2] Compendium der Metallurgischen Chemie," Strauss, Bonn, 1882, p. 122.
[3] "Grundzüge der Siderologie," Felix, Leipsic, 1900, I, 260.

Slags that have been chilled are likely to be glassy and represent solid solutions; when cooled slowly, they show crystalline aggregates which reveal the mineral composition, unless they are too silicious. Vogt (*op. cit.*) examined slags having mainly CaO, MgO, MnO, FeO and small amounts of Fe_2O_3 and Al_2O_3 as bases. He distinguishes the minerals shown in Table 158.

TABLE 158.—MINERAL CONSTITUENTS OF SILICATE SLAGS

Silicate degree	Formula	Mineral	Crystal system
6.00	$\overset{II}{R}Si_2O_7$	?	?
4.00	$\overset{II}{R}Si_2O_5$	?	?
3.00(?)	$\overset{II}{R}_2Si_3O_8$	Globulites in acid slags	?
2.00	$\overset{II}{R}SiO_3$	Babingtonite $(MnO.RO)SiO_2+Fe_2O_3\cdot3SiO_2$	Triclinic.
		Rhodonite, $MnO.SiO_2$	Triclinic.
		Augite varieties, $CaO(MgO)SiO_2$	Monoclinic.
		Wollastonite, $CaO.SiO_2$	Monoclinic.
		Bronzite, Hypersthene, $FeO.MgO.SiO_2$	Orthorhombic.
		Enstatite, $MgO.SiO_2$	Orthorhombic
		Hexagonal lime silicate, $CaO.SiO_2$	Hexagonal.
1.50	$\overset{II}{R}_4Si_3O_{10}$	Åkermanite, $R_4Si_3O_{10}$, new tetragonal mineral free from Al_2O_3, rich in CaO	Tetragonal.
1.30	$\overset{II}{R}_2SiO_4$	Willemite, $2ZnO.SiO_2$	Orthorhombic.
		Monticellite, $CaO.MgO.SiO_2$	Orthorhombic.
		Tephroite, $2MnO.SiO_2$	Orthorhombic.
		Fayalite, $2FeO.SiO_2$	Orthorhombic.
		Olivine, $2MgO.SiO_2$	Orthorhombic.
		Melilite, $12RO.2Al_2O_3.SiO_2$	Tetragonal.
0.67	$\overset{II}{R}_3SiO_5$	Gehlenite, $3RO.Al_2O_3.2SiO_2$	Tetragonal.

Their manner of formation is shown in Figs. 273 and 274. In slags of a silicate degree of 2.5 and higher there is formed, upon cooling, an enamel or glass, *i.e.*, solid solution, which contains globulites and crystallites of bi-silicate. From 2-silicate slags enstatite crystallizes out, if the ratio MgO (+FeO) : CaO (+MnO) is greater than 2.44 : 1; augite will form, if more CaO is present than corresponds to the ratio 1.40 MgO : 1 CaO; if the CaO-content exceed the ratio 0.3 MgO : 1 CaO, monoclinic wollastonite or hexagonal CaO metasilicate will form. Rhodonite crystallizes from a 2-silicate slag, if more MnO is present than in the ratio 1 MnO : 0.9 (Ca.Mg.Fe)O; and augite, if less than 1 MnO : 7 (Ca.Mg.Fe)O; with an average content of MnO, *e.g.*, 1 MnO : 1.82 (Ca.Mg.Fe)O, rhodonite and augite may form simultaneously. From 1-silicate slags olivine crystallizes out, if the ratio (Mg.Mn.Fe)O : CaO (in addition to Na_2O and K_2O) exceeds 1 : 1.10–1.20 (the first limit corresponds to a high percentage of Al_2O_3 (say 20), the second to a low one (say 3). Melilite will form if there

is a larger percentage of CaO than 1 (Mg.Mn.Fe)O : 1.25–1.30 CaO (the limits varying again with the percentage of Al_2O_3).

The pyroxene group of minerals and the hexagonal lime silicate crystallize out only if the silicate degree exceeds 1.50 to 1.60, Al_2O_3 being reckoned as base; the boundary between the hexagonal lime silicate and one of the tetragonal minerals lies at a silicate degree of 1.42 to 1.45; the limit between augite and tetragonal mineral lies at 1.50 to 1.53; and between augite and enstatite and olivine slags at 1.5; Al_2O_3 only slightly affects these boundaries.

Åkermanite forms only between the O-ratios 1.35 and 1.50; gehlenite separates from a fused mass rich in Al_2O_3 which is of a silicate degree below 0.8.

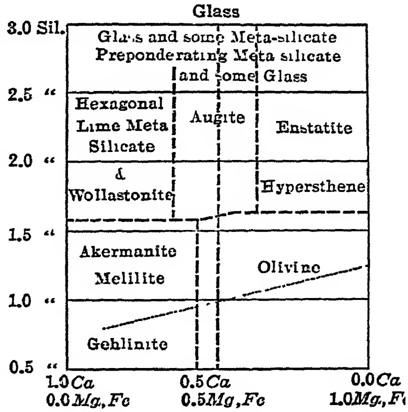
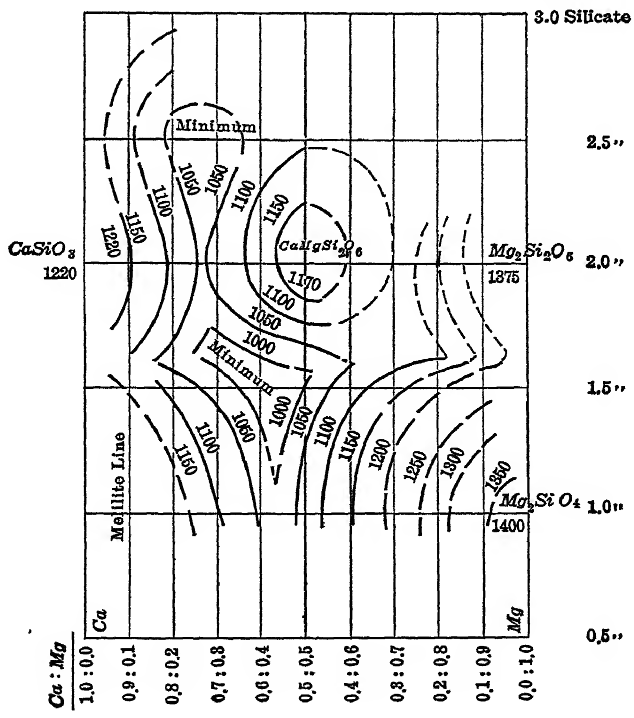


FIG. 273.—Constitution of silicate-slag minerals.

In Fig. 273 is drawn an inclined line showing the limits of the formation of spinell, $RO.R_2O_3$, which belongs to the non-silicate slags. Spinell is found only in basic slags rich in Al_2O_3 and separates before the silicate minerals; $MgO.Al_2O_3$, *e.g.*, is found in iron blast-furnace slags; $ZnO.Al_2O_3$ forms more readily than $MgO.Al_2O_3$; slags containing both



FeO and Fe_2O_3 are liable to show some magnetite, Fe_3O_4, as long as the silicate degree does not exceed 2; the magnetite separates after spinell, but before or simultaneously with the silicate minerals; Fe_2O_3 forms compounds with other monoxide bases; Mn_2O_3 shows a similar behavior.

The solid-line isotherms in Fig. 274 are believed to be reliable; those represented by broken lines are approximately correct, those in dotted lines represent calculated values. Eutectic mixtures are seen to exist between augite and olivine (ratio 70 : 30); melilite and olivine (ratio 74 : 26), augite and hexagonal CaO-silicate (ratio 55 : 44); probably also between augite and enstatite.

In the determination of the cooling curves of silicate slags many difficulties are encountered, as they usually do not show decided retardations upon solidification, but cover a wide melting range because they pass gradually from the liquid to the solid state and *vice versa.* The only freezing-point curves known are those of the CaO-, MgO- and Al_2O_3-SiO_2 series and even here there are gaps that cannot be filled out at present.

194. Fusibility.—The fusibility of a silicate slag depends upon the silicate degree and the character of the bases. The following two methods have been used in recent years for approximating the effects the fluxes have upon SiO_2; they are the determination of the total heat of solidification of fused silicates used by Åkerman,[1] and the determination of the formation temperatures used by Grenet,[2] Gredt,[3] Hofman,[4] Boudouard,[5] Day-Shepherd-Rankin[6] and others.

In the determination of the total heat of solidification, a slag mixture is melted in a crucible, cooled and partially solidified, and the remaining molten part poured into a calorimeter. The latent heat of solidification *plus* the heat evolved in cooling from the melting-point down to ordinary temperature gives the total heat of solidification. From this can be calculated the melting temperature according to the formula $Q=C_0^T T+L$, in which Q is the total heat of solidification in g. cal. per g., C_0^T the average specific heat between o and $T°$ C., T the fusion temperature, and L the latent heat of fusion. $C_0^T T=0.29$–0.33 with SiO_2-CaO-Al_2O_3 slags, the lower figure to be used for the more readily and the higher for the less readily fusible slags; L ranges between 73 and 120, average 95 g. cal. per g.

In determining the formation temperature two methods may be used. In one the components of the slag mixture, ground finely to a uniform limiting size, are mixed dry, molded with a dextrine solution to the form of a Seger cone (§97), dried, and then heated on a platinum foil support in a crucible containing Seger cones to serve for control of temperature. As Seger cones are not pyrometers, but only pyroscopes, accurate temperature determinations cannot be made with them; further disadvantages in their use are that the time factor and the

[1] *Stahl u. Eisen,* 1886, VI, 281; *J. I. and St. I.*, 1886, I, 310.

[2] Babu, *op. cit.*, Vol. I, p. 500.

[3] *Stahl u. Eisen,* 1889, IX, 756.

[4] *Tr. A. I. M. E.*, 1889, XXIX, 682.

[5] *J. I. and St. I.*, 1905, I, 339; critique by Richardson, *I. and St. Mag.*, 1905, X, 295; reply *Rev. Mét.*, 1906, III, 217.

[6] *Am. J. Sc.*, 1906, XXII, 265; 1909, XXVIII, 293; 1911, III, 211.

personal equation of the observer play an important rôle in judging of the result. Nevertheless, in making series of tests data of metallurgical value are obtained, even if they are not scientifically correct.

In the other method followed by Shepherd, Day, Rankin and others, the slag mixtures are heated in an electric resistance furnace, the temperatures are accurately measured by means of a thermo-electric or an optical pyrometer, and the physical changes in the mixtures noted. This method gives accurate data, but usually is out of reach of the average worker when dealing with temperatures exceeding say 1,200° C.

The formation temperatures of silicate slags are lower than the temperatures at which the formed silicates melt, provided the components are ground finely and mixed intimately. This has been shown by Day-Shepherd,[1] Hofman-Mostowitsch[2] and others; it upsets the older theory of Plattner which held the reverse.

In practical smelting where the different parts of a charge are neither finely divided nor intimately mixed, the Plattner theory holds good. The components of a slag will always have to be brought to a higher temperature than the melting-point of a formed slag in order that the slag may form at all. It is the common practice in most smelting operations to add formed slag to the charge, as in melting readily it induces the charge to melt.

The curves so far obtained by the three methods outlined give a general idea of the constitution of the slags. Some of the leading results of Åkerman, Boudouard, Gredt, Grenet, Hofman, Day-Shepherd-Rankin and others are given below. References to the older researches by Lampadius,[3] Sefström,[4] Berthier,[5] Percy-Smith,[6] Bischof[7] and Plattner[8] are given in the foot-notes.

195. Sodium and Potassium Silicates.—Alkali silicates form at low temperatures. Leaving assaying out of consideration, they are of minor importance in smelting work, but are formed in the refining of precious metals. Na_2SiO_3 melts at 1,007° C.,[9] K_2SiO_3 at 890° C.[10] Figs. 275–276[11] give the data of Grenet, which are lower than the later investigations just quoted, the sodium bi-silicate, with 49 per cent. SiO_2, being shown to have a formation temperature of only 735° C. The curves, however, show that temperatures rise gradually with an increase of SiO_2 until the mixture contains about 86 per cent. SiO_2 and that the further rise is sudden. The rise of formation temperature of potassium silicates with

[1] *J. Am. Chem. Soc.*, 1906, xxviii, 1092.

[2] *Tr. A. I. M. E.*, 1908, xxxix, 628.

[3] *Journal des Mines*, 1805, xviii, 171; also, "Handbuch der Allgemeinen Hüttenkunde," Dietrich, Göttingen, 1801–1810, I, 127.

[4] *Jernkont. Ann.*, 1828, I, 155; also Erdmann's *Tech. Oek. Chem.*, 1831, x, 145.

[5] "Traité des Essais," Thomine, Paris, 1834, I, 430.

[6] Percy, J., "Fuel," Murray, London, 1875, p. 59.

[7] *Dingler Pol. J.*, 1862, clxv, 378.

[8] Merbach, "Die Anwendung der Erwärmten Gebläseluft in der Metallurgie," Leipsic, 1840, p. 288.

[9] Kultascheff, *Zt. anorg. Chem.*, 1903, xxxv, 187.

[10] Doelter, *op. cit.*, p. 659.

[11] [illegible]

increase of SiO_2, starting with the 6-silicate containing 65 per cent. SiO_2, is uniform throughout and much quicker than that of the sodium silicate. Alkali silicates are glassy, more or less transparent and usually colored by impurities; they are pasty at the formation temperature, but very fluid when superheated.

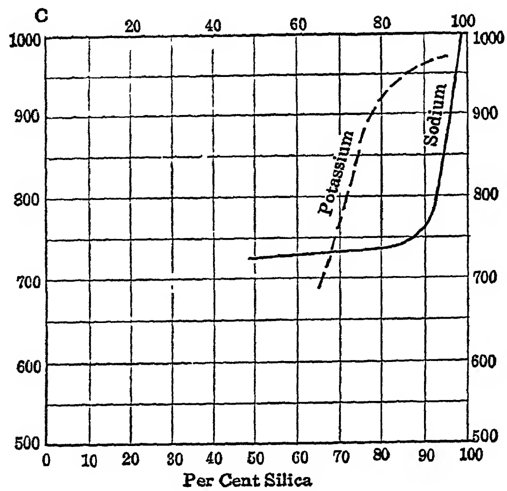


FIG. 275.—Potassium and sodium silicates.

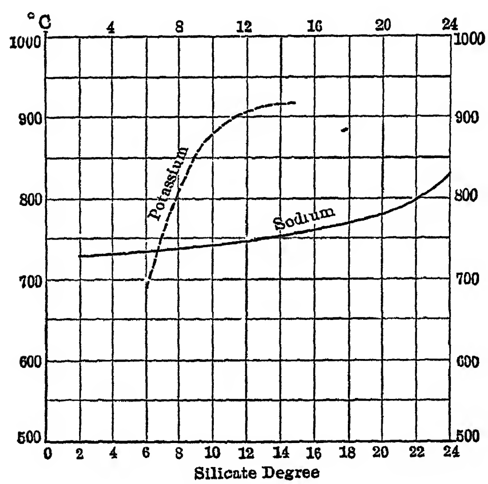


FIG. 276.—Potassium and sodium silicates.

Researches relating to the effects of Na_2SiO_3 upon other silicates have been made by Kultascheff[1] and Wallace.[2]

196. Aluminum Silicates.—Pure aluminum silicates are not formed in metallurgical practice, unless it be in an electric furnace, on account of their high

[1] *Zt. anorg. Chem.*, 1903, XXV, 187.

[2] *Op. cit.*, 1909, LXIII, 1; *Metallurgie*, 1910, VII, 115.

formation temperatures. The older research on the formation temperatures is that by Seger, represented in Figs. 227 and 228. The results of the more recent work by Shepherd and Rankine[1] are shown in Fig. 277; the similarity of the curves is striking.

Aluminum silicates are whitish, have a stony texture and are very viscous.

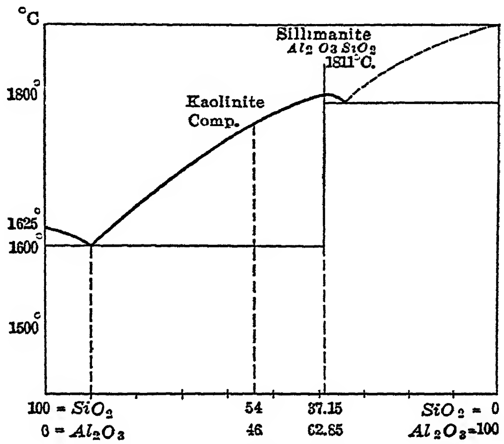


FIG. 277.—Alumina-silica series.

197. Calcium Silicates.—Pure calcium silicates are made as little as aluminum silicates in metallurgical practice and for the same reasons unless we accept the statements of Mathesius[2] and Johnson[3] that in the iron blast-furnace slag, Al_2O_3 is chemically indifferent, acting only as a diluent which affects the viscosity.

Fig. 278 by Day and Shepherd[4] represents the freezing-point curve of the lime-silica series.[5] It starts at *A* with the melting-point of SiO_2 at 1,600° C.;[6] the line *AB* is dotted, as the melt is too viscous to allow an exact location of the freezing-points. At *B* (1,417° C.) is a eutectic (37 per cent. CaO) of tridymite and calcium bi-silicate; at *C* (1,512° C.) the calcium bi-silicate ($CaSiO_3$) which inverts at 1,200° C. into wollastonite; at *D* (1,430° C.) a second eutectic (54 per cent. CaO) of calcium bi-silicate and singulo-silicate; at *E* (2,080° C.) the calcium singulo-silicate (Ca_2SiO_4); at *F* (2,015° C.) a third eutectic (67.5 per cent. CaO) of calcium singulo-silicate and CaO. The curve shows the absence of the supposed compounds $2CaO.3SiO_2$, $4CaO.3SiO_2$ and $3CaO.SiO_2$.

While the freezing-point curve takes precedence over the determinations made by Åkerman, Boudouard, Gredt and others, the curve of total heat of solidification by Åkerman is reproduced, as it is essential for the study of the

[1] *Am. J. Sc.*, 1909, XXVIII, 293.
[2] *Stahl u. Eisen*, 1908, XXVIII, 1121.
[3] *Tr. A. I. M. E.*, 1912, XLIV, 123.
[4] *J. Am. Chem. Soc.*, 1906, XXVIII, 1089.
[5] Schott, A., "Kalksilicate und Aluminate," Doctorate dissertation, Heildelberg, 1906.
[6] This is lower than the figure assumed by Seger (1,800° C.) and found by Kanolt (1,750° C).

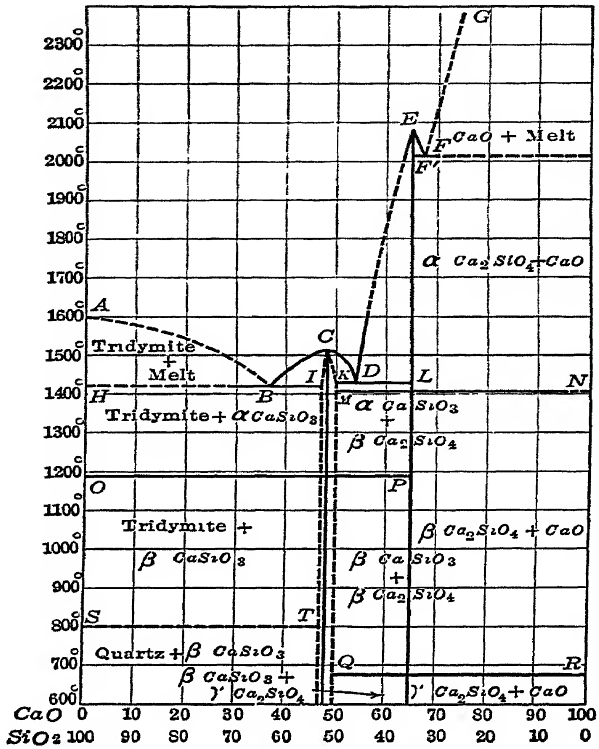


FIG. 278.—Lime-silica series.

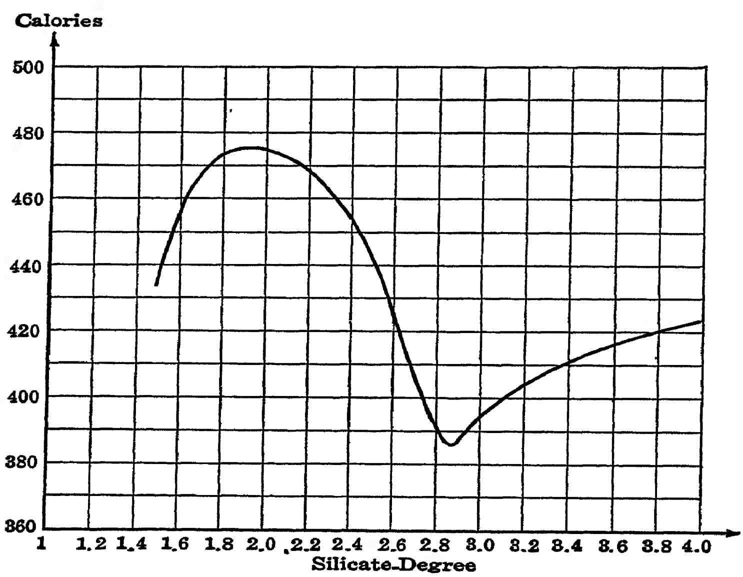


FIG. 279.—Total heats of solidification of lime silicates.

other results obtained by the same method for which there exist no freezing-point curves. Figs. 279 and 280[1] represent the total heats of solidification of calcic silicates with silicate degree, or per cent. SiO_2, as abscissa and calories as ordinate. The general character of the curve resembles that of branch *BCD* in Fig. 278. The maximum (50 per cent. SiO_2) practically coincides with the bi-silicate (51.8 per cent. SiO_2), while the minimum with 60.4 per cent. SiO_2 is some distance away from the eutectic *D* (46 per cent. SiO_2) of Fig. 278.

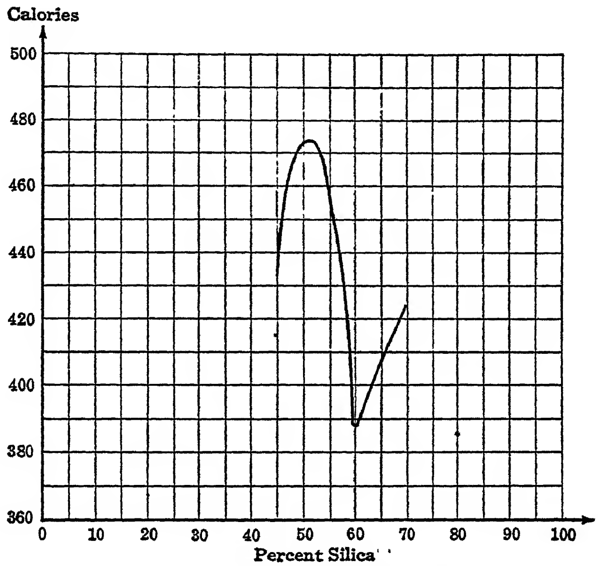


FIG. 280.—Total heats of solidification of lime silicates.

Calcic silicates high in SiO_2 are a bluish-gray and usually have a crystalline texture; from the 1- to the 2.5-silicate they are fluid, between the 2.5- and 4-silicate they are viscous and then again fluid; the 3-silicate appears to be the most viscous. The electric conductivity of $CaSiO_3$ in the liquid state has been studied by Doelter.[2]

198. Calcium-Manganese Silicates.—The influence of MnO upon the fusibility of calcium silicates is shown in Fig. 281. This gives the total heats of solidification of slags ranging from 1- to 3-silicates. The curves are all V-shaped, *i.e.*, up to certain replacements of CaO by MnO varying with the different silicates, the total heat of solidification decreases, reaches a minimum and then again increases. The colors of these slags vary; they are gray with small, and reddish with large, amounts of MnO; the 1-silicate shows green, yellow, and brown colors. All the slags are very fluid and have a crystalline texture. MnO has a tendency to increase the fluidity of all slags.

[1] The curves of Åkerman (*Stahl u. Eisen*, 1886, VI, 281, 387) have been taken from Babu's "Métallurgie Générale," Vol. 1, instead of from the original, as having been replotted they occupy less space and are thus better suited for insertion in the text.

[2] *Tonind. Z.*, 1911, xxxv, 118.

199. Calcium-Magnesium Silicates.—These slags are formed only in exceptional cases, *e.g.*, in smelting in the charcoal iron blast-furnace, but even here they contain small amounts of Al_2O_3, MnO and FeO.

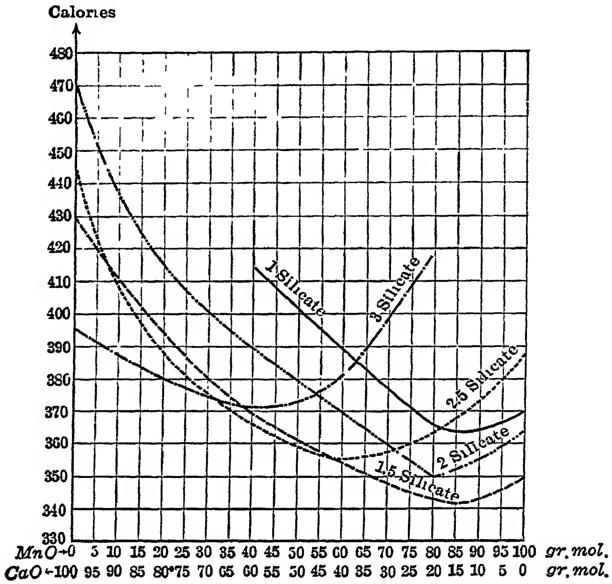


FIG. 281.—Total heats of solidification of certain calcium manganese silicates.

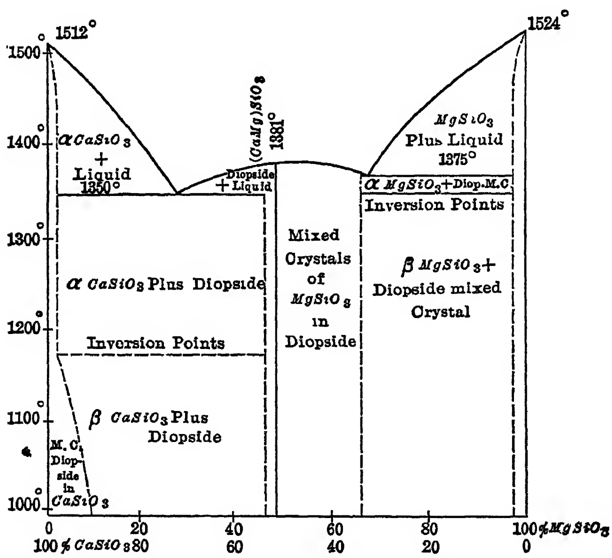


FIG. 282.—Equilibrium diagram of the calcium-magnesium bisilicate series.

The equilibrium diagram of this series of bi-silicates, Fig. 282, has been traced by Allen and White.[1] Starting with $CaSiO_3$ freezing at 1,512° there is shown the

[1] *Am. J. Sc.*, 1909, XXVII, 1.

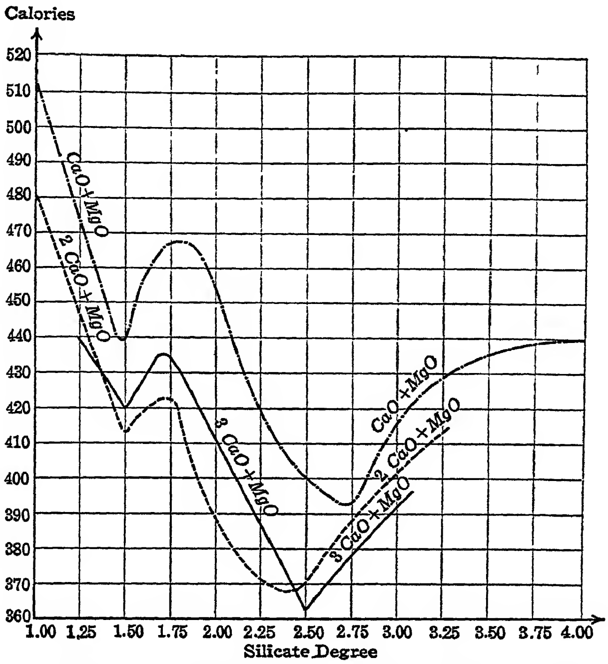


FIG. 283.—Total heats of solidification of certain calcium-magnesium silicates.

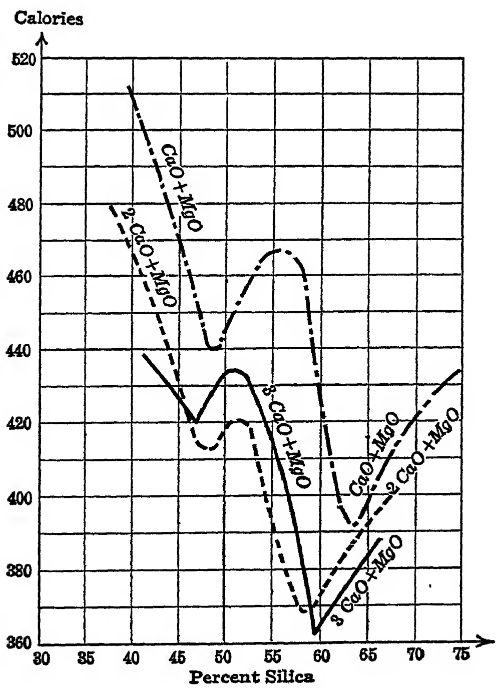


FIG. 284.—Total heats of solidification of certain calcium-magnesium silicates.

first eutectic $CaSiO_3-(CaMg)SiO_3$ with 72 per cent. $CaSiO_3$ freezing at 1,350° C., the chemical compound $(CaMg)SiO_3$ with 53 per cent. $CaSiO_2$ freezing at 1,381° C., the second eutectic $(CaMg)SiO_3-MgSiO_3$ with 32 per cent. $CaSiO_3$ freezing at 1,375° C., and the $MgSiO_3$ freezing at 1,524° C. The transformations shown in the diagram have only mineralogical interest.

Figs. 283 and 284 give the total heats of solidification of slags with definite proportions of CaO and MgO ranging from CaO+1MgO to 3CaO+1MgO; in Fig. 283 the abscissa denotes the silicate degree ranging from 1 to 4, and in Fig. 284 the percentages of SiO_2 ranging from 30 to 75. Fig. 285 gives similar data, the silicate degree of the several curves is fixed and the molecular ratio of CaO

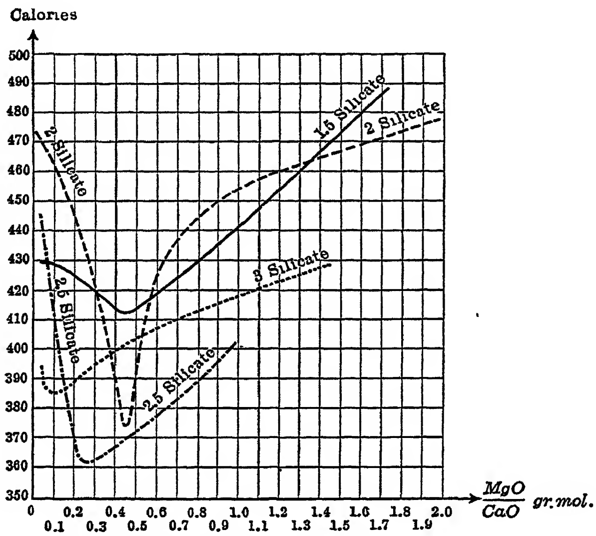


FIG. 285.—Total heats of solidification of certain calcium-magnesium silicates.

and MgO varied from 0 to 2.0. The curves in Figs. 283 and 284 show two eutectic minima, one with the 1.5-silicate; the other lies close to the 2.5-silicate; between them there is a maximum at the 1.75-silicate corresponding to the mineral augite. The lowest melting heat of Ca-MgO silicates with from 2 per cent. MgO (see Fig. 280) to 42 per cent. MgO (from CaO+MgO, Fig. 284) lies near 59 to 63 per cent. SiO_2; the second lowest near 45–48 per cent. SiO_2. Fig. 285 has four V-shaped curves; these show that within a range of the 1.5- to 3-silicate the replacement of CaO by MgO lowers the melting heats to certain points and then raises them beyond those of the pure calcium silicates. The influence of MgO appears to be weaker upon the 1- and 3-silicates than upon the 2- and 2.5-silicates. The thermal values show that in a 2-silicate the lowest melting heat is reached with a molecular ratio MgO : CaO=0.44 (=31.7 by weight) or that the weight of CaO must be approximately three times that of MgO. The slag in question has the composition: SiO_2 54.2, CaO 34.5, MgO 11.3. The same ratio gives the lowest melting heat for the 1.5-silicate, while

slags more acid than the 2-silicate require a ratio MgO : CaO that is lower than 0.44.

Calcium-magnesium slags are gray to grayish-blue; have a crystalline texture up to the 2-silicate; the textuıe becomes more vitreous between the 2- and 3-silicates; the fluidity decreases with the increase of SiO_2; slags with a silicate degree of 2 to 3 are very viscous. The data given in Figs. 283–285 are combined to a triaxial diagram, Fig. 286.[1] The "isocals,"[2] which are drawn 20 cal. apart show 2 minima; the first with SiO_2 55 to 60 per cent. and MgO 5 to 10 per cent. (SiO_2 58, CaO 33, MgO 9 per cent., 358 cal.); the second with

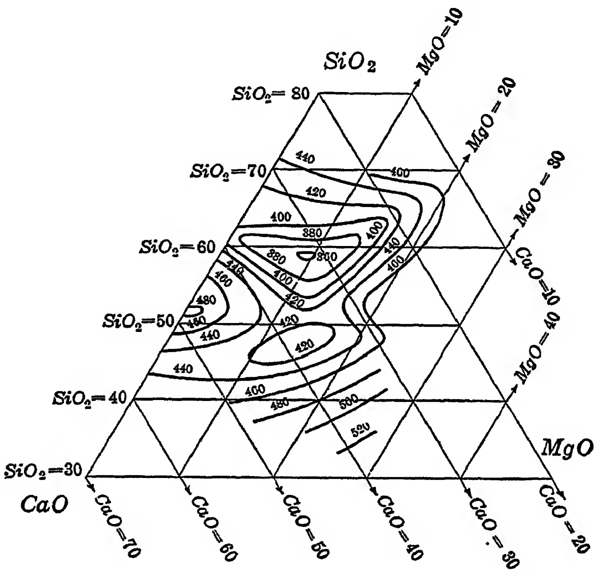


FIG. 286.—Triaxial diagram of calcium-magnesium silicates.

SiO_2 45 to 50 per cent. and MgO 10 to 18 per cent. (SiO_2 47, CaO 38, MgO 15 per cent., 410 cal.).

200. Calcium (Magnesium)-Aluminum Silicates.—Considering that these silicates represent the slags made in the iron blast-furnace, their study is of paramount importance. They have been investigated by Åkerman, Gredt, Boudouard, Rieke,[3] Theussner[4] and others. The work of Shepherd and Rankin[5] and Shepherd-Rankin-Wright [6] deals with cements.

The results of Åkerman[7] are given in detail in Figs. 287 to 291 and are in

[1] Babu, *op. cit.*, I, 515.

[2] Howe's term (*Tr. A. I. M. E.*, 1898, XXVIII, 347) for lines representing equal quantities of heat.

[3] *Stahl u. Eisen*, 1908, XXVIII, 16.

[4] *Metallurgie*, 1908, V, 657.

[5] *J. Ind. Eng. Chem.*, 1911, III, 211; *Stahl u. Eisen*, 1911, XXXI, 1395.

[7] *Am. J. Sc.*, 1909, XXVIII, 293.

[6] *Stahl u. Eisen*, 1886, VI, 281, 387

part summarized in Fig. 292. The curves in Fig. 287 give the total heats of solidification of slags with definite proportions of CaO and Al_2O_3, ranging from 1 CaO : Al_2O_3 to 30 CaO : Al_2O_3, and an increasing silicate degree; the curve

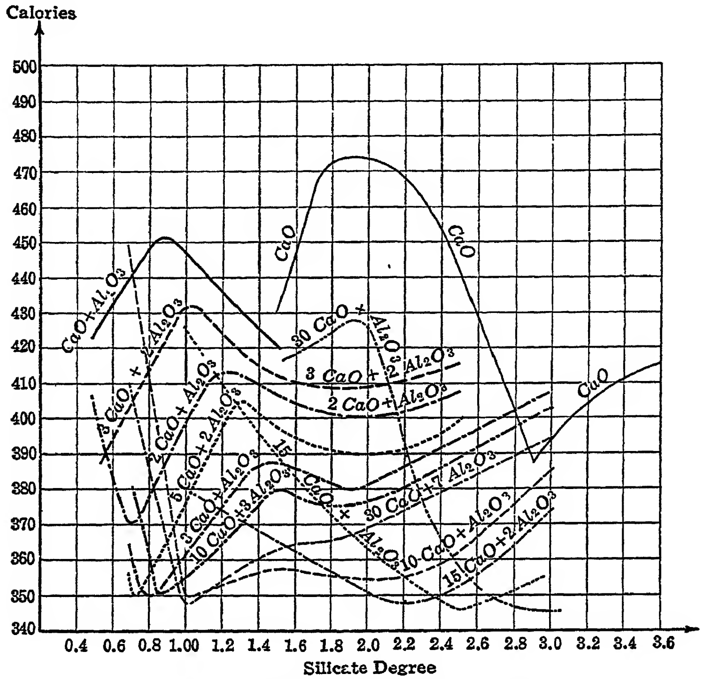


FIG. 287.—Total heats of solidification of certain calcium-aluminum silicates.

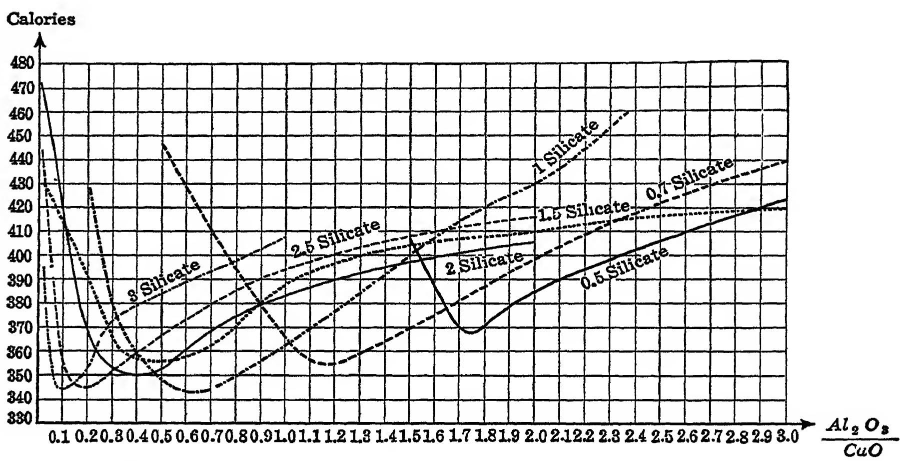


FIG. 288.—Total heats of solidification of certain calcium-aluminum silicates.

for Ca-silicates is added to assist in the study of the effects of Al_2O_3. All the curves show that a small addition of Al_2O_3 lowers the melting heat. With a single exception, every curve has one minimum, some have two. The minima

which have approximately the same total heat of fusion appear to be grouped in two regions, between the 0.6- and the 1-silicates, and beyond the 2-silicate, the curve 15 $CaO+2Al_2O_3$.

$$\frac{\text{O in } Al_2O_3}{\text{O in CaO}}=0.4,\ \frac{\text{mol. wt. O in } Al_2O_3}{\text{mol. wt. O in CaO}}=0.133,\ \frac{\text{wt. O in } Al_2O_3}{\text{wt. O in CaO}}=0.245$$

forming the dividing line. Slags running high in Al_2O_3 have two minima with intervening maximum; the first is more pronounced than the second; their general form suggests two eutectic mixtures (Ca aluminate+Ca-Al silicate and Ca-Al silicates+SiO_2) and a chemical compound (melilite).

Fig. 288 gives the total heats of slags with fixed silicate degrees and increasing ratios of O in Al_2O_3 : O in CaO. Small additions of Al_2O_3 to Ca-silicates quickly reduce the total melting heats and the slags soon reach their minima; the lower the silicate degree the larger is the amount of Al_2O_3 necessary to reach the minimum; after the minimum has been reached the rise of the curve is slower than the preceding fall. The minima have the following percentage compositions.

Table 159.—Minima of Calcium-aluminum Silicates

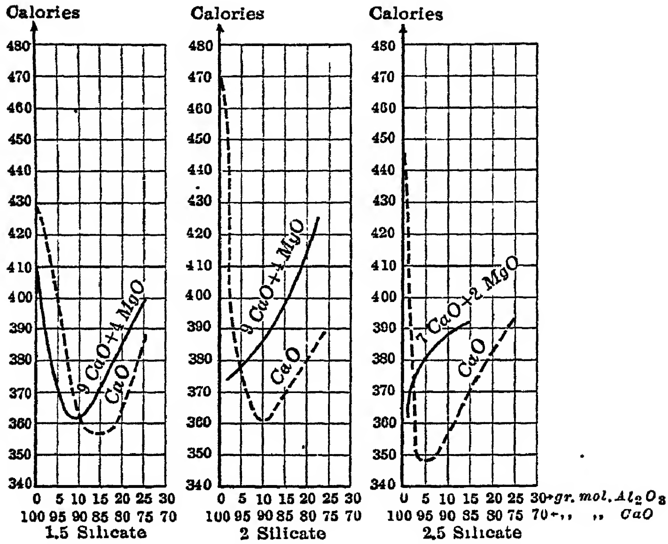
Silicate degree	0 5	0 7	1.0	1.5	2 0	2.5	3.0
SiO_2, per cent....	26 31	32 35	38.51	48 11	54.40	59.06	62.68
CaO, per cent...	35 36	38 81	45 22	39 66	38.29	36.43	35.02
Al_2O_3, per cent..	38.33	28 84	16.27	12 23	7.30	4.51	2.30

Calcium-aluminum silicates are gray; when high in SiO_2 and Al_2O_3 the texture is vitreous and the flow rather viscous; when low in these constituents, the texture is crystalline and the flow fluid. Their electric conductivity in the liquid state has been investigated by Doelter.[1]

In the iron blast-furnace slag CaO is likely to be replaced to some extent by MgO on account of the use of a dolomitic limestone as flux. Figs. 289 to 291 give the total heats of solidification of calcium-aluminum silicates of the 1.5-, the 2.0-, and the 2.5-degree in which CaO has been replaced by MgO in the two proportions: $9CaO+4MgO$ and $7CaO+4MgO$, and the molecular ratio of Al_2O_3 and CaO varied. Fig. 289 shows that the heats of fusion of Ca-Mg-Al 1.5-silicates are lower than those of the corresponding Ca-Al silicates as long as the ratio Al_2O_3 : $(9CaO+4MgO)$ is $<10 : 90$, but higher when $>10 : 90$; Figs. 290 and 291 indicate that with the 2.0- and 2.5-silicates the total heats of fusion of the Ca-Mg-Al silicates are again lower than the Ca-Al silicates, but only as long as the percentage of Al_2O_3 is small; they are very much higher when it is large.

These slags are gray, have a crystalline texture when the percentage of Al_2O_3 is low, but become vitreous with an increase of Al_2O_3. All slags are more or

[1] *Thonind. Z.*, 1911, xxxv, 118.



FIGS. 289 to 291.—Total heats of solidification of certain calcium-magnesium-aluminum silicates.

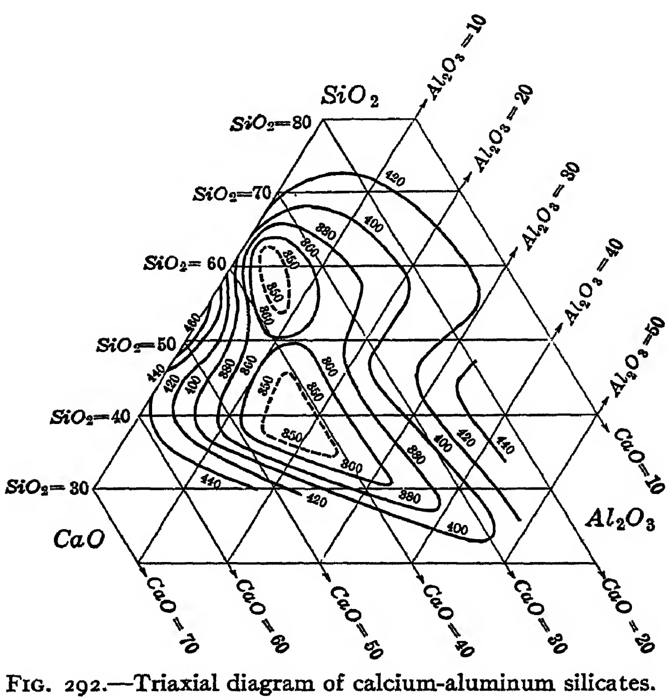


FIG. 292.—Triaxial diagram of calcium-aluminum silicates.

less viscous excepting the 1.5-silicates in which the ratio Al_2O_3 : ($9CaO+4MgO$) is $<5 : 95$.

In Fig. 292 the data of Figs. 287 and 288[1] are combined to a triaxial diagram.[2] The isocals, drawn 20 cal. apart, show two minima with the same total heats of fusion separated by the maximum with 50 per cent. SiO_2. The accurate position of one of the minima is the point represented by SiO_2 56, CaO 36, Al_2O_3 8 per cent., and that of the other by SiO_2 40, CaO 44, Al_2O_3 16 per cent.; charcoal blast-furnace slags belong to the first category and coke blast-furnace slags to the second along the lime side of the valley. Here the isocals 360, 380 and 400 show by their diagonal extension toward the Al_2O_3 side that with an

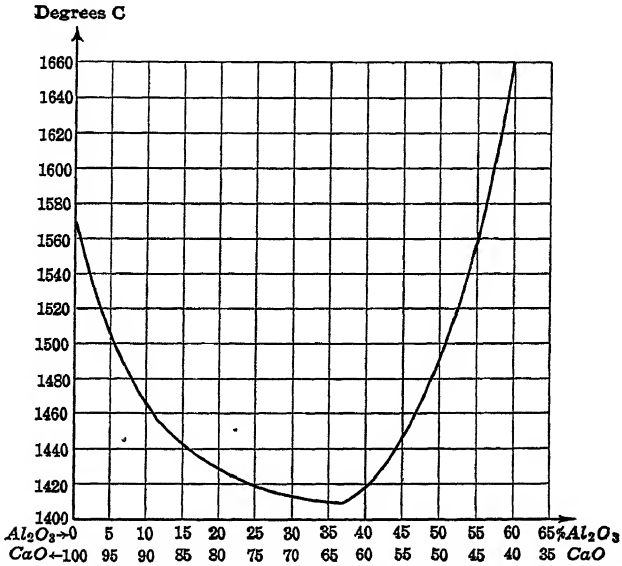


FIG. 293.—Formation temperatures of certain calcium-aluminum singulosilicates.

increase of Al_2O_3 the total heats of fusion remain the same as long as the percentage of SiO_2 is reduced to about one-half of the increase of Al_2O_3.

The effect of MgO upon Ca-Al silicates can be seen by comparing the triaxial diagram of Ca-Mg silicates, Fig. 286, with that of the Ca-Al silicates, Fig. 292. The major axes of the minimum areas of Fig. 286 cross those of Fig. 292 approximately at right angles. The inference to be drawn is that, while the addition of calcium-aluminum silicate to calcium-magnesium silicate lowers the areas of the minima, it will cause a rounding off of the isocals with the result that, starting from the center of a minimum area, equivalent changes of constituents will cause only slight variations in the total heats of fusion. Thus the maximum (SiO_2 51, CaO 36, MgO 13 per cent., 420 cal.), Fig. 286, which separates the two minima of the calcium-magnesium silicates, and the maximum, Fig. 292, of

[1] Babu, *op. cit.*, I, 524.

[2] See also Howe, *Tr. A. I. M. E.*, 1898, XXVIII, 350.
Ashley, *op. cit.*, 1901, XXXI, 855.

the calcium-aluminum silicates (SiO_2 51, CaO 38, Al_2O_3 12, 360 cal.), have approximately the same percentages of SiO_2 and CaO and show a difference of only 60 cal.

The formation temperatures of the calcium-aluminum silicates and the effect of MgO upon the one which showed the greatest fusibility have been determined by Gredt.[1] His curve is reproduced in Fig. 293. Starting with Ca-singulo-silicate which forms at 1,570° C.,[2] the formation temperatures fall at first rapidly and then more slowly with the additions of Al_2O_3 until the minimum is reached at 1,410° C. with the ratio of 0.608, by weight, or with approximately 32 Al_2O_3 : 68 CaO; they then rise again quickly. A comparison of these forma-

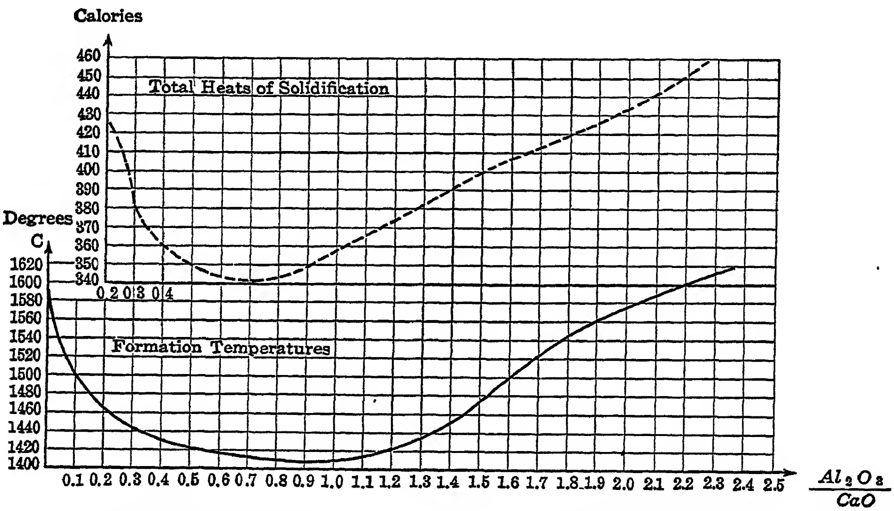


FIG. 294.—Formation-temperatures and total heat of solidification of certain calcium-aluminum silicates compared.

tion temperatures with the total heats of solidification of Åkerman, as plotted by Babu,[3] Fig. 294, shows a general similarity of the results obtained by the two methods. There is, however, a difference in the loci of the minima; Gredt's minimum lies at the O-ratio 0.9 and that of Åkerman at 0.7. Åkerman[4] has tabulated some of Gredt's results and the melting temperatures of similar mixtures calculated from his calorimetric figures. These data also show a general resemblance in results, but bring out many differences in detail. The higher temperatures are explained[5] by the fact that temperatures assigned to the Seger cones have been found by later calibrations to be too high.[6]

The effect of MgO upon the calcium-aluminum singulo-silicate of lowest

[1] *Stahl u. Eisen*, 1889, IX, 756.

[2] Compare Fig. 278.

[3] *Op. cit.*, I, 528.

[4] *Stahl u. Eisen*, 1890, X, 424; *J. I. and St. I.*, 1890, II, 760. Howe, *Eng. Min. J.*, 1890, XLIX, 637, 675.

[5] Boudouard, *J. I. and St. I.*, 1905, I, 339.

[6] Compare Tables 67 and 68.

formation temperature is shown in Fig. 295. Starting with 1,410° C. (SiO_2 40.0, CaO 37.3, Al_2O_3 22.7), the formation temperature is seen to fall gradually with the replacement of CaO by MgO until the minimum of 1,350° C. has been reached at the ratio (by weight) of MgO : CaO = 0.625 or of 37.5 MgO : 65.2 CaO, the slag having the composition SiO_2 42.0, CaO 20.8, MgO 13.2, Al_2O_3

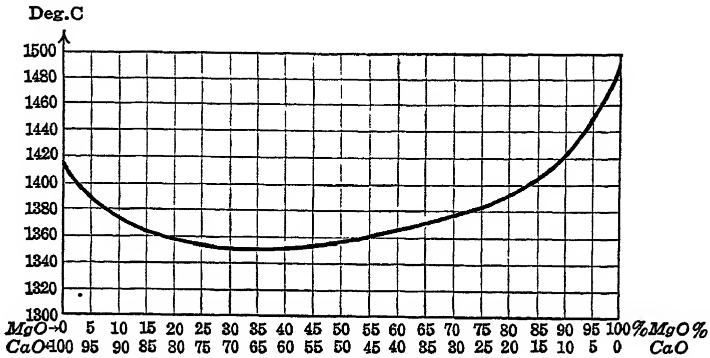


Fig. 295.—Calcium-aluminum silicate with the replacement of calcium by magnesium.

24.0; the temperature then rises again gradually until the pure magnesium-aluminum singulo-silicate is reached at 1,497° C. Gredt's idea of calculating the effects of MgO upon the other 1-silicates of his series, upon the assumption that

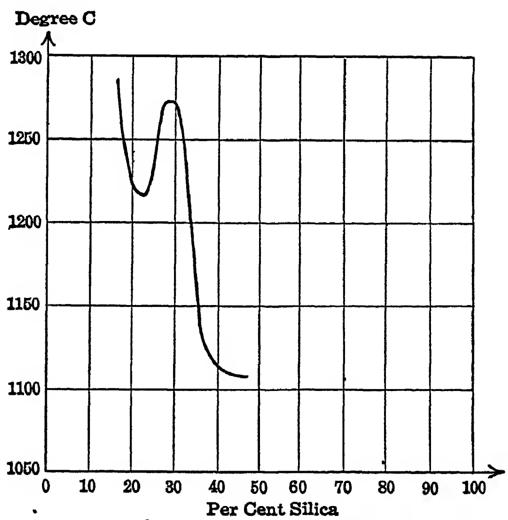


Fig. 296.—Formation temperatures of ferrous silicates (Hofman).

proportional replacements will have similar effects upon temperatures, does not appear to be justifiable considering the results obtained by Åkerman.

The more recent and elaborate work of Boudouard[1] upon formation tempera-

[1] §194; *Rev. Gén. Chim., Pure et Appliquée*, 1906, IX, 455; *Compt. rend.*, 1907, CXLIV, 1047; *Metallurgie*, 1907, IV, 816.

tures of calcium-aluminum silicates substantiates in a general way the conclusions arrived at by Åkerman and Gredt.

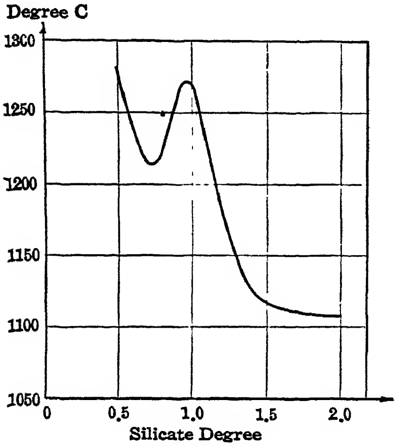


FIG. 297.—Formation temperatures of ferrous silicates (Hofman).

201. Iron Silicates.—These silicates are formed in puddling pig iron, and the reheating of muckbar and steel ingots, puddle cinder holding FeO, and Fe_2O_3 in igneous solution. Figs. 296 to 297 give the data of Hofman[1] plotted by Babu.[2] The curves show that in the range of the investigation the lowest formation temperature, 1,110° C., is that of the bi-silicate ($FeO.SiO_2$) with 45.45 per cent. SiO_2 and the highest, 1,280° C. that of the sub-silicate 4 $FeO.SiO_2$ with 17 per cent. SiO_2. Between these extremes there is a maximum at 1,270° Si. with SiO_2 29.20 per cent., which corresponds to the mineral fayalite, $2FeO. SiO_2$, and a minimum which may be a eutectic.

According to the investigations of H. Steffe[3] represented in Fig. 298, the formation temperature in the range $4FeO.SiO_2-4FeO.3SiO_2$ shows little variation, and then rises suddenly with an increase in SiO_2. Of the two curves,

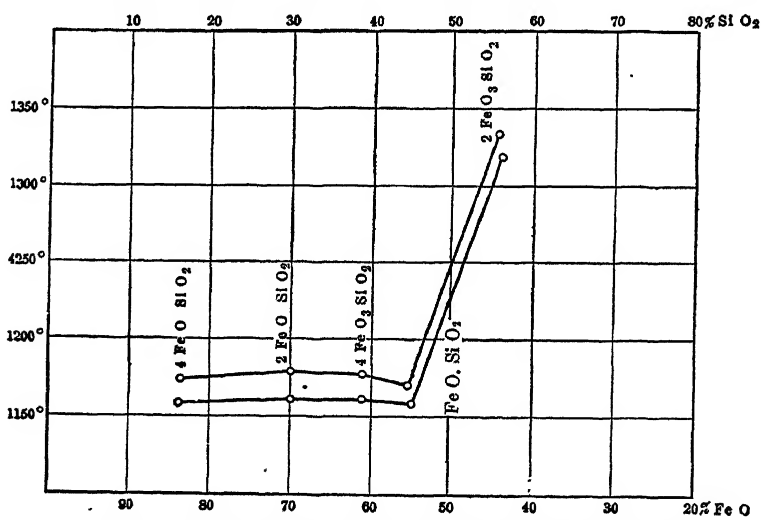


FIG. 298.—Formation temperatures of ferrous silicates (Steffe).

the lower represents the temperature at which the mixtures began to sinter, and the upper the one at which they were completely fused. The determina-

[1] *Tr. A. I. M. E.*, 1889, XXIX, 700.
[2] "Métallurgie Générale," Vol. 1, pp. 503 and 504.
[3] *Doctorate Thesis*, Berlin, 1908.

tions of Steffe made in an electric resistance furnace in a current of N, the temperatures being measured with a Le Chatelier pyrometer, are more accurate than those of Hofman made in a gas-furnace and with Seger cones. Nevertheless the latter, while scientifically not correct, give data which resemble

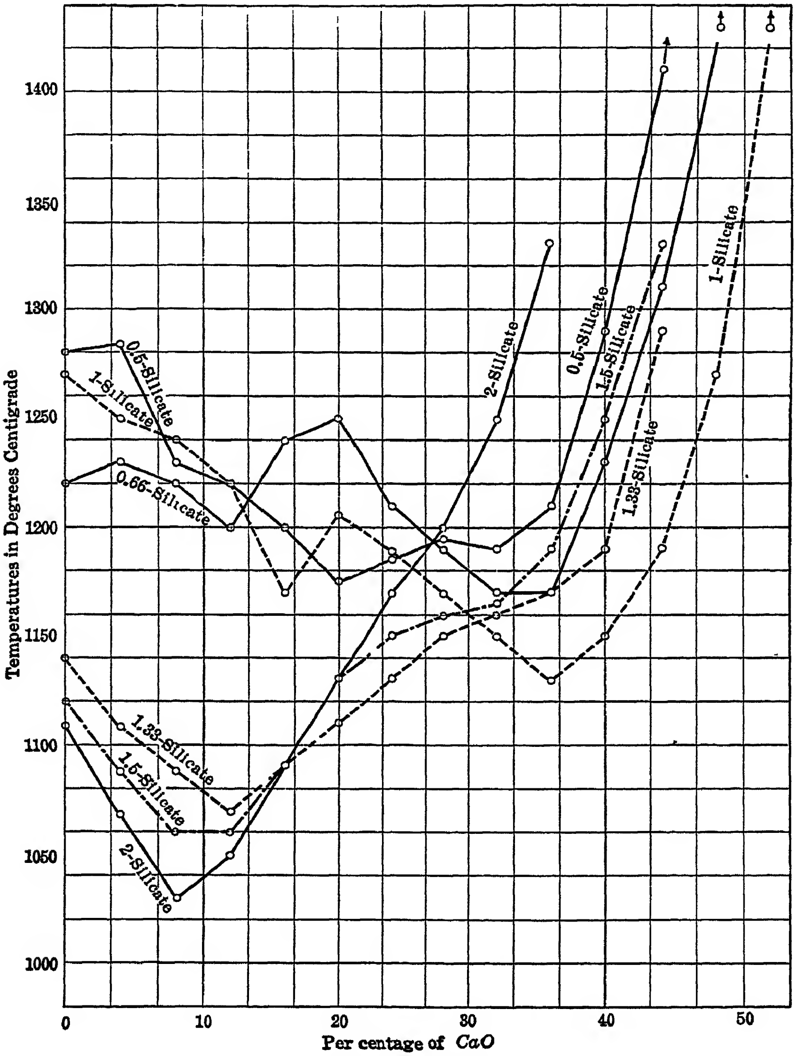


FIG. 299.—Formation-temperatures of some ferro-calcic silicates (Hofman).

furnace conditions and have been borne out in a general way by furnace practice, especially in the case of pyritic smelting where formation temperatures are of such vital importance. Ferrous slags are more or less viscous; those of a silicate degree below the 1-silicate are more fluid than those above it.

202. Iron-Calcium Silicates.[1]—These silicates are characteristic for slags

[1]Hofman, *Tr. A.I.M.E.*, 1899, XXIX, 682; see also comment by Ashley, *op. cit.*, 1901, XXXI, 855.

TABLE 160.—IRON-CALCIUM SILICATES

Formulæ of silicates	O in bases to O in acids		Per cent.													
$5RO\ SiO_2$	5 : 2	FeO	85 70	81.55	77 38	73.23	69.05	64 90	60 74	56.57	52 41	48 25	44.10	39.92	35.67	31.60
		CaO	0.	4.	8.	12.	16	20.	24.	28.	32.	36.	40.	44.	48.	52.
		SiO_2	14.30	14 45	14.62	14.77	14.95	15.10	15.26	15.43	15.60	15 75	15 90	16 08	16 33	16.40
$4RO\ SiO_2$	4 : 2	FeO	82.80	78.60	74.40	70.23	66.01	61.81	57.61	53.41	49 22	45 02	40 81	36 61	32 41	28 20
		CaO	0.	4.	8.	12.	16.	20.	24.	28.	32	36.	40.	44	48	52
		SiO_2	17.20	17.40	17.59	17.77	17.99	18.19	18.39	18.59	18 78	18 08	19 19	19 30	19.59	19.80
$3RO\ SiO_2$	3 : 2	FeO	78.30	74.05	69 80	65.51	61 30	57.05	52.80	48.55	44 30	40 06	35 80	31 55	27 52	23.05
		CaO	0.	4.	8.	12.	16.	20.	24.	28.	32.	36	40	44.	48.	52.
		SiO_2	21.70	21.95	22 20	22.49	22 70	22.95	23.20	23.45	23 70	23 94	24.20	24.45	24.48	24.95
$5RO\ 2SiO_2$	$\frac{5}{2}$: 2	FeO	75.00	70.71	66.43	62.15	57 86	53.57	49.28	45.	40.71	36.43	32 15	27.86	23.50	19 28
		CaO	0.	4.	8	12.	16.	20	24.	28.	32.	36	40	44.	48.	52.
		SiO_2	25.00	25.28	25.57	25.85	26 14	26 43	26.72	27.	27.29	27 57	27 86	28 14	28 43	28.72
$2RO\ SiO_2$	2 : 2	FeO	70.80	66.25	61.91	57.58	53.24	48 90	44.60	40.30	35.90	31.70	27.30	22 90	18 56	14 21
		CaO	0.	4.	8.	12.	16	20.	24.	28.	32	36	40.	44	48.	52
		SiO_2	29.20	29.75	30.09	30 42	30.76	31.07	31.40	31 70	32 10	32.30	32 70	33 10	33 44	33 70
$5RO\ 3SiO_2$	$\frac{5}{3}$: 2	FeO	66.66	62.26	57.92	53.58	49.24	44 93	40.56	36 22	31 88	27 54	23 20	18 86	14 52	10 10
		CaO	0.	4.	8.	12.	16.	20.	24.	28.	32.	36	40	44.	48	52.
		SiO_2	33.33	33.74	34.08	34.42	34.76	35.07	35.44	35.78	36.12	36 46	36 80	37 14	37 48	37 9
$3RO\ 2SiO_2$	$\frac{3}{2}$: 2	FeO	64.30	60.00	55.60	51.20	46.70	42 25	37.84	33 44	29 04	24 63	20 22	15 80	11 40	6 92
		CaO	0.	4.	8.	12.	16.	20.	24.	28.	32.	36.	40	44.	48	52
		SiO_2	35.70	36 05	36.40	36.80	37.30	37.75	38.16	38.56	38 95	39 37	39.78	40.20	40 60	41.0
$4RO\ 3SiO_2$	$\frac{4}{3}$: 2	FeO	61.54	57.10	52.66	48.22	43 78	39.34	34.89	30.46	26 01	21 58	17 13	12 69	8 .6	3 80
		CaO	0.	4.	8.	12.	16.	20	24	28	32.	36.	40	44	48.	52 8
		SiO_2	38.46	38.90	39.34	39.78	40.22	40.66	41 11	41.54	41 99	42 42	42 87	43 31	43.75	44 10
$5RO\ 4SiO_2$	$\frac{5}{4}$: 2	FeO	60 00	55.54	51.08	46.63	42.17	37.72	33.26	28.80	24 36	19 89	15 43	10 97	6 52	2 06
		CaO	0.	4.	8	12.	16.	20.	24.	28.	32	36	40	44	48.	52
		SiO_2	40.00	40.46	40.92	41.37	41.83	42.28	42.74	43.20	43 64	44.11	44 57	45.03	45 48	45 94
$RO\ SiO_2$	1 : 2	FeO	54.55	50.00	45.47	40.96	36.44	31.98	27.43	22.81	18.40	13.89	9.37	4 86	0 35	0 00
		CaO	0.	4.	8.	12	16.	20.	24.	28.	32.	36	40	44.	48	48 27
		SiO_2	45 45	46.00	46.53	47.04	47.56	48.02	48.57	49.19	49.60	50.11	50 63	51.14	51 65	51 73
$4RO\ 5SiO_2$	$\frac{4}{5}$: 2	FeO	48.98	44.40	39.91	35.23	30.65	26 07	21.49	16 90	12.41	7.74	3.16	0 00	..	
		CaO	0.	4.	8.	12.	16.	20.	24.	28.	32.	36.	40	42.75		..
		SiO_2	51.02	51.60	52 09	52 77	53.35	53 93	54.51	55 10	55.59	56 26	56 84	57 25		
$3RO\ 4SiO_2$	$\frac{3}{4}$: 2	FeO	47.37	42.76	38.19	33.58	28.98	24.37	19.77	15 16	10.55	5.95	1 35	0 00		
		CaO	0.	4.	8.	12.	16.	20	24.	28.	32.	36.	40.	41 18		
		SiO_2	52.63	53.24	53.81	54.42	55.02	55.63	56.23	56 84	57.45	58 05	58 65	58 82		
$2RO\ 3SiO_2$	$\frac{2}{3}$: 2	FeO	44.44	39.77	35.14	30.51	25.88	21.24	16.60	11 96	7.32	2 68	0.00			
		CaO	0.	4.	8.	12	16.	20.	24.	28.	32.	36.	38 36			
		SiO_2	55.55	56.23	56.86	57.49	58.12	58.76	59.40	60.04	60.58	61 32	61 64			
$3RO\ 5SiO_2$	$\frac{3}{5}$: 2	FeO	41.86	37 05	32.41	27.76	23.13	18.47	13.83	9.18	4.53	0.00				
		CaO	0.	4.	8.	12.	16.	20.	24.	28.	32.	35.				
		SiO_2	58.14	58.95	59 59	60.24	60.87	61.53	62.17	62.82	63.47	64.10				
$RO\ 2SiO_2$	$\frac{1}{2}$: 2	FeO	37.50	32.79	28.08	23.37	18.66	13.95	9 24	4.53	0.00					
		CaO	0.	4.	8.	12.	16.	20.	24.	28.	31.82					
		SiO_2	62.50	63.21	63.92	64 63	65.34	66 05	66.76	67.37	68.18					

made in smelting Pb, Cu, Ni and other non-ferrous ores. The FeO is frequently replaced more or less by MnO; the CaO by MgO, BaO and ZnO; the SiO_2 and CaO by Al_2O_3. A list of the different iron-calcium silicates is given in Table 160. The table is useful in studying the range of composition within which lie slags made in various smelting operations.

Fig. 299 gives the formation temperatures of iron-calcium silicates ranging from the 0.5- to the 2-silicate degree. The curves in a general way show that a progressive replacement of FeO by CaO has a tendency to lower the formation temperatures and then to raise them beyond the starting point. The curves form two groups. The first, containing the 0.15-, 0.66-, and 1-silicates, shows slight irregularities, due perhaps to the peroxidation of small amounts of Fe. Each curve, however, has two minima; the first lying within the range of 10 to 20 per cent. CaO has higher formation temperatures than the second with 32 to 36 per cent. CaO; the minima are separated by maxima.

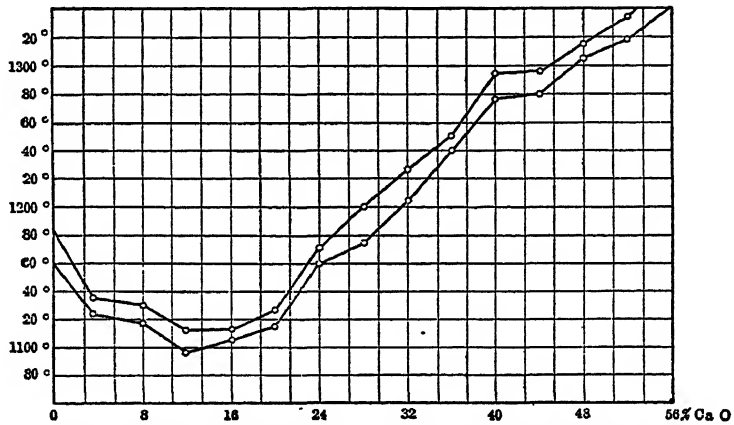


FIG. 300.—Formation-temperatures of some ferro-calcic silicates (Steffe).

The second group, containing the 1.33-, 1.5-, and 2-silicates, is more regular. The fall in temperature is rapid, reaches the single minimum between 8 and 12 per cent. CaO, and is followed by a quick rise which is more rapid the greater the silicate degree.

In Fig. 300 are reproduced the curves by Steffe (*loc. cit.*). They show a greater regularity than those of Hofman; while the latter finds with the singulo-silicate the lowest formation temperature with about 35 per cent. CaO, Steffe finds it near 12 per cent. Furnace practice bears out the jog at about 16 per cent. CaO in Hofman's curve, as well as the decrease in the formation temperature beyond 12 per cent. CaO. The cause of this is not known.

Fig. 301 shows the effect an increase in the percentage of SiO_2 has upon the formation of an iron-calcium silicate in which the ratio FeO : 2CaO = 2 : 1 is kept constant. From the 0.5- to the 1.25-silicate there is a slight fall of 10° C.; this fall increases at first slowly and then more quickly until the minimum is reached at 1,110° C. with the 2.50- to 2.75-silicate; the curve then rises quickly

to 1,130° C. with the 3-silicate and, at last, abruptly ascends the temperature scale. While a low formation temperature is desirable, a slag of the 2.50- to 2.75-degree cannot well be made, as it is too viscous.

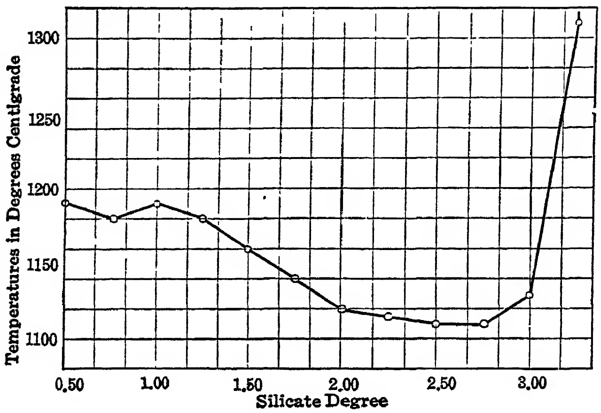


FIG. 301.—Effect of increase of silica upon the formation temperature of a ferro-calcic silicate.

The results of Figs. 299 and 301 have been combined by Babu[1] into a triaxial diagram, Fig. 302. The isotherms, drawn 50° C. apart, show a minimum area of 1,030° C. between SiO_2 40 to 50 per cent. and CaO 5 to 15 per cent. As the CaO

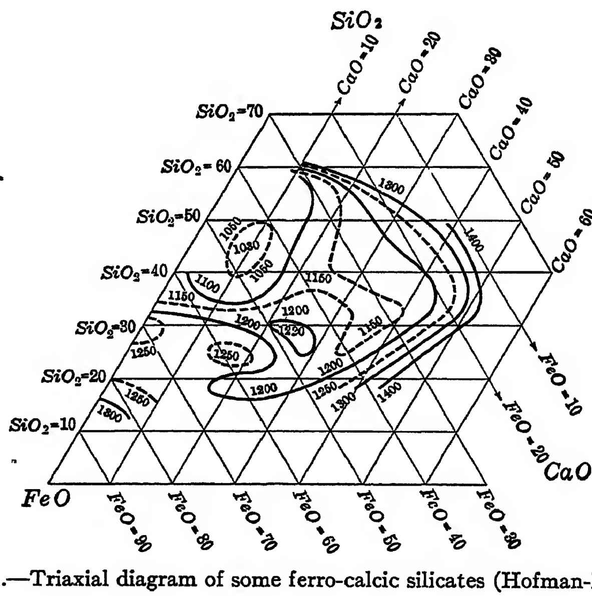


FIG. 302.—Triaxial diagram of some ferro-calcic silicates (Hofman-Babu).

increases at the expense of the FeO there is at first (1,100 to 1,200° C.) a slow and then (1,200 to 1,400° C.) a quick rise in formation temperature. The two

[1] *Op. cit.*, p. 536; see also Ashley's plot in *Tr. A. I. M. E.*, 1901, XXI, 855, using an isosceles right-angle triangle.

large areas 1,100 to 1,150° C. and 1,150 to 1,200° C., show that there exists a latitude in the choice of slags having the same formation temperatures.

In smelting in the blast-furnace, sub-silicates are not often made; 1-silicates are most common. While 1.5- and 2-silicates are occasionally formed in blast-furnace work, they belong rather to reverberatory furnace practice. The 1-silicates of the lead smelter are found in the neighborhood of the first minimum; in copper smelting there is a greater freedom of choice as regards silicate degree and replacement of FeO by CaO on account of the smaller danger of slagging Cu in the presence of S than there is in the case of Pb. Many copper metallurgists at present figure Al_2O_3 as an acid and make bi-silicate slags on this basis. Sub silicates are, as a rule, very fluid and slightly magnetic, the 1-silicates reasonably so; 2-silicates are usually rather viscous and are likely to carry uncombined SiO_2. The presence of CaO, within limits, increases the fluidity of iron silicates.

203. Replacement of Silicate Constituents.—The effects of the replacement of oxides have been studied by Hofman[1] on a 1-silicate containing SiO_2 32.10, FeO 35.90, CaO 32 per cent., forming at 1,150° C.

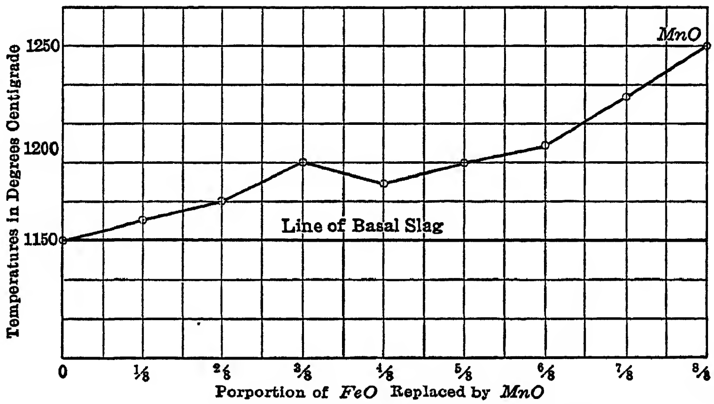


FIG. 303.—Effect of replacement of FeO by MnO.

(1) REPLACEMENT OF FEO BY MNO, Fig. 303.—The formation temperature is seen to increase proportionally with the replacement, rising from 1,150 to 1,250° C. The curve can be considered as a straight line, *i.e.*, no eutectic mixtures or chemical compounds are formed. The fluidity of the slags increases with the substitution of MnO for FeO; the color remains unchanged, and the luster is slightly glassy; small particles are attracted by the magnet.

The series MnO-SiO_2 has been studied by Doernickel.[2]

(2) REPLACEMENT OF CAO BY MGO, BAO, ZNO, Fig. 304.—Curve I, CaO by MgO: The formation temperature of 1,150° C. iises very decidedly from the start and continues to do so to 1,400° when all the CaO has been replaced by MgO. The experimental evidence agrees with the fact noted in practice, that

[1] *Loc. cit.*

[2] *Metallurgie*, 1911, VIII, 201; *Min. Ind.*, 1911, XX, 484.

the presence of MgO in a charge increases the fuel consumption in a blast-furnace. The fluidity of the slag is smaller than when MgO is absent; the grayish-black color becomes lighter with an increase in MgO; the luster is vitreous; the slag is magnetic, less so when MgO prevails than when does CaO.

Curve II, CaO by BaO: The formation temperature of 1,150° C. is lowered gradually and evenly, reaching a minimum of 985° with 6/8 replacement and then rises again slightly. The presence of BaO is not favored in blast-furnace

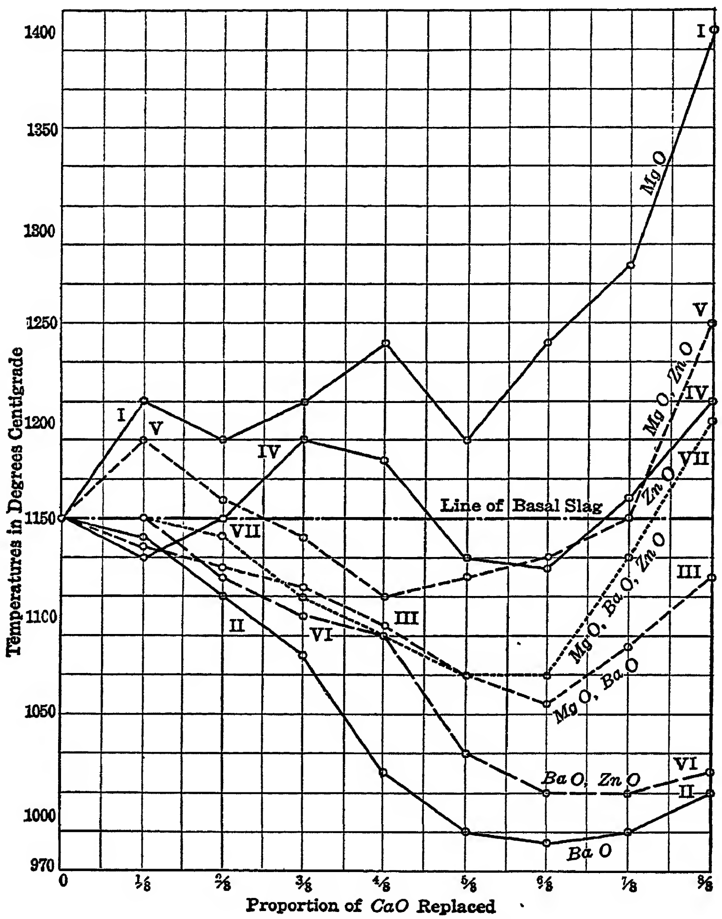


FIG. 304.—Effect of replacement of CaO by MgO, BaO, ZnO.

slags, as its high specific gravity causes them to separate imperfectly from matte. The slags are not very fluid, are opaque, steel gray to black, have a vitreous luster and are attracted by the magnet, especially when rich in BaO.

Curve III, CaO by MgO and BaO: The refractory character of MgO is more than counteracted by the powerful fluxing effect of BaO; the formation temperature falls uniformly to the minimum of 1,055° C. with 6/8 replacement and then rises again, but remains below the basal line of 1,150° C. While the

formation temperature has been lowered, the fluidity has not been increased as the slags do not flow well; the color remains unchanged, the luster is slightly vitreous and the material is still magnetic.

Curve IV, CaO by ZnO: Some metallurgists[1] believe that in a slag ZnO is present as a silicate, others[2] that it is held in igneous solution. It has been proved experimentally in many instances that nearly all of the ZnO can be removed by treatment of powdered slag with dilute H_2SO_4. The irregularity of the curve falling beneath and rising above the formation temperature of the basal slag seems to indicate that there are conditions, not yet understood, which promote or obstruct the scorification of ZnO. When the difficultly fusible $ZnSiO_x$[3] is to be formed, the formation temperature may be expected to rise; when ZnO is only held in igneous solution, it ought to fall on account of the increased percentage of FeO which has a strong dissolving power for ZnO. This oxide is undesirable in a slag, as it decreases the fluidity; zinckose slags are opaque, dark-brown to black, have a luster submetallic to vitreous, and are attracted by the magnet.

Curve V, CaO by MgO and ZnO: The curve is irregular; it corroborates the practical experience of smelting men that MgO and ZnO together intensify the undesirable properties of each other. The fluidity varies, the least viscid is the slag in which about 1/2 of the CaO has been replaced. The slags are opaque, the color is steel-gray to black, the luster submetallic to vitreous; they are attracted by the magnet.

Curve VI, CaO by BaO and ZnO: The curve is smooth, the irregularity of the ZnO curve having disappeared, the general trend is similar to that of the BaO curve, only the temperatures lie somewhat higher. The slags do not flow readily, they are opaque, the color ranges from steel-gray to brownish-black, the luster is submetallic to vitreous, the magnet attracts slags rich in CaO more decidedly than when rich in BaO and ZnO.

Curve VII, CaO by MgO, BaO and ZnO: Again the curve is smooth, the rise and fall of the MgO-ZnO curve have been eliminated by the addition of BaO; the refractory effect of MgO is seen by comparing curves VI and VII.

Curves I–VII show that Plattner's statement "the formation temperature of a silicate slag decreases with the number of bases" has no general application.

204. Rôle of Alumina in Iron-Calcium Silicates.—The influence of Al_2O_3 upon the formation and melting temperatures of ferrous slags made in smelting lead, copper, nickel, and other similar ores is of less importance than of the non-ferrous slags made in the iron blast-furnace; nevertheless occasions arise not infrequently when its presence has to be considered. It is difficult to fix the status

[1] Iles, *School Min. Quart.*, 1897–98, XIX, 197; also "Lead Smelting," Wiley, New York, 1902, p. 32.

Hutchins, *Eng. Min. J.*, 1903, LXXVI, 959.

Bretherton, *Eng. Min. J.*, 1908, LXXXVI, 483.

Bartlett, *Min. Ind.*, 1896, V, 629.

[2] Austin in "Pyrite Smelting," Eng. Min. J., New York, 1905, p. 205.

[3] $ZnSiO_3$ melts at 1,479° C.; Zn_2SiO_4 at 1,488° C.; Stein, *Zt. anorg. Chem.*, 1907, LV, 159, measurement with Wanner pyrometer.

of Al_2O_3, as with an excess of SiO_2 it may play the rôle of a base forming aluminum silicate, and with an excess of base that of an acid forming an aluminate (spinel). The conditions under which one or the other takes place have not been definitely settled. Most iron blast-furnace men and many copper metallurgists consider Al_2O_3 as an acid and figure their charges accordingly;[1] Henrich[2] does the same for ferrous slags. The opinions of many lead and copper metallurgists[3] differ as to the conditions according to their several experiences.

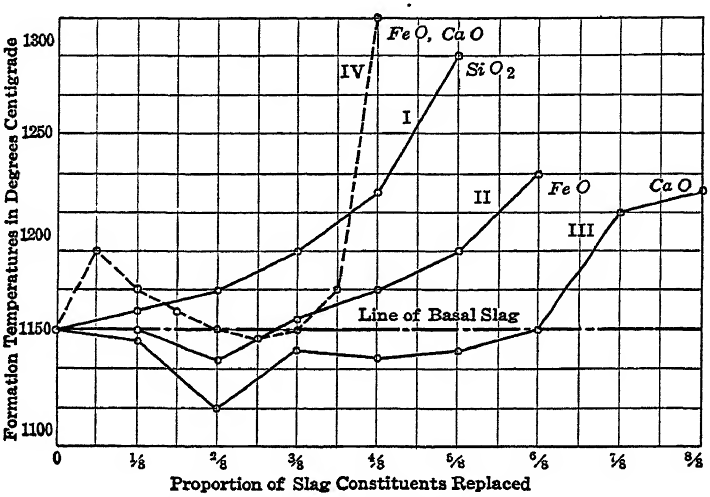


FIG. 305.—Effect of replacement of slag-constituents by Al_2O_3.

Austin[4] holds that Al_2O_3 is indifferent with slags up to SiO_2 36 per cent., being held in igneous solution and acting as a stiffening ingredient, while with higher percentages of SiO_2 it acts as a base. The opinions of Mathesius[5] and Johnson[6] for iron blast-furnaces are similar. The laboratory experiments of Hofman,[7] represented in Fig. 305, give some evidence upon the subject by replacing the constituents of a ("basal") slag, severally and together by Al_2O_3.

Curve I, SiO_2 by Al_2O_3: This replacement raises the formation temperature of the basal slag in an increasing ratio. The slags show lack of fluidity, are

[1] *Eng. Min. J.*, 1908, LXXXVI, 270 (Shelby), 483 (Bretherton), 730 (Koch), 1111 (Hooper), 1264 (Shelby).

[2] *Op. cit.*, 1886, XLII, 40, 128, 240; 1890, L, 744.

[3] Hahn, *Min. Res. U. S.*, 1882, p. 328.
Iles, *op. cit.*, 1883–84, p. 433, also "Lead Smelting," Wiley, New York, 1902, p. 117.
Schneider, *Tr. A. I. M. E.*, 1883, XI, 57.
Howe, *Eng. Min. J.*, 1883, XXVI, 306.
Peters, *ibid.*, 322.
Stone, *ibid.*, 323.
Hofman, "Lead," Scientific Publ. Co., New York, 1889, 288.

[4] *Eng. Min. J.*, 1904, LXXVIII, 253.

[5] *Stahl u. Eisen*, 1908, XXVIII, 1121.

[6] *Tr. A. I. M. E.*, 1912, XLIV.

[7] *Tr. A. I. M. E.*, 1899, XXIX, 717.

opaque and have a color ranging from dark-gray to black, the luster is dull to slightly vitreous, and the smaller the substitution the greater is the attractive power of the magnet.[1]

Curve II, FeO by Al_2O_3: The formation temperature is little changed up to a substitution of 3/8 of the FeO and then rises. The slags are fairly fluid, especially those with a substitution of 3/8 to 1/4 of the FeO; they are opaque, the color ranges from dull-brown to dark-gray, the luster is slightly vitreous; they are magnetic.

Curve III, CaO by Al_2O_3: The formation temperature sinks below that of the basal slag and remains lower until 3/4 of the CaO has been replaced, when it rises suddenly. This shows that, within limits, figuring Al_2O_3 as replacing CaO in a slag is a justifiable proceeding; MgO may not be substituted for CaO, as MgO and Al_2O_3 are incompatible (Vogt). A combination of curves I and III suggests that if part of the SiO_2 and CaO are replaced by Al_2O_3, the formation temperature will remain unchanged, a proceeding which is very common in smelting highly aluminous lead ores.

Curve IV, FeO and CaO by Al_2O_3: The curve is very irregular and suggests that this dual replacement has no justification in practical smelting; similarly ZnO may not be figured in a slag as replacing Al_2O_3 (Vogt).

Melting-points of Silicates.—The following melting-points of silicates by Stein[2] taken with a Wanner pyrometer are of interest. Several of them do not agree with more accurate determinations given in other parts of this treatise. The data in deg. C. are: quartz viscid at 1,600, fluid at 1,750; $MgSiO_3$ 1,565; $CaSiO_3$ 1,512; $FeSiO_3$ 1,500–1,550; $MnSiO_3$ 1,470–1,500; $ZnSiO_3$ 1,479; $SrSiO_3$ 1,287; $BaSiO_3$ 1,368; $2MgO.SiO_2$ $<$1,900; $2ZnO.SiO_2$ 1,488; $2SrO.SiO_2$ 1,593; $2BaO.SiO_2$ 1,600–1,900. Data for Na_2SiO_3 with other silicates are furnished by Wallace,[3] Kultascheff,[4] Van Klooster.[5]

205. Non-silicate Slags.—These slags, as stated in §191, are composed either of a metallic oxide and an acid other than SiO_2, such as P_2O_5, SO_3, B_2O_3, Sb_2O_5, As_2O_5, TiO_2, etc., or of one or more oxides, such as PbO, CaO, Fe_2O_3, Mn_2O_3, Al_2O_3, etc.

Pure non-silicate slags are hardly ever formed in metallurgical practice. Usually the non-silicate slag is associated with a silicate slag. The general object of forming a non-silicate slag is to free a metal produced in a smelting operation from harmful elements by means of fluxes. These elements, which become oxidized in the purifying or refining process, furnish the acid, or oxide, which combines with the fluxing base. The more suitable, therefore, the working conditions for the oxidation of the impurity, for the union of the acid, or oxide, with the fluxing base, and for the mutual solubility of the non-silicate and the silicate slags, the more successful will be the purification

[1] Shelby, Fe_3O_4 in slags, *Eng. Min. J.*, 1908, LXXXVI, 1264.

[2] *Zt. anorg. Chem.*, 1907, LV, 159.

[3] *Zt. anorg. Chem.*, 1909, LXIII, 1.

[4] *Op. cit.*, 1903, XXXV, 187.

[5] *Op. cit.*, 1910, LXIX, 122, 135.

of the metal, provided that the amount of slag formed is sufficiently large in proportion to the metal to be refined.

(1) Dephosphorization and Phosphate Slags.[1]—The effects of P upon the physical properties of industrial iron are of vital importance; Fe_3P is readily soluble in iron, but practically insoluble in slags. In order to remove P from iron it has to be oxidized. This is readily accomplished at temperatures below 1,300° C.; above this temperature P_2O_5 is likely to be reduced by C, Mn and Si. Now P_2O_5 is readily soluble in basic, and practically insoluble in acid, slags. According to Grüner,[2] 30 per cent. SiO_2 is the upper limit at which a slag will take up P_2O_5. The lower the temperature of the oxidizing fusion, and the more basic the slag, the more complete will be the oxidation of P by O, CO_2, Fe_xO_y and its elimination from the metal. The addition of CaF_2 aids the removal of P, as it renders fusible a basic slag at a temperature at which it otherwise could not be melted (see p. 432). According to the prevalence of FeO or CaO as base two phosphate slags have to be considered.

In puddle cinder and in the slag formed in the Bell and Krupp pig-washing processes, in all of which FeO is the leading base, P is present as $3FeO.P_2O_5 = Fe_3P_2O_8$[3] ($Mn_3P_2O_8$); the tri-basic phosphate being held in solution by the basic iron cinder which contains 15 to 25 per cent. $SiO_2+P_2O_5$, the rest being mainly Fe_xO_y (Mn_xO_y); the P content ranges from 1 to 5 per cent.

Hilgenstock[4] showed that in the slags of the basic bessemer and basic open-hearth processes P_2O_5 was combined with CaO to $4CaO.P_2O_5 = Ca_4P_2O_9$ which forms hexagonal crystals.[5] Here the tetra-basic phosphate is dissolved in the basic calcic silicate containing $SiO_2+P_2O_5$ 12 to 30 per cent., CaO (+ MgO) 30 to 50 per cent., the rest being mainly FeO with some MnO. The P_2O_5 content of basic bessemer slags reaches 18, of basic open-hearth slags perhaps 8 per cent. Another compound, $4CaO.P_2O_5+CaO.SiO_2 = Ca_4P_2O_9+Ca.SiO_3$, forming blue monoclinic crystals, has been recognized by Stead and Risdale.[6] The latest studies are those of Matweieff, Blome, Steinweg and Van Kloaster.[7]

(2) Desulphurization by Means of Slag.[8]—Vogt has shown that the S-content of slags is due to the presence of simple sulphides (CaS, MnS, FeS,

[1] Jüptner, *J. I. and St. I.*, 1900, II, 298.
Matweieff, *Rev. Mét.*, 1910, VII, 848.
Blome, *Metallurgie*, 1910, VII, 659, 698.
Steinweg, *op. cit.*, 1912, IX, 28.
Kroll, *J. I. and St. I.*, 1911, II, 126.

[2] *Ann. Min.*, 1879, XV, 149.

[3] Finkener, Mitt. Kgl. techn. Versuchsanst., 1883, p. 284; or Wedding-Phillips-Prochaska, "The Basic Bessemer Process," Eng. Min. J., New York, 1891, p. 151.

[4] *Stahl u. Eisen*, 1883, III, 498; 1886, VI, 525.
Groddeck-Broockman, *op. cit.*, 1884, IV, 141.

[5] Bücking-Linck, *Stahl u. Eisen.*, 1887, VII, 245.
Hilgenstock, *ibid.*, p. 557.

[6] *J. I. and St. I.*, 1887, I, 222.

[7] *Zt. anorg. Chem.*, 1910, LXIX, 122.

[8] Vogt, J. H. L., "Die Silikatschmelzlösungen," Dybwad, Christiania, 1903–04, Vol. I, p. 96.
Jüptner, *J. I. and St. I.*, 1902, I, 304; also *Stahl u. Eisen*, 1902, XXII, 387, 432.

ZnS, MgS; Cu_2S is insoluble[1]), and that the sulphides separate first when the molten slag cools, proving that the solution of sulphide in molten slag is approximately saturated. Analyses of slags show that they contain at most 2.5 to 3 per cent. S, hence the solubility of S-ide in slag at oidinary temperature is small. The capacity of a slag to absorb S-ide increases with its temperature and basicity, and varies with the character of the bases. In the iion blast-furnace FeS (MnS) go into the pig iron, and CaS, MnS and FeS into the slag. As C, Si, Mn and P diminish the solubility of S-ide in pig iron, and as CaO and MnO increase its solubility in slag, the higher the content of C, Si, Mn, and P in the pig and the higher that of CaO and MnO in the slag, the more S-ide will the latter be able to dissolve. In a matting furnace, the matte dissolves less than 1 per cent. slag; the bases FeO and ZnO are favorable to the solution of S-ide in the ferrous slag. In smelting, beside fulfilling the preceding conditions, it will be necessary to make a large amount of slag if a considerable quantity of S-ide is to be taken up by it. Between the dissolving power of pig iron, or matte, and of slag in contact with it, there will be established an equilibrium, *i.e.*, the S will be distributed between the two at a constant ratio which, as has been shown, depends upon the composition of the products and the temperature. Jüptner[2] has calculated the coefficients of distribution $\frac{\text{S in slag}}{\text{S in metal product}}$ for pig irons, basic bessemer and open-hearth steels, and for mattes.

Desulphurization of pig iron by Mn in the mixer, of steel by calcareous slags and fluxes in the basic bessemer and open-hearth steel processes belong to a metallurgy of iron and steel.

(3) Other Non-silicate Slags.—Sulphates (§77) and borates[3] are formed practically only in the dry assay of ores; antimonate, arsenate with some stannate of lead are collected as intermediary products in the refining of argentiferous lead, but nothing is known about their constitutions. The formation temperatures of some silicotitanates of Ca and Al with as much as 14 per cent. TiO_2 have been investigated by Lennox and Cox.[4] They were found to lie between 1,270 and 1,350° C.; this evidence supports the conclusions arrived at by Rossi[5] in his experiments of smelting titaniferous iron ores.

206. Oxide Slags.—Until recently very little was known about these slags, and much remains to be done to clear up obscure points. Plumbates of alkali, $Pb(OK)_2$ and $Pb(ONa)_2$, of silver $Pb(OAg)_2$, have been prepared in the wet way. Whether the igneous solutions of CaO and other alkali earths in an excess of fused PbO enclose plumbates[6] remains to be seen.

[1] Wanjukoff, *Metallurgie*, 1912, IX, 1, 48.

[2] *Loc. cit.*

[3] Guertler, *Zt. anorg. Chem.*, 1904, XL, 268, 337.
Van Klooster, *op. cit.*, 1910, LXIX, 122.

[4] *Colo. College Publication*, Colorado Springs, Colo., 1906, Engineering Series, Vol. 1, p. 1.
Smolenski, Bi-Titanates and Bi-Silicates, *Zt. anorg. Chem.*, 1911, LXIII, 293.

[5] *Tr. A. I. M. E.*, 1892–93, XXI, 832.

[6] Borchers, *Metallurgie*, 1905, II, 1.
Le Chatelier, *Compt. Rend.*, 1893, CXVII, 109.

It is known that sesqui-oxide bases, R_2O_3, can enter into combination with monoxide bases. Stead and Risdale[1] recognized a compound $CaO.Al_2O_3$; Boudouard[2] found the same compound; Shepherd-Rankin-White[3] traced in part the equilibrium diagram of the CaO-Al_2O_3 series,[4] shown in Fig. 306, and found four compounds and three eutectics. Starting from the left (the CaO) side of the diagram: the compound $3CaO.Al_2O_3$ begins to be decomposed at 1,530° C., the compound $5CaO.3Al_2O_3$ melts at 1,386° C., and $CaO.Al_2O_3$ at 1,587°, while $3CaO.5Al_2O_3$ is decomposed at 1,710° C. The eutectic $3CaO.Al_2O_3-5CaO.Al_2O_3$ with 47.6 per cent. Al_2O_3 freezes at 1,385° C.; the eutectic $5CaO.Al_2O_3-CaO.Al_2O_3$ with 54.8 per cent. Al_2O_3 freezes also at 1,385° C.; the eutectic $CaO.Al_2O_3-3CaO.5Al_2O_3$ with 67 per cent. Al_2O_3 freezes at 1,575° C. The spinel $CaO.Al_2O_3$ is found in blast-furnace slags and zinc retorts[5] Duface[6] as well as Shepherd-Rankin[7] have identified $MgO.Al_2O_3$; Bourgouis[8] has done the same for $MnO.xAl_2O_3$ and $MnO.Mn_2O_3$.

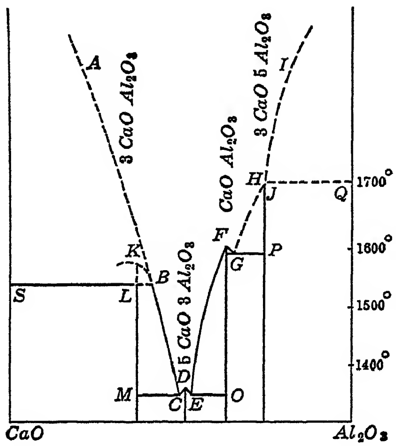


FIG. 306.—Equilibrium diagram of lime-alumina series.

Stead-Risdale[9] recognized $3CaO.Fe_2O_3$ and $CaO.Fe_3O_4$. Hofman-Mostowitsch[10] prepared $CaO.Fe_2O_3$(?) from $CaSO_4$ and Fe_2O_3 and found it a ruby-colored crystalline substance which sintered at 1,200, fused at 1,230 and was completely liquefied at 1,250° C. The series $CaO-Fe_2O_3$ has been investigated by Kohlmeyer-Hilpert.[11] The equilibrium diagram, Fig. 307, shows two hidden chemical compounds, $3CaO.1Fe_2O_3$ and $1CaO.4Fe_2O_3$; two compounds with well-defined melting-points, $3CaO.2Fe_2O_3$ and $2CaO.3Fe_2O_3$; and an eutectic $3CaO.2Fe_2O_3-2CaO.3Fe_2O_3$ which corresponds to $1CaO-1Fe_2O_3$, the mixture examined by Hofman-Mostowitsch.

A barium ferrite has been prepared by Mostowitsch.[12] Zinc ferrite is dis-

[1] *J. I. and St. I.*, 1887, I, 222.

[2] *Op. cit.*, 1905, I, 339.

[3] *Am. J. Sc.*, 1909, XXVIII, 293; *J. Ind. Eng. Chem.*, 1911, III, 211; *Stahl u. Eisen*, 1910, XXX, 1513.

[4] Schott, O., "Kalksilicate und Aluminate," Doctor dissertation, Heidelberg, 1906.

[5] Stelzner-Schulze, *Berg. Hüttenm. Z.*, 1881, XL, 145, 150.

[6] *J. Pharm. Chim.*, 1901, XIV, 25.

[7] *Loc. cit.*

[8] *Am. Chim. phys.*, 1883, XXIX, 433.

[9] *Loc. cit.*

[10] *Tr. A. I. M. E.*, 1908, XXXIX, 645.

[11] *Metallurgie*, 1910, VII, 194, 225.

[12] *Metallurgie*, 1909, VI, 450; *Eng. Min. J.*, 1909, LXXXVIII, 601.

cussed by Hofman,[1] Wells,[2] Burleigh,[3] Lindt.[4] Copper, lead and ferrous ferrites have received some attention. The review of the subject by Kohlmeyer[5] is timely on account of the importance of these compounds and their former neglect, as is the microscopical investigation of Matweieff.[6]

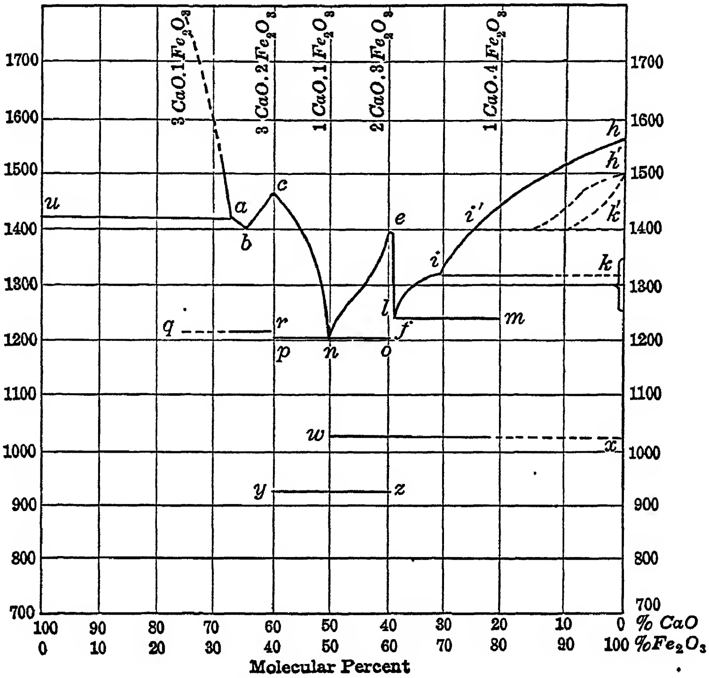


FIG. 307.—Equilibrium diagram of lime-ferric oxide series.

207. Fluidity.[7]—The degree of fluidity desired in a slag depends upon the kind of furnace in which the slag is produced, and upon the necessity of a good separation of metal, or speise and matte, from the slag. If the slag is to be poured from a crucible or tapped from a blast-furnace, it must be more fluid than when it is raked out from a copper reverberatory furnace or from a basic open-hearth steel furnace. The separation of metal (speise and matte) is not so perfect from a slag that is viscid as from one that is fluid; nevertheless, as long as the difference in specific gravity is sufficient, viscous slags made in some reverberatory-furnace smeltings often give satisfactory economic results. In order to become fluid, most slags, when formed, have to be brought to the flowing or working temperature, and this lies some 100° C. above the melt-

[1] *Tr. A. I. M. E.*, 1905, XXXV, 856.
[2] *Eng. Min. J.*, 1908, LXXXVI, 420.
[3] *Electrochem. Ind.*, 1904, II, 355.
[4] *Metallurgie*, 1909, VI, 409, 745.
[5] *Metallurgie*, 1910, VII, 289.
[6] *Rev. Mét.*, 1910, VII, 848.
[7] Greiner, E., "Viscosität in Silicatschmelzen......," Doctorate dissertation, Jena, 1907.

ing temperature of the metal or metallic compound that is to be produced in smelting. If it lay below, the slag would become liquefied before the desired chemical change in the charge to be reduced had been completed. Systematic experiments on fluidity have not been made. An apparatus for this purpose has been suggested by Johnston.[1] In the discussion of fusibility (§ 194 and following) the fluidity of the different slags has been noted. In general, acid slags are viscous, basic slags fluid as long as the basic character of the slag is not too pronounced as, *e.g.*, with basic open-hearth slags with <25 per cent. $SiO_2+P_2O_5$. A viscous slag passes through a more or less pasty stage before it solidifies; a fluid slag passes quickly from the liquid to the solid state.

208. Texture.—The texture of a slag depends upon its chemical composition and the manner of cooling. It may be vitreous, porcelanic, stony, or crystalline. Acid ferruginous slags and slags from the charcoal iron blast-furnace show a vitreous texture when cooled quickly, a porcelanic when slowly. The latter, if low in Fe and Mn, swell up to a pumice-like mass when sprayed with water. Basic slags rich in alkaline earths have a stony texture, be they cooled quickly or slowly; such slags containing about 45 per cent. CaO and little Al_2O_3 will disintegrate quickly when exposed to air on account of the CaS being oxidized by O, CO_2 and H_2O. A crystalline texture is not uncommon with slags ranging from the 1- to the 2-silicate degree; slags of a higher silicate degree rarely show any crystalline texture, and if of a lower degree, only when they are rich in FeO. Ferrous slags containing Al_2O_3 are likely to be vitreous.

209. Specific Gravity.—The specific gravity of slags varies mainly with the character of the bases. It may be as high as 5 with basic ferruginous slags; 3 is a common figure for iron blast-furnace slags; it may reach 2.5 if the CaO is replaced to some extent by MgO. With the ferro-calcic silicates of the lead smelter a common range is 3.4 to 3.6; copper smelters in treating pyritic ores often make heavier slags.

210. Color.—This varies greatly; some slags are white, others black, others again red, brown, green, blue. Slags with alkali-earth bases, such as iron blast-furnace slags, are gray;[2] the presence of a small amount of FeO gives them a greenish tinge, large amounts of FeO and FeS make them black. The ferruginous slags of the lead and copper smelter, as a rule, are a brownish-black; Cu_2O makes slags reddish-brown, MnO yellowish-brown; the combined presence of FeO and MnO yellowish-green; in the presence of MnS bluish tints have been observed in slags with little FeO and much MnO; TiO_2 is said to give a bluish color.

211. Hardness.—This varies much with the rate of cooling. Chilled vitreous slags are less hard than slags that have been cooled down slowly.

[1] *Electrochem. Met. Ind.*, 1906, IV, 262.

[2] Harpf-Langer-Fleissner, *Oest. Zt. Berg. Hüttenw.*, 1909, LVII, 709, 727, 746, 762.
Fleissner, *op. cit.*, 1910, LVIII, 75, 91, 104, 122, 141, 158, 169, 186.
Fleissner, H., "Einenhochofenschlacken, ihre Eigenschaften und ihre Verwendung," Knapp, Halle, 1912.

212. Specific Heat, Latent Heat of Fusion, Heat of Formation.—The specific heats of iron blast-furnace slags have been found by Åkerman[1] to lie between 0.29, the specific heat of the more readily fusible slags, and 0.33, the specific heat of the less readily fusible slags. The latent heats of fusion[2] have been found to range from 73 to 120, averaging perhaps 95 cal. Very few data exist for the ferruginous slags of the lead and copper smelter. Richards[3] gives $0.20185+0.0000302\,t°$ C. as the mean specific heat of a copper blast-furnace slag SiO_2 35.5, FeO 39.7, MnO 1.0, CaO 11.4, MgO 2.7, Al_2O_3 9.2, CuO 0.42, S 0.42; the actual specific heat at $t°$ C. is then $0.20185+0.0000604\,t$, which at its melting-point of 1,114° C.$=0.269$. He determined the latent heat fusion as 40 cal. Vogt[4] gives 85 cal. as the latent heat of fusion of a fayalite (Fe_2SiO_4) slag The heat of formation from constituent oxides has been given in Table 57.

213. Economic Use of Waste Slags.—Many attempts, more or less successful, have been made to utilize waste slags for industrial purposes in view of the large amount of space that has to be provided to store them, and the increasing distances over which they have to be hauled to reach the edge of the dump. Except for the manufacture of slag-wool and slag-cement from suitable iron blast-furnace slags, the main economic use of slags has been in the line of building materials. Slags from lead and copper furnaces are sometimes cast into the forms of bricks, blocks, tiles, etc.;[5] they must be acid, as otherwise they crack; further they ought to be cooled slowly (annealed) so as to lose their glassy texture and brittleness, and become strong and tough. The manufacture of cast blocks from iron blast-furnace slags has not been successful, but granulating[6] basic blast-furnace slag, drying, grinding, mixing with lime to bring the lime content to about 50 per cent., and molding by machinery into bricks is an established industry.[7] Both ferrous and non-ferrous slags of a strong or crystalline texture are utilized for macadamizing roads and ballasting

[1] *Stahl u. Eisen*, 1886, VI, 281, 387.

[2] Vogt, "Silicatschmelzlösungen," Vol. II, p. 65.

[3] *Electrochem. Met. Ind.*, 1907, V, 266.

[4] *Loc. cit.*

[5] Peters, E. D., "Modern Copper Smelting," Scientific Publ. Co., New York, 1895, p. 142.
Hofman, H. O., "Metallurgy of Lead," Scientific Publ. Co., New York, 1899, p. 362.
Egleston, *School Min. Quart.*, 1890–91, XII, 189.
Klette, *Zt. Berg. Hütten. Sal. Wesen. i. P.* 1892, XL, 501; *Berg. Hüttenm. Z.*, 1893, LII, 150.
Lang, *Eng. Min. J.*, 1907, LXXXIII, 567, 1240; Clements, *ibid.*, p. 775.
Wilder, *ibid.*, p. 1252; LXXXIV, 673 (Wagner-Primrose); 1909, LXXXVIII, 916 (Lang); 1910, LXXXIX, 501, 696 (Wethey); *Min. Sci. Press*, 1907, LXCVI, 205 (Smith), 553 (Hixon); 1908, XCVII, 224 (Ross).

[6] Air Granulation at Buderus Works, *Iron Age*, 1912, XC, 1215.

[7] Eckel, E. C., "Cements, Limes and Plasters," Wiley, New York, 1905, pp. 675-689.
Zsigmondi, *Dingler Pol. J.*, 1892, CCLXXXIV, 233.
Lürmann, *Stahl u. Eisen*, 1897, XVII, 991; *Eng. Min. J.*, 1898, LXV, 314.
Witte, *Thonind. Z.*, 1897, XXI, 468.
Meyer, *op. cit.*, 1899, XXIII, 253.
Editor, *op. cit.*, 1906, XXX, 1251.
H., *Berg. Hüttenm. Z.*, 1896, LV, 342.
Helmhacker, *Eng. Min. J.*, 1898, LXV, 460.
Waterhouse, *Eng. Mag.*, 1909, XXXVII, 361.

railroads. Ferrous slags, after crushing, are used in making cement concrete.[1] Slags from the basic bessemer process with >16 per cent. P_2O_5 are used as a fertilizer. In England non-ferrous slags have been run into glass pots charged with alkali for the manufacture of bottle glass and railroad ties. Utilization of the heat of waste slag for raising steam[2] is practiced in some instances, as well as for warming blast.[3]

214. Reducing Smelting.—The usual state of chemical combination from which metal is recovered by a reducing fusion is that of oxide or oxide salt. In some instances sulphide and arsenical ores are treated direct, *i.e.*, without being first converted into oxides.

(1) OXIDES.—The heats of formation of oxides, Table 28, furnish the key to the study of the affinities of the metals for O. A few oxides, such as those of Au, Ag, Pt, Hg, are decomposed by heat alone; the others require a reducing agent. With the exception of MgO, all metallic oxides can be deoxidized by means of solid C. The reduction of some begins at a comparatively low temperature, say 500° C.; others require 1,000° C. to complete the process, and the most refractory, as *e.g.*, Al_2O_3, give up their O only at the temperature of the electric arc (3,500° C.). Many of the metals reduced from refractory oxides by means of C combine with this and form carbides. The reduction temperatures of oxides nearly always lie below the boiling-points of the metals, Zn, Cd, and As forming the exceptions; it may lie, however, above or below the melting-point of the resulting metal.

The leading reducing agents are C and CO; H, C_xH_y and CN are of minor importance; in special instances a metal or alloy, and an element like P and S come into play. The oxidation of C begins at about 400° C., and the affinity of C for O increases with the temperature. The reaction $2MO + C \rightleftarrows M_2 + CO_2$ takes place at a low temperature, or when C acts upon a readily reducible oxide; at a high temperature or acting upon an oxide difficult of reduction the reaction is $MO + C \rightleftarrows M + CO$; the line of division between the two reactions lies at about 1,000° C. The formation of CO_2, or CO, is governed, however, not only by the temperature, but also by the relative proportion of O to be removed and of C employed; viz., an excess of O favors the formation of CO_2, an excess of C the formation of CO. As the reduction by means of solid C does not penetrate much into a metallic oxide of moderate size, it is necessary to have both oxide and C finely divided and intimately mixed, but even then the reduction is slow. The process may be quicker if the metallic oxide is fused, as then an intimate contact is more easily effected.

Carbon monoxide reduces many metallic oxides at comparatively low temperatures. Its reducing action begins at about 200° C., increases to about 1,000° C. and then decreases; hence below 1,000° C. it is a strong, above 1,000° C.

[1] Kesner-Doolittle, *Min. Rep.*, 1907, LIV, 9, 54.

[2] Lang, *Eng. Min. J.*, 1909, LXXXVIII, 916.
Göpner-Vautin, *Metallurgie*, 1910, VII, 161.
Johnson, *Met. Chem. Eng.*, 1911, X, 662.
Perkins-Barker, *Eng Min. J.*, 1912, XCIII, 267.

[3] Lang, *Eng. Min. J.*, 1909, LXXXVIII, 916.

a weak reducing agent. Being a gas it readily penetrates the oxide to be reduced and acts quickly as long as the oxide has not been sintered or fused. The reaction taking place in reduction is expressed by $M_xO_y+yCO \rightleftarrows M_x+yCO_2$, or $M_xO_y+CO \rightleftarrows M_xO_{y-1}+CO_2$. The resulting CO_2 with its oxidizing power opposes the reducing effect of CO; with enough CO_2 an equilibrium is established, the gas having become indifferent, neither reducing nor oxidizing; with still more CO_2 the gas mixture becomes oxidizing. The proportions of CO and CO_2 necessary to neutralize one another vary with the metal and the temperature. The equilibrium conditions of Fe_xO_y have been especially studied.

Another reversible reaction of CO is $2CO \rightleftarrows C+CO_2$, *i.e.*, CO conducted over rough surfaces, over finely-divided substances, such as Fe_2O_3, NiO, CaO, C, heated to temperatures ranging from 200 to 1,000° C. is more or less decomposed as shown.

The reduction of CO_2 by means of C, beginning about 400° C., has been discussed in §142.

H has a greater deoxidizing power than CO for the same temperatures. The reaction $MO+H_2 \rightleftarrows M+H_2O$ is reversible and much limited in its practical application; it may play some part in the reduction of metals like Sn, Cu, Pb which do not decompose H_2O at a red heat. As the production of H is an expensive process, the use of this reagent is confined to its incidental formation by the action of H_2O-vapor upon an element oxidized by it, such as incandescent C. C_xH_y are contained in a smoky flame and are set free with the gases issuing from the green stick used in poling Cu. Their reducing effect is a combination of those of C and H. Heavy C_mH_n, which upon heating are split into C and CH_4, have a stronger reducing power than the light C_xH_y.

CN formed by the action of N upon incandescent C plays an unimportant part in the reduction of Fe_xO_y in the iron blast-furnace. Alkali cyanides form a valuable reducing agent in the assay laboratory.

Metals or alloys which have a strong affinity for O are sometimes added in small quantities to a metal bath holding some of its oxide in solution or suspension in order to reduce this oxide. Al, which of all ordinary metals has the strongest affinity for O, is added in small quantities to molten steel, to brass and other alloy-castings; Mg to Ni; spiegeleisen and ferromanganese to bessemer and open-hearth steel; P, or Cu_xP, to metallic Cu or Cu alloys.

Sulphur or metallic S-ide, lastly, is an important reducing agent in the roasting and reaction process of smelting lead ores, in the reverberatory smelting of copper ores, and in the converting of copper matte; the S combines with the O of the oxidized Pb, or Cu, and passes off as SO_2; the reactions with lead ore take place at about 700° C., those with Cu ore and matte at about 1,100° C.

The deoxidation of metallic oxide MO, by a reducing agent R, may be endo- or exo-thermic, $MO+R=M+RO\pm$ cal. Most reducing processes absorb more heat than they generate and require external sources of heat to make up the deficit. The most striking exception to this general truth is the

aluminothermic process of Goldschmidt.[1] Here Al in a more or less finely divided state is mixed with the metallic oxide to be reduced (and perhaps some flux added to retard the reaction) and charged into a crucible lined with MgO. A fuse consisting of a mixture of finely divided Al and BaO_2 is placed on top and ignited either by a Mg ribbon serving as a fuse or more simply by a match. The heat set free by the oxidation of the Mg, viz., $Mg+O=MgO+197{,}000$ cal., brings the primer to the reaction temperature ($3BaO_2+2Al_2=3BaO+Al_2O_3+356{,}000$ cal.); this excess heat starts the reaction (taking, *e.g.*, Fe_2O_3 as oxide: $Fe_2O_3+2Al=Al_2O_3+2Fe+197{,}000$ cal.) at one point of the charge which is sufficiently exothermic to go on, not only without any external heating, but to fuse the resulting Al_2O_3 and reduced metal, the calculated temperature with Fe reaching 2,694° C.[2] The process is well suited for reducing refractory metallic oxides (MnO_2, Cr_2O_3, TiO_2, etc.) and producing a metal free from C. It has further found application in welding, soldering and annealing in places where it is not convenient or possible to use other methods of heating. Striking examples are the welding of steel rails in place and the patching of large iron and steel castings. Metallic sulphides are completely decomposed by Al.[3]

(2) SULPHIDES AND ARSENIDES.—Metallic S-ides and As-ides are commonly given an oxidizing roast to convert them into oxides before they are subjected to a reducing fusion. In some cases, however, they are smelted raw (direct) or after an imperfect roast which has left unaltered some of the original S-ide or As-ide. The reducing fusion then becomes a sulphurizing, or arsenizing operation in which the metals are collected in a matte, or a speise, in the order of their affinities for S or As. In the presence of an excess of one metal (*e.g.*, Fe) that has a stronger affinity for S or As than another (*e.g.*, Pb) and a corresponding lack of S or As, the second metal will be set free in the metallic state (precipitated). Common examples of metal acting as a reducing agent are the decomposition of PbS, Pb_xAs, Sb_2S_3 and of HgS by means of Fe. The Fournet-Schütz series (§60) and the heats of formation of metallic S-ides, Table 30, and As-ides, Table 36, give the facts regarding the affinities of metals for S and As.

[1] Hofman, *Tech. Quart.*, 1902, XV, 101, with cross-references.

Caldwell, *Iron Trade Rev.*, Oct. 8, 1903, p. 54.

Goldschmidt, *Zt. Electrochem.*, 1903, IX, 858; *Electrochem. Ind.*, 1903, I, 527, 534; *Berg. Hüttenm. Z.*, 1903, LXII, 571; 1904, LXIII, 611; *Metallurgie*, 1905, II, 33; *J. Frankl. Inst.*, 1905 CLX, 187.

Mathesius, *Stahl u. Eisen*, 1903, XXIII, 925; *Iron Age*, Sept. 10, 1903, p. 8.

Scholl, *Electrochem. Ind.*, 1903, I, 175.

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Hulbert, *Iron Age*, 1911, LXXXVII, 1538.

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[2] Richards, J. W., "Metallurgical Calculations," I, 46.

[3] Ditz, *Metallurgie*, 1907, IV, 786.

215. Oxidizing Smelting.—This process is suitable for metals, mattes, speises, and ores; the oxidizing agents are air, steam, metallic oxides and oxide salts.

Air is the common oxidizer. Steam, beside serving in some cases as a mechanical stirrer of molten metal, will be used as an oxidizing agent with metals which decompose it above a red heat, such as Mn, Fe, Co, Ni, Zn, Sb, Sn, but then only under conditions such that the heat absorbed by its decomposition may serve to some extent in regulating the temperature.

Among the metallic oxides, those of Mn, Fe, Pb and Cu are the most common; FeO silicates with <30 per cent. SiO_2 act as O-carriers in that when melted with free access of air, the excess FeO is converted into Fe_2O_3 which, acting as an oxidizing agent, *e.g.*, upon C in pig iron, is again reduced to FeO. Salts like $NaNO_3$, $HKSO_4$ serve for refining precious metals.

The general object of melting a crude metal and exposing it to the strongly oxidizing influence of heated air is to remove impurities which have a greater affinity for O than they have for the metal itself. Thus Sn, As, Sb and Zn are removed from Pb; Fe from Cu; Si, M, C and P from Fe, etc. The order of affinity of metals for O, as deduced from the heats of formation of oxides, Table 28, is sometimes changed by the temperature. Under normal conditions Si burns first in the acid bessemer process, when the air is put on, and heats the bath to the temperature at which C begins to oxidize; if the metal at the start is at the C-ignition temperature, the C will burn at once and prevent some of the Si from becoming oxidized, or reduce again some of the SiO_2 that had been formed. The relation of C and P in the basic bessemer process is similarly affected, as the C is all oxidized before the P begins to combine with O, and the P_2O_5 will be reduced again to P, if there is not present an excess of base to hold it. The removal of impurities from metal is not accomplished altogether directly by the O of the air, but to a considerable extent indirectly. The O does not simply combine with the impurities, amounting to <10 per cent. of the weight of the metal, and leave the metal intact; on the contrary, it also oxidizes the surface of the metal itself; the formed oxide, acting as a carrier, transfers its O to the impurity and oxidizes it. The more intimate the contact of metallic oxide and impure metal, the more effective will be the elimination of the impurity; thus Cu_2O or NiO soluble in Cu, or Ni, are powerful purifiers. When the oxides are not soluble, they have to be stirred into the metal. This is done by hand (puddling pig iron; fining copper), by compressed air (bessemerizing pig iron; fining copper), and by steam (refining Zn-Pb alloys). With hand stirring, it is desirable that the oxide be somewhat pasty, with the other methods it has to be as fluid as the metal. In order to prevent the oxide from being scorified and thus become practically inactive, it is necessary that an oxidizing fusion giving an acid or basic oxide be carried on in a furnace with respectively an acid or basic (neutral) hearth. The progress of an oxidizing fusion is observed by the changes in color and texture of the oxide withdrawn from the surface of the metal and by the changes in the metal itself. If the impurity that is being eliminated passes off as a gas, it may cause the metal

to boil and may burn, as does CO, with a blue flame; or it may give off a characteristic odor, as does SO_2, and project metal upward with sufficient force to divide it so finely that when falling down again it will have the appearance of a fine spray (copper-rain).

The oxidizing fusion of matte is practiced mainly with copper and nickel matte. Examples are the purification of impure copper matte to free it from Pb and Fe, the roast-smelting and converting of rich copper matte in order to produce metallic Cu, and the concentrating and converting of copper-nickel matte. The oxidation in these processes is effected more directly by the action of air than indirectly as is the case with metals. Where the quick and cheap process of converting is not advisable on account of the presence of Pb and Zn, which cause losses in precious metal, the slow oxidizing fusion has been mostly replaced by the quicker operation of roasting part of the matte, mixing it with raw matte, and melting the charge.

Speise is concentrated by an oxidizing fusion in which arsenides having much affinity for O, notably Fe, are oxidized, scorified, and drawn from the fluid bath. Here also the oxidation is effected in the main directly by the O of the air.

The treatment of ore by an oxidizing fusion is confined for the present to the process called pyritic smelting in which the heat set free by the quick oxidation of S and Fe of pyritic ore through the O of the blast is utilized to combine the FeO, as soon as formed, with the siliceous gangue and slag it.

216. Melting.—This may be termed a neutral fusion, as the intention is to avoid reduction and oxidation. The object of melting is, either to liquefy a metal or alloy and cast it into some special form, or to prepare alloys; special precautions are taken to protect the metals from contact with air by covering the charge with charcoal or some flux.

217. Smelting Apparatus in General.—A smelting furnace demands a temperature sufficiently high to liquefy both ore and flux; this temperature, with fuel furnaces, requires the combustion of a certain amount of fuel in unit time and space, and the fuel needs a certain volume of air. The four classes of furnaces given in §167 furnish the important types. Hearth and blast-furnaces are run with compressed air or blast; reverberatory and closed-vessel furnaces with natural or forced draft, and converters again with blast. Electric furnaces are taken up in pp. 380, 389–394.

218. Hearth Furnaces Worked with Blast (§167).—A small pile of ore mixed with charcoal and served by bellows[1] represents the primitive fusion; surrounding the pile with stones or making a hole in the ground to hold the charge together and enable a better utilization of heat is the next step; and this gives the earliest form of hearth. The furnace was used quite extensively in the production of wrought iron (Catalan forge, American bloomary), but has become about obsolete, at least in the United States, in the iron industry on account of the rigid requirements as to ore and fuel, the small capacity, the imperfect yield, the character of the product, and the expense of fuel and labor. Impure copper was once fined and even brought to the tough-pitch stage in a

[1] Percy, J., "Iron and Steel," Murray, London, 1864, p. 264.

hearth furnace, furnishing the so-called rosette copper.[1] This type of furnace has not only held its own in the smelting of rich non-argentiferous galena ores of suitable size, but has furnished the basis for one of the cheapest and most satisfactory methods of extracting quickly part of the lead as long as provision is made to save the fumes. The lead-ore hearth shares the general disadvantage that oxidation and reduction go on simultaneously and thus interfere with one another, but as long as there is no objection to making a rich residue to be worked up in a blast-furnace, it fulfills most requirements.

219. Blast-furnaces (§167).—Blast-furnaces were developed out of hearth furnaces by increasing the height of the walls. While primitive blast-furnaces were used by the ancients in smelting copper ores (Discorides), in the iron industry the first pig iron was produced about the 14th century (Beck) with the application of water-power to the working of the blowing apparatus, after the height of the furnace had been sufficiently raised to carburize the reduced iron.

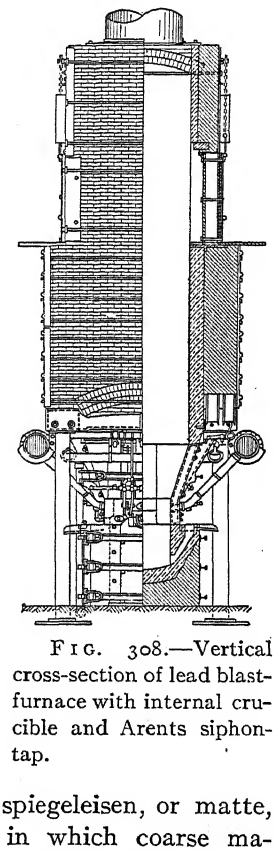


FIG. 308.—Vertical cross-section of lead blast-furnace with internal crucible and Arents siphon-tap.

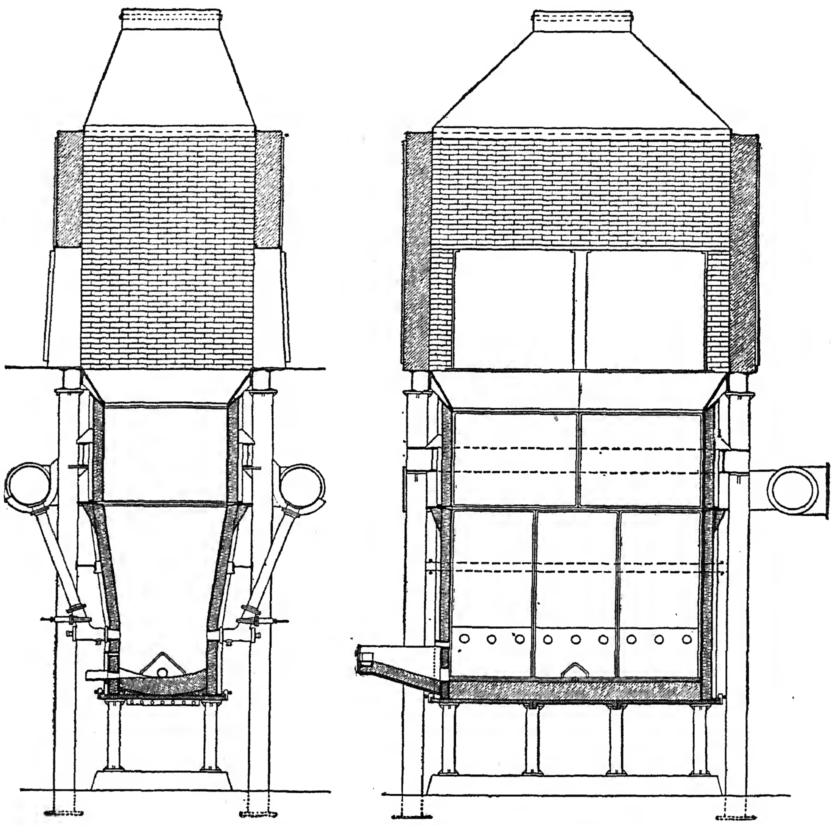
A blast-furnace (Figs. 247 and 308) consists essentially of two parts, the shaft and the crucible. The shaft reaches from the throat, where the charges are fed and the gases withdrawn, to the tuyère level, where the air is forced in to burn the fuel; and the crucible extends from the tuyère level to the bottom and holds the melted material; the crucible with its enclosing brickwork is called the hearth.

The general *shape* of the shaft must be such as to favor an even descent of charge and a uniform ascent of gas current.

The *vertical section* of the shaft (profile, lines) may show sides that are either parallel, or taper toward the bottom, or taper both toward top and bottom and form a bosh, Fig. 247; the downward taper may be uniform or may be suddenly increased so as to secure a contracted tuyère zone. Parallel sides are found at present only in cupola furnaces used for re-melting pig iron, spiegeleisen, or matte, *i.e.*, low-temperature furnaces (1,000 to 1,100° C.) in which coarse material is fused without undergoing any important chemical changes. Furnaces with a uniform downward taper are used with smelting operations demanding a medium-high temperature (1,200° C.) and with it a low-pressure blast, as is the case in small circular blast-furnaces treating readily reducible lead and copper ores. The reasons for making the tuyère area smaller than the throat area are that the original volume of the charge is reduced by the combustion of the fuel and the fusion of the deoxidized ore and the flux; further, the gradual enlargement of the furnace section from tuyères to throat retards the

[1] Percy, J., "Metallurgy, Fuel, Fire-clays. Copper, Zinc, etc." Murray, London, 1861, p. 409.

ascending gas current sufficiently to heat the descending charge and to act chemically upon it. With large-scale operations requiring a high-pressure blast the furnace is built with a bosh, Figs. 247 and 309. The contracted tuyère section secures a rapid combustion and with it a concentrated and intensified heat resulting in a quick fusion. By suddenly enlarging the horizontal section through a bosh the zone of fusion is narrowed and the velocity of the ascending gases quickly checked; the slowly moving current then gives up its heat to the charge above, acts chemically upon it and drops fine particles of charge or



FIGS. 309 and 310.—Vertical sections of copper-matting blast-furnace.

volatilized metal. The angle of bosh, *i.e.*, the space included between the slope of the bosh and a horizontal, is flatter with lead furnaces, Fig. 308, requiring a stronger reduction and greater retardation of gas current than with copper furnaces, Figs. 309 and 310, treating S-ide ores, in which it is usually desired to have the gases rush upward and oxidize some of the S. High-temperature furnaces (1,500° C.), *e.g.*, in smelting iron ores, Fig. 247, always have a steep bosh of considerable height. The height, however, is purposely limited. If the inverted cone of the circular iron blast-furnace were carried to the throat,

it would interfere with the regular descent of the charges because with a greater friction at the sides (between stationary wall and moving charge) than in the center (*i.e.*, in the loose charge) the central part would descend more quickly than that near the sides. In order to decrease the friction at the walls and to permit the closing of the throat by means of a bell and the withdrawal of the gases, the shaft is narrowed from the bosh-line to the throat. The diameter of throat and top of bosh have to be rightly proportioned, as they influence the even descent of the charge.[1] Cylindrical furnaces have proved a failure in iron smelting.[2]

The *horizontal section* has been made square, polygonal, circular, oblong and elliptical. Circular and oblong furnaces have replaced almost wholly other forms. When a square or polygonal furnace is put out of blast, the inside shows a circular section, as in smelting the corners very soon fill up. In regard to quality of work, the circular furnace gives the most satisfactory results, as there is an even distribution of blast and heat, and as the loss in heat by radiation is small because a circle offers the smallest circumference for the largest area. The quantity of the work, however, which is governed by the diameter is limited by the pressure of the blast. The iron blast-furnace permitting a high pressure is therefore always circular, Fig. 247, the diameter at tuyères reaching 15 ft. 4 in. with a pressure of 15 lb. per sq. in.; lead, copper and other non-ferrous furnaces, Figs. 308–310, which do not allow the use of a high pressure on account of the excessive reduction of Fe_xO_y to Fe instead of to FeO and the volatilization of metal, would be limited in capacity if they were made circular; they are, therefore, made oblong. The distance between the tuyères, spaced along the sides, is so chosen as to allow a moderate blast pressure, while the length is adapted to the desired capacity. Thus in lead blast-furnaces the distance between tuyères reaches 45 in. with a blast pressure of 3 lb., and in copper blast-furnaces 54 in. with a pressure of 2 lb. and less; the lower pressure of the copper furnace is due to the smaller height and the greater coarseness of the charge.

In a few instances copper blast-furnaces have been made elliptical, *i.e.*, when the shaft is constructed of a single boiler-iron water-jacket (Herreshoff furnace).

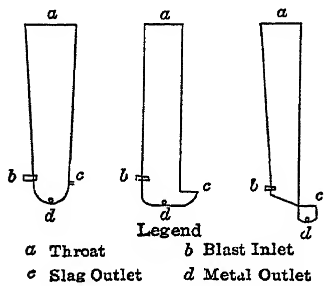
In some instances it has been noticed in lead and copper smelting that the slags from circular furnaces showed lower values than from oblong. A probable explanation of this phenomenon is that in a circular furnace the matte, dropping in the furnace, is blown toward the center and the slag drawn at the periphery, while in an oblong furnace, tapped at the ends, some matte is blown toward the tap-hole.

With regard to the *disposition of the crucible*, blast-furnaces are classed as having an internal crucible, an external crucible, a crucible partly internal, partly external. The internal crucible, Figs. 247 and 311, is situated underneath the shaft and forms the continuation of it; metal (speise, matte) and slag are collected wholly inside the furnace and tapped intermittently from different levels.

[1] Richards-Lodge, *Tr. A. I. M. E.*, 1887–88, XVI, 154.

[2] Taylor, *op. cit.*, 1884–85, XIII, 489.

The Arents siphon-tap, Fig. 308, which removes lead continuously, forms the only exception. This disposition has the advantage that there is a good utilization of heat and with it a good separation of metal and slag; the disadvantage is the inaccessibility which makes it difficult to remove accretions when they form on the bottom. It should be used when the formation of accretions is not to be expected as in the smelting of lead ores, some oxide copper ores and iron ores. The external crucible, Fig. 313, which may be fixed or movable, is placed outside the furnace wall. The charge as soon as melted flows over the inclined bottom of the shaft and collects in the external basin (settler) where the separation of metal and slag occurs; the metal is tapped intermittently (the Orford fore-hearth forming an exception) and the slag overflows continuously. With all large furnaces the blast is trapped by some means, Fig. 310; in small ones with a low-pressure blast (tin-smelting) this is not the case. The advantage of the arrangement is that the formation of accretions is reduced to a minimum, as the melted mass does not stay in the furnace, and as no half-melted charge drops on to the bottom to fill the spaces made vacant by intermittent tapping. The drawback is imperfect utilization of heat and consequent imperfect separation of metal and slag. With the large capacities of modern furnaces and with the use of covered fore-hearths, both losses have been very much reduced.



FIGS. 311 to 313.—Blast-furnace with internal, internal-external, and external crucible.

The external crucible will be used when the formation of accretions is to be expected. It has been generally adopted in smelting S-ide copper ores, and in some instances when matte is concentrated in the blast-furnace. In iron-foundry practice an external crucible is used in the Krigar cupola. With a furnace having a crucible partly internal and partly external, Fig. 312, the internal crucible is so to say continued outside the furnace-wall and ends in a fixed fore-hearth of small size; the metal is tapped intermittently, the slag overflows continuously. It permits the introduction of tools to remove accretions forming on the bottom, or at least to raise them into the hotter zone above where they may be dissolved by the metal or the slag. This form, used extensively in former days, has become about obsolete in its original form. In the modified form of Mathewson[1] it does excellent work in a lead blast-furnace as long as the ore does not contain too much zinc, and the furnace puts through not less than 50 tons in 24 hr. Here the lead is removed continuously by the Arents siphon-tap; the matte is tapped intermittently and the slag overflows in a steady stream. Excepting the slight change in level by the occasional tapping of matte, the melted material is kept constantly at the same height, and this has a favorable effect upon the regularity of the smelting as a whole; the separation of matte and slag is satisfactory, so that the re-smelting of foul slag is reduced to a low figure.

[1] Hofman, "Metallurgy of Lead," 1899, p. 237.

The *throat* of a blast-furnace is either open, Figs. 308–310, or closed, Fig. 247. It is said to be open when the furnace gases are allowed to come into contact with air before they enter the dust chamber and chimney. This will be the case when they have no fuel value, *i.e.*, when the percentage of CO_2 is either higher than of CO, as is usual with lead ($3CO_2 : 1CO$), copper and other non-ferrous smelting processes, or only slightly lower, *e.g.*,[1] CO 19.61, CO_2 17.36, H 2.45, CH_4 0.14, N 60.44. The throat is closed when the percentage of CO in the gases greatly exceeds that of CO_2 as is the case with the iron blast-furnace: CO 24, CO_2 12, H 2, CH_4 2, N 60. The air is kept excluded while the gases pass off through pipes to the fireplaces where their calorific value is to be utilized. The withdrawal of the gases, which may take place above or beneath the surface of the charge, is closely connected with the manner of feeding the charges.

The *working height* of a blast-furnace, *i.e.*, the vertical distance from tuyère level to top of charge, ranges from 6 ft. with furnaces smelting roasted pyrite to 100 ft. with the iron blast-furnace. The greater the height, the more effective the heat absorption, the stronger the chemical action of the ascending gas current and the larger the capacity, as the width has to grow with the height.

The height is limited by the composition of the ore, which may be exposed for too long a time to the chemical effect of the gases; by the coarseness and friability of ore, flux and fuel; by the unmixing of the charge during its descent; by the pressure of the blast, and by the danger of forming blow-holes which is contrary to the requirement that the gases be evenly diffused and ascend uniformly through the charge. If in treating lead, copper or other non-ferrous ores, the reducing effect of the shaft is too great, the Fe_2O_3 is deoxidized to Fe and forms a hearth accretion (sow), instead of to FeO which is to combine with SiO_2 and form slag. With fine ore the height of the charge has to be smaller than with coarse ore; coke has a greater compressive strength and is less friable than charcoal.

The *smelting power* (capacity) depends upon the cubic content of the furnace. With the iron blast-furnace the smelting power is expressed in terms of cubic content of furnace; with other furnaces in terms of square foot of hearth area. The power does not increase directly as the cubic content, but in a constantly diminishing ratio. The consumption of fuel decreases as the capacity of the furnace increases, but a similarly diminishing ratio occurs between the two, and a point is soon reached beyond which there is little or no advantage in increasing the internal space. With increase of horizontal section there has to be a corresponding increase in height on account of the requirement of a stronger blast pressure. A wide furnace has decidedly the disadvantage that the descent of the charges and the ascent of the gases are irregular, as the charge in the center traveling downward ahead of that at the side, pushes the lighter components, especially the fuel, against the wall, while the gases passing up along the wall through the fuel, do not prepare sufficiently the central ore column for the smelting zone.

[1] Kerl, B., "Grundriss der Allg. Hüttenkunde," Felix, Leipsic, 1879, p. 147.

The *construction* of a blast-furnace has to conform to the general principles given in §169. Formerly the blast-furnace was a heavy mass of masonry[1] with arched passages for access to the lower part; at present, Figs. 247 and 308, the construction is lighter; the upper part of the shaft, the stack, is supported by hollow cast-iron columns and thus made independent of the lower part, the bosh, which rests upon the walls of the crucible. This gives free access to the most vulnerable parts. In some recent constructions the superstructure of the stack (feeding-platform, etc.), which used to rest upon the top of the shaft, is carried independently by structural iron and thus relieves the throat from unnecessary pressure. The stack is usually built of brick; in some instances it is constructed of soft-steel water-jackets (Figs. 309 and 310). In a brick stack (Figs. 247 and 308) there are distinguished the inner wall of fire-brick, and the outer wall of red brick; between the two is left an expansion space, Fig. 247, filled with loose material, such as sand and ground brick. The outer wall of a circular furnace is either enclosed by an iron shell, Fig. 247, or it is braced by heavy iron bands or rods, Fig. 308; oblong furnaces usually are not sheathed; they have cast-iron corner-pieces, Fig. 308, through which pass tie-rods. An iron furnace-shell makes repairs in the stack difficult; it is, however, used with almost all iron blast-furnaces on account of the stability it gives to the structure. The boshes, which have to stand mechanical wear and resist chemical action and heat, are water-cooled. In some iron blast-furnaces the walls are sheathed and water is sprayed on the sheathing (spray-jackets) or made to flow over them (flow-jackets); small (1-in.) wrought-iron pipes with water circulating through them have been placed between the sheathing and brickwork. In most modern furnaces bosh-plates (water-cooled iron, copper or bronze boxes), Fig. 247, have replaced the older contrivances, as they maintain the lines of the furnace which the older devices do not. In practically all lead furnaces, Fig. 308, water-jackets enclose the bosh and tuyère region; with oblong furnaces they are usually cast-iron; sometimes, however, of soft steel. Cast-iron jackets are cheap, last with fair water for about three years if well made, and have value as scrap if the furnace is situated near a foundry; the lighter steel jackets are preferred if the furnace is situated at a distance sufficiently great from an industrial center for the smaller freight charge to make up for the greater first cost. Small circular lead furnaces usually have steel jackets. Copper blast-furnaces treating S-ide ore generally are water-jacketed throughout, Figs. 309 and 310, and this makes steel jackets almost a necessity.

The reason for having several, ordinarily two, tiers of water-jackets is that to carry out the usual practice of burning off S during the descent of the charge, a large volume of blast is required which causes the heat to creep up; this would make the charge adhere to the brickwork and form excessive wall accretions. The advantages of water-jackets are that the furnace is easily built, blown in and blown out, that wall accretions are readily removed, that any slag suited to the ore can be made independent of the furnace walls, that the number of tuyères

[1] Overman, F., "A Treatise on Metallurgy," Appleton, New York, 1877, p. 505 and following.

can be made large and thereby the furnace driven at a greater capacity.[1] The disadvantages are the need and cost of water that may be neither too hard nor too muddy, and the loss of heat. Supposing[2] a circular water-jacket furnace, 4 ft. in diam. at tuyères and 10 ft. high, with a 2-in water-space to consume 7,000 liters water per hour, the water to enter at 20 and leave at 70° C. The difference of 50° C. equals 50 Cal. per liter or kg. and corresponds to a loss in 24 hr. of $7{,}000 \times 50 \times 24 = 8{,}400{,}000$ Cal. With coke of a calorific power of 7,500 Cal., this loss equals 1,120 kg. coke. If the furnace puts through 60 tons of charge with 13 per cent. = 15,600 kg. coke, the percentage of total coke consumption required for heating the water-jacket is 7.2.[3] The usual range is 4–16 per cent. That the advantages outweigh the disadvantages is proved by the fact that there is hardly a blast-furnace in which the boshes and the tuyère-section are not water-cooled. When first used, the water-jackets were frequently lined with brickwork[4] which was to protect them in blowing in. Soon, however, it was found that this precaution was unnecessary, as jackets become quickly coated with a thick layer of sintered material as soon as fusion begins; in fact, the large water supply needed in starting, reaching two to three times the regular amount, must be reduced to below normal when smelting begins in order to melt off some of the very thick incrustation that has formed.[5]

The *crucible walls* are usually of brick. In lead furnaces, Fig. 308, the inner wall is of fire-brick, the outer of red brick; in copper and copper-nickel furnaces, Figs. 309–310, the corrosive action of the matte has been in many cases the cause of building the inner wall of chrome brick; in the iron blast-furnace, Fig. 274, the highest grade of fire-brick is used almost exclusively; occasionally carbon brick have been substituted. The crucible wall is always enclosed either by a cast-iron or a steel casing; in the iron blast-furnace the casing is always water-cooled. The bottom-wall of the crucible is usually built on an iron plate resting upon the foundation. In some instances I-beams have been placed between the plate and the foundation in order to furnish an air-space which prevents the bottom from becoming too hot and makes it possible to detect leakages. Water-cooling[6] has been suggested.

220. The Blast.—The blast is forced into the furnace through horizontal tuyères, Figs. 247, 308 and 311, *i.e.*, pipes which enter the furnace through an embrasure in the brickwork, the tuyère-arch. The opening in the brick wall is almost universally water-cooled by a closed annular casting of iron, brass or

[1] Glenn, *Eng. Min. J.*, 1883, XXXVI, 274.
Canby, *Tr. A. I. M. E.*, 1912, XLIV, 736

[2] See also Howe, *Bull.* No. 26, U. S. Geol. Surv., 1885, p. 89.

[3] L. S. Austin (*Min. Sc. Press*, 1908, XCVII, 525) gives an example showing 10.7 per cent.

[4] Hofman, *Min. Ind.*, 1905, XIV, 410.

[5] Cooling towers: Henrich, *Tr. A. I. M. E.*, 1895, XXV, 43, 460; Patteson, *Cass. Mag.*, 1909, XXXV, 701; Schmitt, *S. Afr. Assoc. Eng.*, 1906–07, XII, 251; Roberts, *op. cit.*, 1909, XV, 33; Yawger, *Min. Sc.*, 1911, LXIV, 480, 495; Spray Engineering Co., Boston, Mass.

[6] Osann, *Stahl u. Eisen*, 1907, XXVII, 1814.
Bosse, Buck, Lürmann, Osann *op. cit.*, 1908, XXVIII, 200, 370, 413.

bronze, or the whole tuyère-arch by a coil of iron pipe around which iron has been cast.

Open spray-tuyères, *i.e.*, tuyère-nozzles cooled by a spray of water[1] instead of by a current, are hardly used at present. With water-jackets this extra cooling is unnecessary as long as the tuyère-pipes terminate with the inside of the wall; if they reach, however, into the furnace in order to project the heat farther toward the center and thus to protect the furnace walls, an additional water-cooled tuyère-block will be imperative, Fig. 247. In smelting furnaces there is usually one row of tuyères; two rows have been used in some instances (Freiberg and Pertulosa[2] lead blast-furnaces) with the idea of burning the C more completely to CO_2, but this arrangement seems to have little to recommend it, since the height of the smelting zone is increased, while good smelting aims to reduce it and to confine the hottest zone within narrow limits.

Cupolas, which only liquefy metal, frequently have two rows of tuyères. The tuyère level depends upon the desired depth of crucible, and this will have to be smaller with a product of high than of low melting-point. If the crucible is too deep, there may be excessive cooling, if too shallow, some oxidation. The number of tuyères and their dimensions are determined by the hearth area. Within limits the distribution of the blast is more even with a large than with a small number. The penetration power of the blast depends upon the velocity with which the air enters the furnace, *i.e.*, upon its pressure. For a given volume of air, the smaller the diameter of a tuyère, the greater the velocity, but if the diameter is too small, the jet will be turned aside by the solid material in the furnace. One tuyère is made to serve 1.5 to 3.5 sq. ft. of hearth area with lead and copper furnaces, and 10 to 18 with iron furnaces run with coke; the diameter with lead furnaces is 2.5 to 4 in., with copper furnaces 3 to 5 in., with iron furnaces 5 to 7 in. The tuyère ratio, *i.e.*, the ratio of square inches tuyère area: 1 sq. ft. hearth area, ranges from 3 to 5 with lead and copper, and from 1.75 to 3.14 with iron furnaces. The blast pressure with coke as fuel reaches 3 lb. with lead, 2 with copper, and 15 with iron blast-furnaces. The tendency in the last 20 years has been toward high pressures on account of the quicker and more even combustion of the coke which results in a higher temperature, a concentration of the smelting zone, a larger smelting power and a cooler throat on account of the rapid descent of the charges. The compressed air from the blast-main enters the sheet-iron bustle pipe (Figs. 247, 308, 309, 310), which surrounds the furnace and is lined with fire-brick (Fig. 247) if the blast is heated. Usually the bustle pipe is suspended or supported in the region of the heads of the pillars carrying the stack, and thimbles on the lower side serve to form the connection with the tuyère pipes.

In water-jacketed lead, Fig. 308, or copper or other non-ferrous furnaces, Figs. 308 and 310, working with air of atmospheric temperature, the nipple and pipe were formerly joined by means of a canvas bag soaked with alum; in recent years on account of the higher blast-pressure, sheet-iron pipes, provided with

[1] Wedding, *Tr. A. I. M. E.*, 1890–91, XIX, 369.

[2] Kochinke, *Freib. Jahrb.*, 1899, p. 133.

gates to permit closing off the air, have been substituted. The tuyère pipe, at first, was an elbow of galvanized iron with peep-hole; later a machined brass nozzle was attached to the pipe in order to obtain a better joint and thus reduce air-leakage; in most modern furnaces a cast-iron tuyère box, Figs. 308 and 309, with blast inlet and outlet, peep-hole and perhaps a slag escape-valve, is screwed to the water-jacket and connected with the thimble of the bustle pipe by a sheet-iron pipe. The tuyère stock of the iron blast-furnace, Fig. 247, is of cast iron on account of the preheated blast in common use. The tuyère pipe here also is L-shaped. It is essential that the three parts, the horizontal and vertical arms and the elbow, shall permit of being easily taken apart; hence they are detached and held together by key-bolts. The water-cooled tuyère nozzle reaching into the furnace for several inches is usually bronze.

221. The Charge.—The charge of a blast-furnace is made up of ore, flux and fuel. Ore and flux should not be fine, as fine material makes the furnace work slowly and irregularly, and the blast causes a large portion of the finest particles to be carried off by the gases as flue dust. The most desirable proportion of coarse and medium-size material varies with the size of the furnace and the character of mineral that is being treated. The fuel commonly used is coke; in special cases charcoal, anthracite and non-coking bituminous coal are employed; gaseous fuel has been tried. As regards inflammability, charcoal, coke and anthracite, stand in the order given. Assuming each of these fuels to be at a red heat and to be blown with bellows: the charcoal will burn readily, the coke with some difficulty, and the anthracite will be blown cold.

The coke for an iron blast-furnace (*i.e.*, a furnace that is high, has a crucible of large diameter and is worked with a high-pressure blast), must be denser and stronger than that best suited for the shorter and narrower copper or lead furnace. Charcoal, which is weaker and more porous than coke, will be better adapted for a small furnace that is not high and is run with a low-pressure blast; thus in smelting tin ores it is the common fuel. In copper and lead furnaces, charcoal makes dirty slags on account of the fines that it forms, and the fuel consumption is higher than with coke, which is due to crumbling and to the fact that the C produces more CO than is the case with coke; gases from charcoal furnaces also run higher in CO than do those using coke. In the iron furnace, charcoal is used for making special iron; the fuel consumption is usually higher than with coke, but there are on record cases where it is lower. This must be due to the physical condition of the fuels, as in either case the C burns to CO before the tuyères.

Anthracite, which is practically impervious to gases, is difficult to ignite and hence not well suited for a large tonnage; further, its property of decrepitating in the furnace gives rise to dirt troubles, *i.e.*, troubles arising from the infusible and non-combustible mixture of fine coal and ashes frequently formed.

Lean, non-coking coal behaves in a manner similar to anthracite. The distillation of the vol. H-C. in the upper furnace keeps this cool by the heat absorbed in the process, but the loss of heat means a correspondingly large con-

sumption of fuel by the smelting process as a whole. The experiment of Blake[1] of substituting natural gas for part of the coke has local interest.

The fuel consumption of the blast-furnace is governed not only by the process and the character of the fuel, but by the altitude,[2] and increases with the latter. The explanation[3] of this practical fact is found in the law of mass action according to which, as applied to the blast-furnace, the velocity of combustion is directly proportional to the concentration of the fuel and the air. The concentration of the solid fuel is constant, while that of the air decreases with the altitude, being directly proportional to the pressure. As the temperature attained is proportional to the rate of combustion, and as the lower pressure of a high elevation will be accompanied by a lower rate, there will result a lower temperature which has to be corrected by an additional amount of fuel. The amount of fuel theoretically required to carry on a smelting operation can be calculated from the thermal balance sheet. The actual fuel consumption is greater and is a matter of experience. It is expressed in different ways. The iron blast-furnace man makes the product the unit and speaks of requiring a certain weight of fuel (± 1 ton) per ton of pig iron; the copper smelter and foundry man make the fuel the unit; the former uses the ratio of weight of charge smelted (12 : 1) and the latter the ratio of weight of pig iron melted (6 : 1) per unit weight of fuel; the lead smelter expresses his fuel consumption in percentage of weight of charge (13 per cent. fuel).

Calculation of Charge.—In a blast-furnace, the SiO_2 and bases of ores and fluxes have to be balanced in such a manner as to furnish a slag of the right composition. Beside composition, the right amount of slag necessary to protect or cover the metal product has to be considered. As composition and quantity are matters of practical experience, precedent serves as a general guide. The usual calculations embody one of the following three modes of procedure in which the slag-forming constituents are apportioned to obtain:

(1) A slag of a definite silicate degree; a method not uncommon in smelting S-ide copper ores in which there is much freedom of choice in the percentage of SiO_2 and the character of the bases.

(2) A slag with fixed proportions of SiO_2, Fe(Mn)O, Ca(Mg.Ba)O; the socalled typical slag of the lead smelter; here there is less leeway on account of the volatility of lead and the difficulty of preventing its scorification.

(3) A slag in which the sum of the percentages of $SiO_2+Al_2O_3$ stands in certain ratios to Ca(Mg)O depending upon the desired quality of the product; a method frequently followed by iron smelters based upon the assumption that Al_2O_3 always plays the part of an acid. In order to avoid calculations which to many are tedious, graphical methods have been substituted. These are in place in works which treat ores the general character of which varies little. The

[1] *Tr. A. I. M. E.*, 1886–87, xv, 661.

[2] Lloyd, *Tr. Met. Inst. Min. Met.*, 1909–10, lxxxi, 1, 11.

[3] Palmer, *Eng. Min. J.*, 1906, lxxxii, 134.

leading methods are those of Balling,[1] Jenkins,[2] Wingham,[3] Richards,[4] Hersam,[5] Brown,[6] Marshall[7] and Mathesius.[8]

222. The Process.—Passing over for the present the oxidizing fusion in pyritic smelting, blast-furnace smelting is in the main a reducing process. In its discussion it is convenient to treat separately the ascending gas current and the descending ore charge. The composition of the gases at the tuyères will vary with the temperature and the amount of fuel present. In the ferrous or iron blast-furnace, requiring much fuel per unit charge, the temperature of 1,400 to 1,500° C. will cause the C to burn to CO, hence the gas-current at the start will consist of approximately[9] N 64, CO 34, H 2 per cent. vol. As the current rises, it cools and, acting chemically upon the charge, will change in composition; CO_2 will appear and increase in amount; further, the heat of the gases will decompose carbonates, hydrocarbons, hydrates and drive off hygroscopic water, so that the gas arriving at the throat with a temperature of 200 to 300° C. has been changed to N 60, CO 24, CO_2 12, H_2 2, CH_4 2 after moisture and dust have been removed In a non-ferrous (lead or copper) furnace there is little fuel per unit charge, the temperature at the level of the tuyères is about 1,000° C., the C burns to CO_2 and CO, so that the current at the start will have a composition of say N 79, CO_2 10, CO 10, H 1. In the furnace the deoxidizing effect of the gas as it ascends cannot be so powerful as that in the iron blast-furnace, as the reducing power of CO is partly neutralized by the oxidizing power of CO_2. The percentage of CO_2 in the gases increases as they rise in the furnace, but less so than in iron smelting. The gases decompose carbonates, hydrocarbons, hydrates, and drive off hygroscopic water; they leave the furnace with a temperature $<200°$ C. and show, after having been freed from water and dust, a composition of about N 73, CO_2 20, CO 5, H 1, CH_4 1, and contain perhaps some SO_2.

The gases from some copper furnaces, run with an exceptionally large volume of air to burn off some S of the charge, will be richer in CO_2 and contain considerable amounts of SO_2; they will leave the throat of the furnace at a temperature higher than 200° C.; the top of the charge in fact is often at a red heat.

In the ferrous blast-furnace the aim is to reduce the whole of Fe_2O_3 to Fe; the slag formed is a silicate of CaO and Al_2O_3; in the non-ferrous furnace the aim is to reduce the bulk of the Fe_2O_3 to FeO that it may combine with SiO_2 and

[1] "Metallurgische Chemie," Strauss, Bonn, 1882, p. 99; translated in Roberts-Austen, W. C., Harbord, F. W., "An Introduction to the Study of Metallurgy," Lippincott, Philadelphia, 1910, p. 299.

[2] *J. I. and St. I.*, 1891, I, 15.

[3] *Op. cit.*, 1892, I, 233.

[4] "Notes on Iron," Mass. Institute of Technology, Boston, 1895, p. 44.

[5] *Tr. A. I. M. E.*, 1901, XXXI, 340.

[6] *J. Can. Min. Inst.*, 1907, X, 280.

[7] *Min. Mag.*, 1909, I, 219.

[8] *Stahl u. Eisen*, 1908, XXVIII, 1121.

[9] Van Vloten, *Stahl u. Eisen*, 1893, XIII, 28.

enter the slag (a mixture of silicates of FeO and CaO), and reduce the rest only to Fe which is to combine with S or As to form matte, or speise.

In the descending ore charge one can distinguish four zones: preparatory heating, reduction, fusion and combustion; the four combined form a single zone of heat interception. In the first, the charge is freed from hygroscopic and chemically combined with H_2O, and the fuel from its vol. H-C; the whole is then brought to a dark-red so as to be prepared for the zone of reduction. In the upper part of this zone CO[1] will act upon all metallic oxides and C only upon those that are readily reduced; in the lower part the reducing power of CO will be weaker, that of C stronger. In the zone of fusion, reactions begun in the second zone will be completed and reactions dependent upon fusion carried through. In the zone of combustion, the blast impinging upon the remaining coke, which fills the furnace more or less some distance above the tuyères and is incandescent, burns it quickly and furnishes the heat and atmosphere necessary for the process. In the zone of fusion,[2] parts of the slag-forming constituents which have a tendency to form eutectic mixtures will fuse, others will only sinter; lower down the melted portion will dissolve that which is sintered and this, with increase of temperature, will offer less resistance to becoming liquefied, until some distance above the tuyères a slag of approximately uniform composition will have been formed. This trickles through the bed of coke, becomes thoroughly fluid and collects in the crucible, where irregularities in composition are equalized. The metals and metallic compounds reduced by CO and C and perhaps some other agent such as S, become liquid, pass downward, undergo further chemical changes in the descent, may be in part volatilized, reach the slag level, affect the slag more or less in passing through it and being protected from oxidation collect in the internal crucible. Metal, speise, matte and slag separate in the order given; they may act upon one another at the contact planes, and are finally withdrawn at two or more levels.

223. General Smelting Operations. Blast-furnaces.—In the management of blast-furnaces there are certain operations which, although they may vary in detail, are common to all: warming, blowing-in, feeding, tapping, blowing-out, correcting irregularities, including banking. Only the general features are discussed here.

WARMING.—The brickwork, especially that which is exposed to a high temperature, must be slowly warmed in order to drive off the moisture gradually, as otherwise the joints will open. The fuel is burned either in a temporary fire-place[3] or in the crucible. In the latter case it is essential that the ashes be removed at short intervals, as they are poor conductors of heat. The excessive draft of the shaft is checked by partly closing the throat or by admitting false air. Warming requires several days with small furnaces, and several weeks with large furnaces.

[1] Reduction by H: Glaser, *Zt. anorg. Chem.*, 1903, xxxvi, 1. Reduction of oxides by CO, H, NH_3, CH_4: Fay-Seeker-Lane-Fergusion, *Brooklyn Polyt. Eng.*, 1910, x, 72; *Eng. Min. J.*, 1911, xci, 406.

[2] See also C. Doelter, "Physikalisch-chemische Mineralogie," Barth, Leipsic, 1905, p. 127.

[3] Sweetser, *Tr. A. I. M. E.*, 1912, xliv, 105.

Blowing-in.[1]—A furnace is usually filled to the top before the blast is started that the ascending gases may warm the charges; the engine is run at first as slowly as possible in order to produce a small volume of gas which, rising slowly, gives up its heat almost entirely to the cold charges. The principle followed in filling is to start with excess fuel and easy smelting material and work toward normal fuel and ore charge. Thus, beginning with kindling, this is followed by dry cord wood, charcoal and coke. The fuels should be accompanied by the fluxes necessary to slag the coke ashes As the most readily fusible material is slag, the first charge or charges should consist entirely of slag; the slag is gradually replaced by ore and flux until the normal charge has been reached. The ratio of fuel and charge is at first above normal and slowly decreases to normal. Fig. 314 illustrates the principle. The heated crucible is filled to above the tuyères with kindling (wood, charcoal); this is followed by a heavy bed of coke with the necessary flux; above this, the vertical and inclined lines indicate three divisions standing for normal coke with its flux, slag, and ore with its flux. The horizontal lines represent the separate charges: line *ab* shows that the normal quantity of coke has to melt only slag, hence it is in excess of what would ordinarily be required; at *cd* some of the slag has been replaced by ore and flux; at *fg* the blowing-in charges have been stopped, and the regular ore charge, normal coke with given weight of ore and flux, have been reached. The kindling, after having been ignited, may be allowed to burn some time with natural draft, but usually the blast is turned on at once. The temperature of the melted material withdrawn from the furnace at the beginning is below normal, but rises gradually as the smelting progresses.

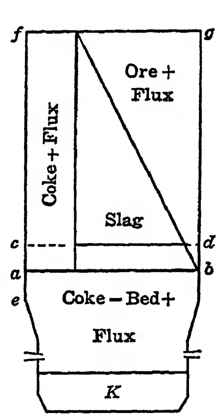


Fig. 314.—Blowing-in of blast-furnace.

Feeding.—In the old small square or oblong brick furnaces which had no water-cooling and tuyères only at the back, the fuel was fed toward the front and the ore charge toward the back, forming a pair of vertical columns. The main object of charging in this manner was to protect the brick walls, as the smelting took place more in the center than along the sides. The method is wasteful in fuel and furnishes an imperfect reduction, as the gases ascend mainly in the fuel column. It has been replaced by feeding fuel and ore charge alternately in horizontal layers, fuel going in first with the non-ferrous, and last with the iron blast-furnaces. The quantity fed at a time increases with the area of the throat; the permissible minimum is governed by the amount of fuel which is necessary to form a distinct layer upon the preceding ore charge. In order to ascertain this, a wooden frame, about 6 in. high, shaped to the form of the throat is filled with coke; its weight will represent the smallest permissible amount.[2] There is an advantage in having large charges,[3] as the gases can pass

[1] Sweetser, *loc. cit.*, for iron blast-furnaces.

[2] Hahn, *Tr. Inst. Min. Met.*, 1899–1900, VIII, 267.

[3] Dwight, *Tr. A. I. M. E.*, 1902, XXXII, 363.

readily through the coke, while this is not probable with the ore; with a thick fuel bed they therefore can be distributed over the whole furnace area whenever they have been forced through the denser ore charge below.

Special attention has to be paid to the distribution of the coarse and fine parts of a charge. As the gases have a tendency to travel upward along the sides of a furnace, feeding fine parts along the side and coarse parts toward the center will cause them to ascend more evenly. Furnaces are fed by hand or mechanically. Shoveling in the weighed charge from the side is more favorable to correct distribution of coarse and fine than feeding by hand from the top, as in throwing a shovelful of material into the furnace, the coarse will travel farther than the fine. As the charges descend more quickly in the center than at the sides, mechanical feeding has to be so contrived that a new charge shall strike the preceding one at the side of the furnace in order that the coarser parts may roll down the inclined surface and collect toward the center leaving the fines at the sides.

Furnaces with open tops, such as lead and copper furnaces, were formerly fed exclusively by shoveling; in recent years the charges of large furnaces are dumped from charge-buggies either from the sides or from the top. In the iron blast-furnace with its closed top, the usual method of dumping the contents of the charge-buggies by hand has been replaced in many instances by mechanical means, which does away with all manual labor (top filler) on the feed floor.

Tapping.—The manner of withdrawing the smelted charge from the furnace depends upon the disposition of the crucible. In copper, lead and other non-ferrous furnaces there is danger of the slag carrying along with it particles of matte. The slag therefore is made to flow through settling-pots before it either goes in waste-slag cars to the dump or is granulated. In the iron blast-furnace there is no necessity for such devices; the slag (cinder) goes straight into cinder cars. The disposition of metal, speise and matte varies too much to be considered here.

Blowing-out.—A furnace is blown out on account of lack of charging material, or of accidents which cannot be remedied in a given time, or for business reasons. Under normal conditions the length of a campaign varies from a week to several years, depending upon the character of the charge and the furnace. The common method of blowing-out is to give some slag charges to replace the last ore charge, to allow the charges to sink and diminish the blast gradually, to close at the same time the flue or pipe which usually carries off the furnace gases, and to allow these to pass off into the open. When the charge has sunk to a certain distance above the tuyères, which varies with the size of the furnace, the blast is stopped, the tuyères are withdrawn and the openings closed; the liquid contents are tapped, the front of the furnace is broken into and the solid part raked out.

Correcting Irregularities.—A furnace is in normal working order when it gives the required tonnage, when (1) at the feed floor the charges descend regularly, the temperature of the throat is right, the gases pass off uniformly and have the correct composition, and (2) on the furnace floor, the cooling water, the blast

pressure, and the appearance of the tuyères show little variation and the products are even as regards quality and quantity. If a furnace working normally has to be stopped for a time, *i.e.*, to be banked, it is fed with blowing-in charges and run until these appear before the tuyères; now the liquid material is tapped, the tuyères are withdrawn and their openings closed, and the stream of cooling water is diminished. Iron blast-furnaces are thus held for months, other furnaces for much shorter periods. If the furnace has to be stopped suddenly on account of floods, strikes, etc., the liquid material is tapped and the furnace closed up. In this condition a furnace may be held only about one-quarter the time that is permissible when the necessary preparation was possible. The correction of disturbances in the regular working caused by a chill, by hanging and irregular descent of charges, faulty fluxing, etc., are made to suit individual cases; the remedies hardly involve principles that admit of a general discussion.

224. Reverberatory Smelting Furnaces in General (§167).—These furnaces are of more recent origin than blast-furnaces, having come into existence in the 17th century (Barba, 1640). They are best classified according to the fuel used as being fired with ordinary solid fuel, fuel dust, liquid and gaseous fuel; a secondary point of view is the mobility of the hearth which usually is stationary, but in some instances has been made tilting (Campbell, Wellman), oscillating (Roe), and even revolving (Danks).

In furnaces fired with solid fuel the highest temperature attainable is about 1,500° C. For higher temperatures preheated gaseous fuel and preheated air become a necessity; natural gas, which cannot be preheated without being decomposed, enters the furnace at ordinary temperature, but furnishes with preheated air the required temperature on account of its high calorific power.

The first consideration with a reverberatory furnace is the capacity of the hearth required for doing economic work; the second is the necessity of obtaining the temperature demanded for smelting. The latter is governed by the calorific power of the fuel, by the correct proportioning to the hearth of the apparatus used for burning the fuel, and by the appliances for furnishing the needed air and for withdrawing the products of combustion.

225. Reverberatory Smelting Furnaces Fired (Direct) with Solid Fuel (Fig. 248).—The leading parts are the fire-box, the fire-bridge, the hearth and the flue leading to the stack. The form (§151) of the *grate in the fire-box* depends upon the kind of fuel that is to be burnt. It is worked with natural or forced draft. With the present tendency to increase the size of the hearth forced draft has in many cases replaced natural draft. Forced draft, however, has the great disadvantage that the intense local action of the heat causes the ashes of the coal to form large massive clinkers, the removal of which during "grating" takes considerable time and correspondingly diminishes the smelting power or tonnage. The heat generated on a grate, supposing the calorific power of the fuel and the draft to be constant, depends upon the amount of fuel that is burned per hour on a square foot of grate area. Some of the data of Grüner have been given on p. 326. Modern practice with large furnaces has changed some of these figures; *e.g.*, in matting S-ide copper ores, the usual figure given is 25 lb.

coal per hr. per sq. ft. grate area; Montana reverberatories[1] burn 42 to 54 lb. or nearly twice the amount. The reason for this difference is that the old small furnaces were built with a view of low fuel consumption; the large modern furnaces, on the contrary, aim to burn fuel as quickly as possible to obtain high smelting power, and thus to utilize the heat as effectively and economically as conditions warrant. The fire-box is lined with fire-brick, rich in Al_2O_3. This will resist the corrosive action of the clinkers better than the one rich in SiO_2.

The *fire-bridge* separates the grate from the hearth. The distance from the grate bars to the top of the fire-bridge, *i.e.*, the depth of the grate, is always large in smelting furnaces (say $3 \pm$ ft.) as compared with roasting furnaces, because the action of the flame has to be reducing rather than oxidizing, and because depth furnishes room for a bed of clinkers which warm the incoming air and prevent fine fuel from falling through the spaces between the grates. The distance from top of the fire-bridge to the hearth depends upon the desired action of the flame; it is higher in roasting, when the flame is to travel along the roof, than in smelting, when it is to be kept close to the charge and heat by radiation as well as by more or less contact. The fire-bridge has to resist the heat of the flame, the corrosive action of the clinkers and the charge, and the longitudinal thrust of the hearth. The material used is therefore a high-grade fire-brick; further, the bridge is usually made hollow that it may be cooled either by a current of air or by a stream of water flowing through a jacket. In small furnaces the longitudinal thrust of the hearth need not be especially considered; in medium-size furnaces it is taken up by a heavy iron plate or by a cast-iron box-girder or by I-beams. The stress may be thus transferred either through the walls of the fire-box to its buckstays and tie-rods, or, better, taken up directly by buckstays and tie-rods of the main body of the furnace. In large furnaces the last method is used exclusively; this makes the body of the furnace independent of the fire-box as it should be. The area of flue above the fire-bridge is about half that of the grate area.

The *hearth* which extends from the fire-bridge to the chimney flue has various shapes and sizes. In small and medium-sized furnaces it is usually oval with the broader end at the bridge; in large furnaces the main body is oblong; the ends are somewhat contracted toward the bridge and very much so toward the flue. The flame passing over the fire-bridge expands upon entering the hearth that it may have free development; in order that its high temperature may reach the flue end, the hearth has to be narrowed and the roof at the same time lowered. The width is limited by the necessity of all parts of the hearth being accessible with tools introduced through the working doors, and by the difficulty of supporting a flat arch; the length by the indispensability of attaining the required smelting temperature at the flue end.[2]

The *hearth area* must stand in a certain ratio to the grate area for a given

[1] Peters, E. D., "Principles of Copper Smelting," Hill Publishing Co., New York, 1907, p. 189.

[2] The largest reverberatory smelting furnace is probably that of Anaconda for smelting S-ide Cu ores, 19×116 ft.

temperature. Strictly speaking, it is the ratio of volume of laboratory to quantity of fuel burnt, or the number of calories generated, per hour. As the height of the arch above the hearth shows little variation, the simpler ratio is sufficient for all purposes. Grüner,[1] gives the following values for hearth areas, assuming the grate area to be 1: steel-melting furnace 0.8 to 1.2; puddling furnace 2.0 to 2.5; reheating furnace (muck-bar packets) 2.0 to 2.5; air furnace (foundry) 2.5 to 3.0; copper-refining furnace 4.0; copper matting furnace 5.0 to 6.0; black copper smelting furnace 4.5; lead smelting furnace (roasting and re-action) 8.0 to 10.0; tin smelting furnace 6.0 to 7.0. The ratios given in 1875 have been changed in many instances by improved practice; striking examples are found in the matting of copper ores and in the refining of copper.

The hearth rests upon a foundation built up solidly when it is essential to keep it as hot as possible, as in matting S-ide copper ores; when this is not the case, air vaults are erected underneath, and the air passing through them is conducted underneath the grate bars (older constructions of copper matting furnaces) or into the laboratory of the furnace to increase the oxidizing effect of the flame, as in the fining of copper; in cases where it is essential that the hearth be kept cool, as in the open-hearth steel furnace, it is erected oniron plates supported by an elevated steel structure. Water cooling of the bottom has been tried, but was found to abstract too much heat; water cooling of the sides and ends is not uncommon with lead softening, refining, and cupelling furnaces to counteract the corrosive action of PbO and of some of its compounds. The material of the working bottom is chosen to suit the corrosive action of the charge. Thus copper and acid open-hearth furnaces have a silica bottom that is burnt in, the air furnace in iron foundry work has a rammed sand bottom, the basic open-hearth steel furnace a bottom of calcined dolomite that used to be rammed in, but now is burnt in; the lead cupelling furnaces have a hearth rammed with marl, cement or bone ash, the lead reverberatory smelting furnaces a bottom of fused gray slag or puddle cinder; the lead softening or refining furnace a hearth of fire-brick rich in Al_2O_3 or of bauxite brick, etc. Bottoms burnt in have been termed monolithic.

A single stationary *roof* usually covers both the fire-place and the hearth; its highest point is above the fire-bridge; it then slopes more or less gradually to the lowest point, which is above the flue. The arch is cylindrical; flat arches appearing cylindrical are formed by the combination of the arcs of several circles. The main axis of the roof usually runs parallel with the sides, rarely parallel with the ends of a furnace; the former arrangement gives a stronger structure, as the sides carrying the skew-backs form a better support than the ends; the latter, however, are of a more uniform temperature. The rise of the arch is 0.5 to 1.0 in. per foot of span. In some instances the roof is made movable as in crucible-steel, malleableizing, ingot-soaking furnaces; it is then built in sections with the bricks held together by heavy iron clamps, or laid in iron frames. The roof, usually 9 in. thick, is either of fire-brick or silica brick; the bricks form a single course and are laid parallel with the sides of the furnace.

[1] Traité de Métallurgie, 1, 269.

The inner side walls are of high-grade fire-brick or magnesite brick, the outer walls of an inferior quality of fire-brick and sometimes of red brick.

The *flue* is separated from the hearth by the flue bridge. The section of the flue is small, having an area from 1/6 to 1/4, and more recently to 1/2 that of the grate area. This narrowing of the flue tends to check the draft; it has, however, the advantage that the gases traveling quickly keep it hot for a long distance and thus reduce the loss of heat by radiation from the hearth. Making the flue area smaller than the chimney area seems a doubtful practice; it is more economical to leave space for a damper to be used when occasion arrives. The heat of the gases issuing from the flue is utilized for raising steam, drying materials, heating air, etc., before the gases are allowed to enter the chimney. Chimney-draft is taken up in §311.

IRONING.—A furnace has to resist the pressure of a heavy roof, and the expansion of the bottom and of the brickwork in general when it is heated. The thrust acting both sideways and endways is taken up by buckstays (rails or I-beams,[1] steel trusses set vertically against the sides and ends of the furnace) and tie-rods which connect tops and bottoms of opposite pairs of buckstays (Fig. 248). In order to prevent overheating and thus diminish the danger of bending, buckstays have been backed with asbestos board. In modern furnaces with a bottom built up solidly, the lower tie-rod has been given up, as becoming heated it fails to serve its purpose;[2] the lower end of the buckstay is imbedded in heavy stone or slag block. The polished cross-section of a broken tie-rod of an Anaconda matting reverberatory furnace[3] showed only a few large crystals of ferrite instead of the usual fracture, because of the prolonged annealing.[4] The tie-rods pass either through holes in the buckstays when they are threaded at the ends and tightened with nuts, or they have loops which enclose the ends of the buckstays; the rods then are provided with turnbuckles. Encasing a furnace with soft-steel or cast-iron plates distributes the pressure more evenly and permits using a smaller number of buckstays with stronger tie-rods than is otherwise permissible, but an iron casing radiates more heat than does brickwork. In most instances only the weakest parts are fortified by cast-iron plates, iron rails, I-beams imbedded horizontally in the brickwork backing against the buckstays. In building, the masonry is placed against the iron work and not *vice versa*. In warming up, care must be taken to loosen the tie-rods as the masonry expands; in cooling down, to tighten them again.

226. Reverberatory Smelting Furnaces Fired with Fuel Dust, Oil or Gas.— These have no separate fireplaces, but the fuel with the necessary air enters the laboratory. The discussion of the laboratory and hearth of furnaces fired with solid fuel applies to this class with the exception that the ends of the oblong hearth often are not contracted as is the case when ordinary solid fuel is used.

The principles and the leading apparatus for burning these fuels have been

[1] Granbery, *Eng. Min. J.*, 1905, LXXX, 487.

[2] Hofman, *Tr. A. I. M. E.*, 1904, XXXIV, 298.

[3] Sample examined, 1907.

[4] Campbell, *Metallurgie*, 1907, IV, 808.

reviewed in §107 (pulverized fuel), §153 (liquid fuel), and §155 (gaseous fuel). Fuel dust, which is used commonly in burning Portland cement mixtures, has been applied only in recent years in reverberatory smelting furnaces for matting copper ore and puddling pig iron. Liquid fuel has been used for a number of years in the Western oil fields and the country contiguous to them. A peculiar form of reverberatory furnace for fusing metals and alloys, to which only brief allusion can be made, is that designed by Schwartz[1]: an onion-shaped steel chamber lined with fire-brick and mounted on hollow trunnions through which pass the oil-burners. The charge is introduced through a manhole at the top, closed during a fusion, and the metal poured near the bottom through a spout which serves also as flue for the escape of the products of combustion. Gaseous fuel, especially producer gas, is the leading fuel of the steel industry. As the secondary air is always preheated, furnaces fired with gaseous fuel will be taken up in connection with preheating (§321) fuel and air.

227. The Charge and Process.—Reverberatory smelting furnaces serve to treat ores, metals, alloys and metallic compounds. The ores are usually finely divided, while metals, alloys and metallic compounds are charged in pieces of varying sizes and even in the liquid state. The atmosphere in the furnace is on the whole more oxidizing than reducing; the reverberatory furnace, therefore, will be used only for the reduction of oxides which readily give up their O and the resulting metals of which have low melting-points, as *e.g.*, with PbO, SnO_2,Sb_xO_y.

The management differs so much with the metals that are to be recovered that little can be said that has a general bearing. The warming and heating up has to be gradual; the tie-rods have to be loosened as the brickwork expands, the charge may be introduced from the side or the top, intermittently or continuously, by hand or mechanically. The liquid products may be removed wholly or only in part; some are drawn off continuously, others at given intervals of time; the gases passing off through the flue carrying off dust and vapor may be madd to pass through dust chambers and condensing apparatus, and their heat may be utilized for industrial purposes.

228. Closed-vessel Smelting Furnaces (§167).—The form of the closed-vessel furnace is governed (1) mainly by the shape of the vessel which may be a crucible, a tube, a muffle, a kettle, (2) by the number of vessels, and (3) by the manner of heating.

229. Crucible Furnaces Heated with Solid Fuel.[2]—Heating metal in a cru-

[1] *Cass. Mag.*, 1903–04, XXV, 72; *Iron Trade Rev.*, Oct. 31, 1901, XXXIV, p. 19; Oct. 13, 1904, XXXVII, p. 38; Dec. 20, 1906, XXXIX, p. 24.

[2] Schmatolla, E., "Tiegelöfen," Seydel, Berlin, 1901.

Buchetti, J., "La Fonderie de Cuivre," Béranger, Paris, 1905.

Wedding, *Verh. Ver. Beförd. Gewerbefl.*, 1898, LXXVII, 190.

Wüst, *Stahl u. Eisen*, 1903, XXIII, 1139, 1229 (Schmatolla)

Irresberger, *op. cit.*, 1904, XXIV, 169, 253.

Ledebur, A., "Lehrbuch der Mechanisch-metallurgischen Technologie," Vieweg, Brunswick, 1905, pp. 181–196.

Krom, *Met. Ind.*, 1909, VII, 287, 324. 358, 404, 436; 1910, VIII, 80; English Melting Furnaces, *Met. Ind.*, 1910, VIII, 294.

cible imbedded in charcoal with the use of blast reaches back into pre-histrric times.[1] About the middle of the 18th century natural draft began to replace forced draft, and held its own until about 1880 when forced draft began to be used again more generally. The solid fuel ordinarily used is coke; occasionally anthracite takes its place and rarely charcoal. The best size for the coke is 2 1/4 to 3 1/4 in., and the space between crucible and wall for successful work is 3 1/2 in. Charcoal being lighter than coke occupies a larger volume for a given weight of fuel and requires more space; about double the weight of that of coke is required, as charcoal burns more to CO than does coke. The amount of fuel required for melting depends also upon the size of the charge. The subjoined formula[2] furnishes a means of calculating the amount of fuel that is necessary: $Q=\frac{M[L+S_m(T-t)]k}{C.P.\times e}$, in which Q=weight of fuel, kg.; M= weight of metal, kg.; L=latent heat of fusion, kg.-cal.; S_m=mean specific heat; T=highest temperature desired, ° C.; t=starting temperature, ° C.; k=constant depending upon the size of the charge components; $C.P.$=the calorific power of the fuel in kg. cal.; e=the efficiency of the furnace (3 to 5 per cent. without and 8 to 15 per cent. with blast).

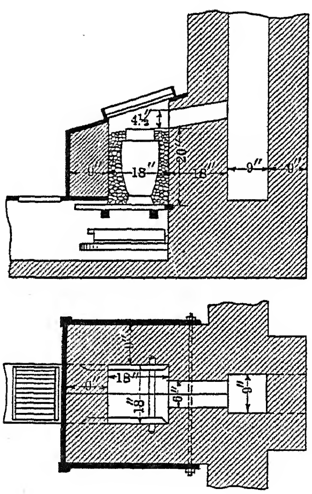


Fig. 315.—Ordinary crucible-furnace for solid fuel and natural draft.

The ordinary form of crucible[3] furnace[4] with natural draft is shown in Fig. 315.[5] It is built with a depressed ash pit in order to give the top of the furnace a convenient working height. The fire-brick walls are enclosed by a sheet-iron casing; sometimes an air space is left between the two to reduce the loss of heat; the grate bars are wrought iron; the inclined cover, an iron frame filled with a tile, is usually hinged and balanced by a weight and chain or wire rope passing over a pulley attached to the wall or the ceiling. Furnaces, however, are made preferably circular, as the crucible is more evenly heated and there is a saving of fuel. In square furnaces the parts of the crucible opposite the corners become hotter than those facing the sides, and the fuel in the corners is in part wasted; the square form requires no special brick and permits poking down the fuel in the corners with less danger of injuring the crucible. In order to keep the bottom of the crucible hot, it stands on a fire-brick support 3 or 4 in. thick and of a diameter slightly smaller than that of the crucible. In the

[1] Beck, "Die Geschichte des Eisens," I, 75.

[2] Stahl, E., "Metallgiesserei," Craz and Gerlach, Freiberg, 1906, p. 85.

[3] Clay-lined Crucible (*Brass World*, 1906, II, 231), Bottom-poured Crucible (*op. cit.*, 1906, II, 231); Annealing Crucible (*Am. Mf.*, 1904, LXXV, 702; Care of Crucible, *Brass World*, 1909, V, 307.

[4] Rockwell Furnace, *Met. Ind.*, 1909, VII, 324.

[5] See also Fig. 249.

illustration the crucible is seen to carry a refractory ring[1] in order to permit raising the fuel column to the lower edge of the flue to prevent the gases from rushing into the flue, to protect the rim of the crucible, and to prevent scrap from falling into the fire. Table 161 gives some of the leading facts about this form of furnace.

TABLE 161.—DIMENSIONS OF CRUCIBLE FURNACES

Capacity of crucible, pounds metal	Height, top of grate to bottom of flue, inches	Grate, length by width, inches	Flue area	Stack area	Stack height
60–80	19	15×15	$\frac{1}{4}$ to $\frac{1}{8}$	$\frac{1}{6}$ to $\frac{1}{3}$	30 ft. or more de-
100	21	17×17	grate	grate	pending upon
200	23	19×19	area	area	draft desired

The leading disadvantage of this form of furnace is that the crucible has to be taken from the fire after every fusion, which makes necessary small charges and causes injury to the crucible by the tongs[2] and by sudden cooling. This was overcome by Piat in 1877[3] who, as shown in Fig. 316, strengthened the sheet-iron casing and suspended the furnace by means of trunnions *z*, having square ends, in a cast-iron frame *l*. When a fusion is finished, the furnace is removed with a holder[4] to the mold, tilted and the metal cast. The crucible *t* is kept in place by means of a tile *m* which has the necessary openings for feeding coke, the exit of the gases and the discharge *s* of the metal. In addition to the advantages given above, the new device permits the use of large charges, which does away with the inconvenience of having several crucibles in one furnace, saves fuel and increases the life of the crucible.[5]

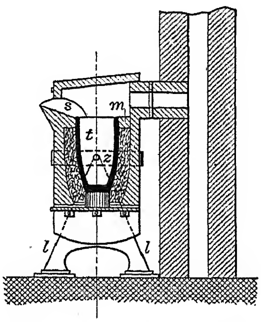


FIG. 316.—Piat crucible furnace.

Medium-weight charges are handled with a differential pulley suspended from a traveling crane. For heavy charges Baumann[6] devised two apparatus, the second and improved form of which is shown in Fig. 317. The furnace *K*, pivoted on the bar *H*, rests with its trunnions *G* on two supporting rods *J*, which

[1] See *Brass World*, 1909, v, 291.
[2] Horne, Crucible-tongs, *Foundry*, 1908–09, XXXIII, 69.
[3] Steffen, *Stahl u. Eisen*, 1890, x, 189.
[4] Duffy, *Brass World*, 1910, VI, 307.
[5] See also Ideal Tilting Crucible Furnace, *Brass World*, 1908, IV, 337.
[6] Irresberger, *loc. cit.*
Buchetti, *op. cit.*, Plate IX; *Stahl u. Eisen*, 1895, XV, 1063.
Eisenbach, *Stahl u. Eisen*, 1898, XVIII, 547.

are connected by pin *F* to end of lever *C* with fulcrum at *B*, while the other, a toothed sector, engages with pinion *A* rotated through worm gear *D* and rod by means of handwheel *E*. Beneath shaft *H* is the ladle to receive the molten metal. One man has full control of the casting.

Large furnaces make it necessary to work with forced draft in order to obtain the air necessary for quick work. The difference between natural and forced draft is brought out by the following data of Irresberger.[1] An ordinary furnace with natural draft will melt 200 lb. bronze with 50 to 60 per cent. coke in 1.5 hr., the crucible lasting 25 charges; a Piat-Baumann furnace will melt 350 lb. in 12 to 18 min. with 18 per cent. coke, the crucible lasting 40 charges,

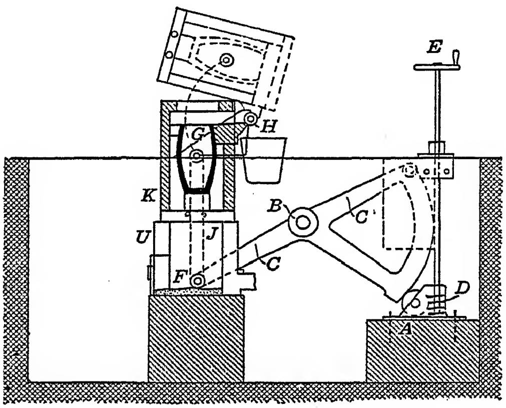


FIG. 317.—Baumann crucible-furnace tilting device.

and 60 charges when protected by a cover; the loss in either case is about 0.3 per cent. The powerful effect of the blast causes the coke ash to slag which obstructs the passage of the air. Two methods are used for correcting this evil: one is to make easy the removal of the clinkers (the grating), the other to introduce the blast from the side. The grate of the Piat-Baumann furnace is shown in Fig. 318. It consists of two parts, the central casting which carries the crucible support and is forced against the carrier bars *D* by means of wedges, and the ordinary loose wrought-iron grate bars on either side of the casting. Cleaning is thus made easy. If a crucible has to be removed, the carrier bars are taken out.

The side-blown furnace of Rousseau with Basse-Selve improvements[2] is represented in Fig. 319. The blast enters an air box surrounding the crucible wall and then the furnace through four rows of tangential openings. The furnace has three openings for feeding coke, and a central funnel for warming the charge (see below). The furnace melts a 600-lb. charge of copper or brass in 40 min. and requires 10 per cent. coke.

[1] *Loc. cit.*

[2] Wedding, *loc. cit.*; also *Met. Ind.*, 1903, I, 111; 1905, III, 179; Reyelbec Furnace, *Met. Ind.*, 1909, VII, 109.

Two methods are in operation for the utilization of the heat of the waste gases: preheating the air by means of a recuperator,[1] and heating the charge either by running two furnaces in tandem,[2] the gases from the melting furnace passing through the warming furnace on their way to the stack, or by placing a funnel-shaped preheater over the crucible and having the gases ascend through it.[3] The last method, started by Piat, has found much favor.

Two forms of Piat-Baumann furnaces are given in Figs. 320 and 321. Fig. 320 represents the type[4] suited for brass or metal of similar melting-point. The

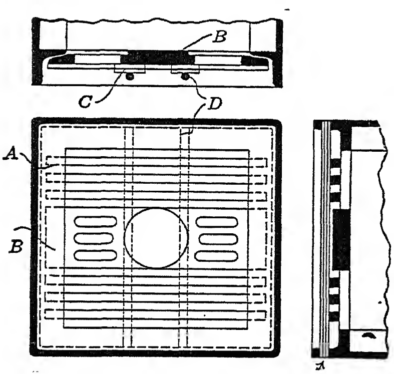


FIG. 318.—Grate of Piat-Baumann crucible-furnace.

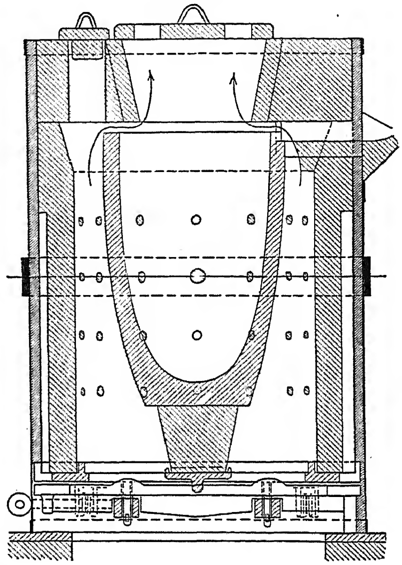


FIG. 319.—Side-blown crucible-furnace of Rousseau with Basse-Selve Improvements.

furnace is covered with a ring of fire-brick which carries the conical preheater, also of firebrick, with cone of crucible material. The pouring spout is closed with a clay plug to force the gases to travel in the direction of the arrows. The mode of operating is: fill the furnace with coke, bring the crucible to a red heat (with a 300-lb. charge and a 7-in. blast pipe the blast-pressure is 0.75 in.), replenish the coke, one-quarter of which has been consumed, charge into the crucible large pieces of metal, put in place the preheater, add the rest of the charge, raise the blast-pressure to 4.75 in. when after 3 or 4 min. the metal in the preheater begins to fuse; now lower the pressure to 0.5 in. to reduce loss and in 10 or 12 min. the fusion is complete. The coke consumption is 15 per cent. The furnace in Fig. 321 is suited for bronze or metal with high melting-point.

[1] Schmatolla, *Stahl u. Eisen*, 1899, XIX, 1158; 1900, XX, 1136.

[2] *Brass World*, 1907, III, 415.

[3] Horner, *Foundry*, 1913, XLI, 113.

[4] Wise, Coke Tilting Furnace, *Met. Ind.*, 1909, VII, 436.

M. V. R., Brass-melting Tilting Crucible Furnace, *op. cit.*, p. 108; Rockwell Furnace, *op. cit.*, 1910, VIII, 80.

The main difference between the two furnaces is that in the furnace of Fig. 321 the cone of crucible material consists of two parts separated by a 2-in. space, of which the lower part is contracted to an opening 3 1/8 in. in diam.; on top of the furnace is placed the mold resting on a pair of bars. The cast-iron support of the furnace shows the blast pipe, the wind box and the ash pit. The last has conical walls; the bottom, rammed with sand, carries an iron plate with U-shaped handle to facilitate removal of metal, in case the crucible leaks, of ashes and of clinkers. The mode of operating is: bring the crucible to a red

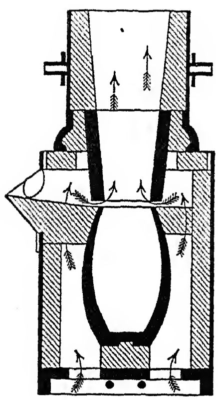


Fig. 320.—Piat-Baumann crucible-furnace with preheating of charge.

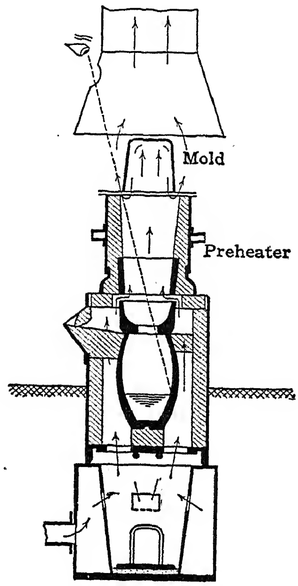


Fig. 321.—Piat-Baumann crucible-furnace with preheating of charge and warming of mold.

heat with a pressure of 0.75-in. water and replenish the coke as above; charge 250 g. pulverized glass, put in place the preheaters, charge first coarser then finer metal, the metal of high melting-point (Cu) coming last in order; start the full blast and diminish gradually; when fused, remove the preheaters and add the necessary Sn, Zn, Cu_xP, heat 5 min. more and pour. The coke consumption is 18 per cent.

The mode of operating the Rousseau-Basse-Selve furnace, represented in Fig. 319, differs from that just given in that the metal is not fused in the preheater; that the crucible is charged with the bulk of the metal, the fine parts only going into the preheater, and that the coke is replenished during a melt through openings in the cover.

Fuel dust is not well suited for heating crucibles on account of the slagging effect of the ashes.

TABLE 162.—PREPARATION OF BRONZE

With 80 lb. copper, time of fusion, 12 min.		With 160 lb. copper, time of fusion, 16 min.		With 240 lb. copper, time of fusion, 18 min.	
Blast					
Time, minutes	Pressure, in. water	Time, minutes	Pressure, in. water	Time, minutes	Pressure, in. water
1–4	8	1–5	8.75	1–6	9.5
5–6	6	6–8	6.75	7–9	7.25
7–8	5	9–12	5.25	10–13	5.5
9–12	3.5	13–16	3.5	14–18	3.5

230. Crucible Furnaces Heated with Oil.[1]—They are used in districts in which residuum can be obtained at a price sufficiently low to compete with coke, as liquid fuel has many advantages over solid fuel[2] (§154). Both the vaporizing and atomizing methods (§153) of burning oil are in operation. The former has been shown in Figs. 209 and 210. Fig. 322 represents an oil-fired crucible furnace with a single burner; the flame is made to impinge upon the crucible support. In some furnaces there are two burners set tangentially in order that the flame may whirl around the crucible and heat it uniformly; in others, as in the Denver[3] and San Francisco[4] mints, the oil, atomized by air with a pressure of 11 to 16 oz. furnished by a blower giving 600 to 700 cu. ft. air per min., is burnt in a small chamber adjoining the shaft holding the crucible. Again,[5] several single-burner furnaces have been combined in such a way that the gas from No. 1 passes into No. 2, No. 2 into No. 3, and No. 3 into the flue. The products of combustion usually encircle the mouth of the crucible and carry off with them

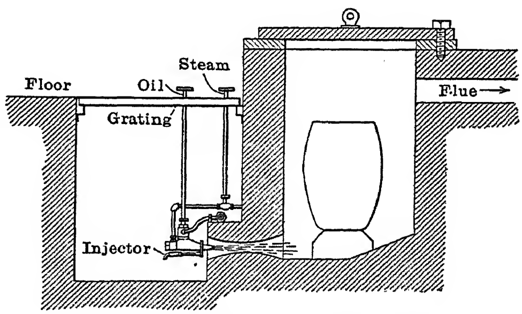


FIG. 322.—Oil-fired crucible-furnace.

[1] Teichmann-Bross, *Stahl u. Eisen*, 1911, XXXI, 843, 1049.
Krom, *Met. Ind.*, 1909, VII, 358, 404.
[2] Belfield, *Brass World*, 1909, V, 39.
Gourley, *op. cit.*, 1909, V, 355.
[3] *Met. Ind.*, 1906, IV, 187.
[4] Durham, *Tr. A. I. M. E.*, 1911, XLII, 875.
[5] *Cass. Mag.*, 1903–04, XXV, 73.
Examples: *Brass World*, 1909, V, 15 (Rockwell).

any fumes arising from the charge. In the Crane furnace[1] they are drawn off beneath the rim. The oil-heated furnace can, of course, be constructed along the lines shown by Piat, Baumann and Basse-Selve. Other examples of oil-fired furnaces are the makes of Holman Bros.[2] and the Hawley Down-draft Furnace Co.[3]

231. Crucible Furnaces Heated with Gas.—Illuminating gas and water-gas are, as a rule, too expensive to be used for melting large amounts of metal; such gas-fired furnaces are found in laboratories.[4] Producer gas is suitable only for furnaces holding several crucibles, as for economic work the gas has to be produced in quantities larger than called for by a single crucible; further, for high temperatures this weak gas, with its 60 vol. of N, as well as the air have to be preheated; this requires regenerative chambers (§325) which are expensive to build and maintain, and are profitable only with continuous work as is the case in making crucible steel.

An innovation in the burning of gas is the Bone flameless combustion[5] which looks promising.

232. The Process.—Little can be said from a general point of view about the process carried on in a crucible. The main reason for using an apparatus of such low efficiency is to protect the charge from outside influences, and the aim generally is to avoid oxidation and volatilization (see Preparation of Alloys, §43).

233. Retort, Muffle-tube, Cementation Furnaces (Kettles).—The first two forms of closed vessels are ordinarily employed when vapors are set free during the heating which are to be condensed and recovered; tube furnaces find some application in liquating, not so much at present as in former days; cementation furnaces are common for carburizing wrought iron and decarburizing cast iron; and kettles for melting, purifying and liquating metals and alloys of low melting-points. The shape and refractory nature of the vessel differ so much with the character (ore, metal, alloy), the size of the charge, and the temperature required that their discussion must become an enumeration of examples which are better treated with the several metals. With very few exceptions (retorting zinc-silver-lead crusts, amalgam, etc.) there is not in operation a single retort, muffle, tube or cementation box; usually a number of them are grouped together and heated from one fire. The furnaces, therefore, usually resemble reverberatory furnaces, the laboratories of which generally have more height and less length than those of the reverberatories common for smelting. Nevertheless, the general principles of construction for smelting-reverberatories apply to these closed-vessel furnaces; the fuel (solid, liquid or gaseous) to be burnt per hour must be adapted to the required volume and temperature of the laboratory, and supplied with the necessary air.

[1] Illustrated examples: Loomis, *Tr. A. I. M. E.*, 1891–92, XIX, 1013 and 1014.

[2] *London Min. J.*, 1908, LXXXIV, 788.

[3] *Met. Ind.*, 1910, VIII, 313.

[4] Reardon, Oil *vs.* Gas, *Met. Ind.*, 1910, VIII, 212.

[5] *Eng. Min. J.*, 1912, XCIII, 177, 309; *Met. Chem. Eng.*, 1912, X, 409; *Tr. A. I. M. E.*, 1912, XLIII, 612. . .*Feuerungstechnik*, 1912, I, 39, 62, 118.

234. Converters.—Converting originated in 1855 with Bessemer in his process for making steel out of pig iron. This remained its sole field until Manhès in 1880 applied the principle of converting to the production of metallic Cu from matte; later its use was extended to the enrichment of copper-nickel matte and nickel matte to a product containing respectively 80 per cent. Cu–Ni, or Ni, practically free from Fe.

A converter is a cylindrical iron vessel with conical hood, lined with refractory material. It has an air box from which high-pressure blast enters the cylinder through tuyère openings and oxidizes part of the liquid alloy or compound that has been poured in through the hood. The form of the vessel and with it the manner of operating has undergone many changes, both in the production of steel and of metallic copper, or copper-nickel matte. The lining of the converter is either acid or basic depending upon the character of the slag that is to be produced. A converter is either upright, Figs. 251 and 252, or horizontal, Figs. 253 and 254, depending upon the position of the main axis of the cylinder. Upright converters are either fixed or rotating. The fixed upright converter is always side-blown (Clapp-Griffith converter), *i.e.*, the blast impinges upon the metal but does not pass through it. A rotary upright converter, *i.e.*, one which swings in a vertical plane, may be bottom blown, when the blast passes through the bath (all large-size steel converters, Figs. 251 and 252), or side-blown (most small-size steel converters (Tropenas) and all matte converters, Figs. 253 and 254). A rotary bottom-blown upright converter may be concentric, *i.e.*, receiving on one side and pouring from the other, Figs. 251 and 252, or excentric, *i.e.*, receiving and pouring from one side; when side-blown it is always excentric. The horizontal converter is always rotating, *i.e.*, revolving on its main axis, and side-blown, Figs. 253 and 254.

The process of converting is always oxidizing; the rapid oxidation in the confined space of the converter of the constituents of the fluid charge which are to be removed furnishes the heat necessary for the process.

235. Minor Operations.—LIQUATING[1] means separating by fusion at a low temperature a metal or metallic compound from one that is less fusible or, in the case of some ores, from the gangue. The separation of metals by this process can only take place when their reciprocal solubility is very small and when they do not form a chemical compound; should they form a eutectic, this would separate out first. The process is imperfect even under the most favorable conditions, as the residue always retains some of the separated metal or compound. It is practiced in the purification (drossing) of Cu–Pb, Fe–Sn, the enriching of Zn–Ag–Pb crust, the desilverization of Ag–Cu (obsolete), the extraction of native Bi and Sb_2S_3 from their ores. The apparatus used are V-shaped troughs (antiquated), cast-iron kettles, presses, reverberatory furnaces, tubes, etc.

CRYSTALLIZING is the operation of separating from alloys that form eutectic mixtures the solvent metal by slow cooling and continuous stirring. It is practiced in the Pattinson processes and has been suggested for purifying Pb–Sn.

[1] Friedrich, *Metallurgie*, 1906, III, 13.

Distilling and sublimating is the heating of an ore or alloy to drive off a volatile metal or metallic compound and collecting it in the liquid or solid state. Thus Zn, Cd and Hg are distilled; As, As_2O_3, As_xS_y and HgS are sublimated; $Ni(CO)_4$ is distilled, but subsequently decomposed and Ni deposited. The form of apparatus depends mainly upon the affinity for O of the volatilized part. If this be small, as with Hg and As_2O_3, it may be desirable to admit O in which case the process becomes a volatilizing roast, and the condenser will have to be of large volume; if the volatilized part be readily oxidized as in the other cases, a closed-vessel furnace will have to be employed, the condenser will have to be small and care will have to be taken in heating the furnace and in keeping the condenser at the correct temperature.

Cementation.[1]—This is usually held to be a process by means of which two substances are heated in contact below melting temperature for the purpose of effecting a chemical change. In this sense it is employed in two ways: in making cement steel by heating wrought iron imbedded in charcoal powder (case-hardening) and thus carbonizing it; and in making malleable iron castings by doing the same with white iron imbedded in finely divided hematite (malleableizing) and thus decarburizing it. In olden times Ag was separated from Au by cementation,[2] the contact substance being S, KNO_3, NaCl. There is another and more local use of the term cementation, viz., the extraction of Cu from sulphide ore by means of weathering and leaching followed by precipitation with metallic Fe (cement copper).

236. Heat Treatment.[3]—This term is employed to denote the heating of a metal or alloy to a given temperature below its melting-point and the subsequent cooling in order to effect certain molecular changes. The cooling can be either slow in which case the metal will be annealed, or sudden in which case it will be quenched. The purpose of annealing is to release internal strain and make a metal tough and elastic which had been made brittle by stresses set up through mechanical treatment or rapid cooling from a more or less elevated temperature; also to cause diffusion of the constituents in heterogeneous alloys. The purpose of quenching is to set up internal strains and make a metal or alloy hard and brittle.

Metals that are to be rolled, hammered, etc., are usually first heated, as the mobility of the crystals is greater at a high than a low temperature, and the strain from the mechanical treatment correspondingly smaller. In some instances it is preferable to roll a metal cold and anneal at certain intervals in the mechanical treatment. Whatever may have been the temperature of the metal or alloy while it was being hammered or rolled, it will resist fracture under

[1] Sang, *Electrochem. Met. Ind.*, 1909, VII, 485.

[2] Percy, J., "Metallurgy of Silver and Gold," Murray, London, 1880, Part I, pp. 363, 379, 384.

[3] Guillet, L., "Traitement thermiques des produits metallurgiques, etc.," Dunod-Pinat, Paris, 1909.

Müller, "Effect of Annealing upon Some Non-ferrous Metals and Alloys," Metall-Erz, 1912–13, I, 219.

stress better if it receives a final annealing after it has received its final shape. The temperature and time required for the best results in annealing vary with the different metals and alloys. As a rule, heating up to a definite annealing temperature has to be gradual, and further, the temperature has to be maintained for a considerable time to permit the particles to rearrange themselves. The lower the temperature to which a metal is brought, the longer will be the time required for annealing. During the heating and annealing the metal must be protected from the effects of O, C, S, etc.; hence metals are often annealed in muffle furnaces or imbedded in sand. In case the annealing temperature is exceeded, the mobility of the particles becomes so great that they form large crystals which cause a deterioration of the physical properties; the metal becomes fissured, owing to the expulsion of gas (CO in steel) or vapor (Zn in brass), and pitted; the metal is said to have been "burnt." If it is desirable to retain some of the hardness or brittleness of the original metal, this may be attained by keeping the temperature just below that at which the annealing would be complete. In fact, strains may be in part released by keeping a metal for a long time at a temperature much below that required for complete annealing. The cooling down of the metal after annealing must be gradual and regular. For this purpose the metal may have to be imbedded in a poor conductor of heat; ordinarily, however, the firing of the furnace is stopped, the doors are closed, and the furnace is left to itself. After the critical temperature and with it the danger line has been passed, the doors of the furnace may be opened again and the further cooling of the metal hastened. Pure metals suffer less from internal strains than alloys; they are burnt only at temperatures very near their melting-points. Alloys are very much more susceptible and often have to be annealed within very narrow ranges of temperature.

QUENCHING is effected by plunging into cold water, oil or other liquid a metal or alloy that has been brought to a more or less elevated temperature. The general effect is that it approximately retains when cold the structural conditions prevailing at the high temperature at which it was held. In addition, the sudden cooling may set up internal strains by interfering with a normal crystallization because this now has to take place inside an expanded space surrounded by more or less rigid walls.

The terms hardening (chilling) and tempering are confined to the heat treatment of certain Fe–C alloys. Details of heat treatment can only be studied in connection with cooling- and freezing-point or the equilibrium diagram curves of the several metals and alloys.

CHAPTER IX

HYDROMETALLURGICAL PROCESSES AND APPARATUS

237. **Introduction.**—Two classes of processes are included in the general heading of hydrometallurgy, viz., *lixiviation* and *amalgamation.* The main distinguishing feature is the character of the solvent, for this governs the kind of materials and of operations. Usually lixiviation and amalgamation are carried on independently of one another; in some cases, as in the Pelatan-Clerici, or Rieken (gold) process they are combined for the solution of Au in KCN and its precipitation on a Hg cathode to form an amalgam.

Lixiviation and amalgamation followed smelting in the treatment of ores. The property of Hg to amalgamate Au was known to the Romans; Au was extracted by means of Hg from ores as early as the 12th century, and gold mills were in regular operation in the 14th century. The amalgamation of Ag was first practised in the 16th century; lixiviation followed amalgamation. While the Cu from mine waters was recovered by precipitating with Fe as early as the 14th century, leaching proper began probably with the Augustin process in 1843.

238. **Lixiviation in General.**—This embodies the separation of a soluble from an insoluble by means of a solvent, and the recovery of the dissolved substance by precipitation or evaporation.

239. **The Raw Material.**—This may be an ore, a speise, a matte, a metal, or an alloy. In an ore the metal, necessarily finely divided, is sometimes present in such a form as to be directly amenable to solvents (finely divided Au in KCN); more frequently it has to undergo a preliminary treatment. Usually this is a roast which may be oxidizing (Te-ide Au ores), sulphatizing (water-leaching some S-ide Cu ores), chloridizing (Cu-bearing burnt pyrite, many Ag ores; sometimes weathering (natural cementation of S-ide Cu ores) serves the purpose, occasionally the necessary decomposition is effected in the wet way (Doetsch process). Leaching in comparison with smelting requires little fuel and often gives a high yield; the main expense lies in chemical reagents, hence as a rule only low-grade ores with a gangue that is not attacked or only slightly so are subjected to this mode of treatment; the operation may be carried on at the mine; further, processes in which the solvent is regenerated are at a premium. Speise, which resists ordinary solvents, is usually dead-roasted before wet treatment. Matte is sometimes treated raw; usually it is first subjected to some roasting operation to eliminate all or part of the S, and to render the metal readily soluble. Metal or alloy is generally dissolved without any preliminary chemical treatment. In some instances ores are leached raw in lump form; in most cases, however, they are first reduced to a small size as is always the case with speise and matte.

240. The Solvents.—The common solvents are H_2O (most Met.Cl_x and Met.$_xSO_4$), aqueous solutions of gases (Cl for Au and perhaps sulphide ores[1]), of acids (conc. H_2SO_4 for Ag, dil. H_2SO_4 for Cu), of bases (NaOH for SnO_2), and of salts (KCN for Au, $Na_2S_2O_3$ for AgCl, NaCl for Cu_2Cl_2); in some instances gases are made to act directly (Cl upon Au ore and melted Au, H_2SO_3 and air upon Cu ore). The solvents used with ores are always dilute and sometimes slightly warm; speise and matte which are free from gangue allow the use of more concentrated solvents and higher temperatures than ores; this is still more the case with metals and alloys.

The density of solutions is usually determined by means of the Beaumé hydrometer. Table 163 gives the conversion values of degrees Beaumé and specific gravity for liquids heavier (metal solutions . . .) and lighter (oils, naphthas . . .) than water.

TABLE 163.—DEGREES BEAUMÉ AND SPECIFIC GRAVITY

Deg. Bé.	Sp. gr., liquid		Deg Bé.	Sp. gr., liquid		Deg. Bé.	Sp. gr., liquid	
	Heavier than water	Lighter than water		Heavier than water	Lighter than water		Heavier than water	Lighter than water
0	1.000	. . . . .	19	1.143	0.942	38	1.333	0.839
1	1.007	. . .	20	1.152	0.936	39	1.345	0.834
2	1.013	. . .	21	1.160	0.930	40	1.357	0.830
3	1.020	. . . .	22	1.169	0.924	41	1.369	0.825
4	1.027	. . . . .	23	1.178	0.918	42	1.382	0.820
5	1.034	. . . . .	24	1.188	0.913	44	1.407	0.811
6	1.041	. . . . . .	25	1.197	0.907	46	1.434	0.802
7	1.048	. . . .	26	1.206	0.901	48	1.462	0.794
8	1.056	. . . .	27	1.216	0.896	50	1.490	0.785
9	1.063	. . . . .	28	1.226	0.890	52	1.520	0.777
10	1.070	1.000	29	1.236	0.885	54	1.551	0.768
11	1.078	0.993	30	1.246	0.880	56	1.583	0.760
12	1.086	0.986	31	1.256	0.874	58	1.617	0.753
13	1.094	0.980	32	1.267	0.869	60	1.652	0.745
14	1.101	0.973	33	1.277	0.864	65	1.747	. . . .
15	1.109	0.967	34	1.288	0.859	70	1.854	. . . .
16	1.118	0.960	35	1.299	0.854	75	1.974	. . . .
17	1.126	0.954	36	1.310	0.849	76	2.000	. . . .
18	1.134	0.948	37	1.322	0.844	. . . . .	. . . .	. . . .

[1] Wolf, *West. Chem. Inst.*, 1910, VI, 176
Warwick, *ibid.* 169.
McCallum-Bruderlin, *ibid.*, 206.
Baker, *Tr. Am. Electrochem. Soc.*, 1907, XVII, 155.

241. Method of Leaching.—The mode of operating and with it the kind of apparatus will vary greatly with the raw material and the solvent. With an ore, there is a large charge and a dilute cold or slightly warmed solvent which extracts a small amount of metal leaving behind a large residue, *e.g.*, leaching Ag and Au ores; with an alloy most of the metal from a small charge is dissolved by a hot concentrated solvent, and the residue is likely to be insignificant, *e.g.*, parting doré silver with H_2SO_4. The terms *extraction* and *solution* are sometimes reserved for this class of operations.

In treating an ore, usually crushed finely, the size of particles influences the manipulation. The particles may be broadly classed as sands and slimes. In American ore-dressing practice the term "sands" is vague because it does not represent a definite range of sizes. As far as leaching is concerned, sands may be defined as particles sufficiently coarse to permit leaching by percolation, and slimes as those which do not. The pulp that flows from a fine-crushing machine may contain particles that pass a 20-mesh and a 100-mesh screen. The amounts of the different sizes formed in wet-crushing depend upon the nature of the ore, the character of the crushing machine and the mode of operating it. A quartzite will form less fines than a talcose or argillaceous ore; a machine that cracks the ore will make less fines than one that has an abrasive or grinding effect; crowding a machine will increase the percentage of fines. A low rate of leaching for sands, *i.e.*, the number of inches in depth a solvent will pass through a charge of ore in a vat per hour, is 1/2 in. (pan tailings), a high rate 12 in. (chloridized silver ore); quartzose ore will show a satisfactory rate with much finer sizes than argillaceous ore; with the former the limit is probably reached when 85 per cent. passes through a 150-mesh screen; with other ores the limit is reached with a much coarser screen. Coming to slimes proper,[1] they may be said to consist of three classes of substances: suspensions, colloidal suspensions, and colloids (=pectoids of Cushman-Hubbard). A suspension is a finely divided substance stirred up in water which will settle out gradually when left in repose; the addition of an electrolyte to the water has no effect upon the settling; the solid matter recovered by evaporating the water shows no change in its physical properties; the solid matter can be separated from the water by filtration. A colloid represents suspended particles which by carrying positive or negative electric charges repel one another continuously and thus counteract the clarify-

[1] Zsigmondi, R., and Jerome, A., "Colloids and Ultramicroscopy," Wiley, New York, 1909.
Julian, H. F., and Smart, E., "Cyaniding Gold and Silver Ores," Lippincott, Philadelphia, 1907, p. 213.
Richards, R. H. "Ore Dressing," McGraw-Hill Book Co., New York, 1909, Vol. III, p. 1397.
Whiting-Blake, *J. Am. Chem. Soc.*, 1904, XXVI, 1339.
Noyes, *op. cit.*, 1905, XXVII, 85.
Cushman, A. S., "The Effect of Water upon Rockpowders," *Bureau of Chem.*, U. S. Dept. Agriculture, *Bull.* No. 92, 1905.
Cushman, A. S., and Hubbard, P., "The Decomposition of Feldspars," *Office of Public Roads*, U. S. Dept. Agriculture, *Bull.* No. 28, 1907; also *Tr. Am. Cer. Soc.*, 1906, VIII, 180.
Ashley, *Tr. A. I. M. E.*, 1910, XLI, 380.

ing force of gravity; the addition of an electrolyte neutralizing the electric charges causes the particles to coagulate, whereupon they settle out and carry down with them ordinary suspensions; the solid matter recovered by evaporation has lost its colloidal property; the solid matter cannot be separated by filtration. A colloidal suspension represents a transition stage between the two extremes, and slimes belong to this category. As far as practical purposes are concerned, a slime may be defined as a colloidal suspension of finely crushed ore after 48 hours repose.

The coagulating power of an electrolyte depends upon the valency of the kation. With the value of a univalent kation equal 1, that of a bivalent is 30 to 40, that of a trivalent 500 to 1,000. If 1 part FeO-salt settles 2,000 parts slimy water, 1 part Fe_2O_3-salt will settle 60,000 parts; 1 part CaO settles 1,500 to 1,800 parts. The figures in Table 164 give the relative weights required to produce the same coagulating effects.

TABLE 164.—EFFICIENCIES OF VARIOUS COAGULATING SUBSTANCES[1]

Substance	Relative efficiency	Substance	Relative efficiency
Aluminum sulphate..........	100	Alum (pot. chrom.)	858
Alum (potash)..........	143	Calcium chloride........	1,095
Ferric iron....................	223	Calcium carbonate..........	1,215
Alum (ammonium)	252	Calcium sulphate	2,87c
Alum (am. chrom. iron).......	295	Magnesium sulphate	3,460
Lime.........................	654	Sodium chloride............	45,900
Magnesia..................	748	Sodium sulphate............	61,700

As regards the mode of operating, it may be said that if the ore permits ready filtration, the usual method of leaching by percolation in a stationary vat will accomplish the result (leaching chloridized ore); occasionally a stationary circular vat with stirrers attached to a vertical or horizontal revolving shaft, or a revolving barrel, or Hofmann's system of trough-lixiviation[2] may be employed to hasten solution. If the percentage of fines (slime) be too large, as is the case with many wet-crushed silver and gold ores, percolation becomes impossible or takes too much time; it is then necessary to separate sands from slimes by hydraulic classification, and treat the sands as indicated above, while the values in the slimes are recovered by agitating with the solvent. Such agitation is accomplished either in a stationary vat by stirrer-arms, a centrifugal pump, by compressed air, or in a revolving barrel. The separation of slime from solvent is effected by coagulating, and then either by settling and decanting, or by forcing through a filter-press, a pressure-filter, or a suction-filter.

The progress that has been made in the last ten years in the treatment of slimes in cyaniding silver and gold ores has brought about the present custom

[1] Julian-Smart, *op. cit.*, pp. 212, 220.

[2] Hofmann, *Tr. A. I. M. E.*, 1887–88, XVI, 662.

of purposely sliming ores (all-slime treatment), as frequently the saving in time and increase in yield more than balance the additional cost of crushing.

Finely-crushed roasted speise and matte are usually sufficiently porous to permit of the cheapest treatment, *i.e.*, percolation; with metal and alloy stationary vats are nearly always in use, and the solution is generally hastened by hand-stirring, sometimes by agitating mechanically (Walker apparatus for granulated copper).

In most leaching operations it is desirable to have a saturated solution for precipitation or evaporation and thus to use, at the same time, as little solvent as possible. This is accomplished by turning partly-charged solution upon fresh ore, and new solution upon ore that is nearly exhausted. This method of operating goes by the name of stage-lixiviation.[1] In a series of five or more tanks through which the solvent is to circulate, tank No. 1, *e.g.*, will contain nearly exhausted ore and receive new solvent, while tank No. 5, receiving liquor from tank No. 4, will contain fresh ore, and the solution passing from No. 5 will be saturated. After a while the ore in tank No. 1 will be exhausted and removed, and the tank will be charged with fresh ore; tank No. 2 will now become the starting tank for new solvent and No. 1 the finishing tank for solvent from tank No. 5. In metallurgical plants the solvent is usually circulated by running it off from one tank into an intermediary receiver and raising it on to the next following.

242. The Leaching Vessel.—This must be constructed of a material which will not be attacked either by the solvent or the solution. Thus with ores the tanks are made of wood,[2] steel or reinforced cement concrete[3] which may have to be coated with tar and pitch, paraffine- or rubber-paint; with speise and matte, wooden vats lined with sheet lead are in common use; metal salts are often dissolved in pans of sheet-lead and of cast iron or steel lined with lead; for parting precious-metal alloys, cast-iron, fayence, and porcelain vessels are in use.

Stationary vats in which materials are leached by percolation have a filter to separate the liquid from the solid. A filter is built of sized quartz gravel or acid-proof perforated brick covered with clean fine sand; or it is made of straw, cocoa-matting and cotton cloth, asbestos cloth, etc. Filters of the latter class always rest upon a perforated false bottom; the same is sometimes the case with those of the former. A wooden grating is always placed over the filter to protect it, if the residue is to be removed by shoveling. The leached ore is discharged from the ordinary stationary vat by shoveling, by an excavating machine, or by sluicing. From mechanically stirred vats filterable ores are discharged into stationary filtering pans or through pressure-tanks into filter-presses to separate the solution from the leached ore; some revolving barrels have internal filters.

[1] Davis, G. E., "Handbook of Chemical Engineering," Davis Bros., Manchester, 1904, II, 154.

[2] "Mechanical Properties of Redwood" by A. L. Heim, U. S. Dep. Agri., *Forest Service, Bull.* 193, Washington, 1912.

[3] *Eng. Min. J.*, 1909, XXXVIII, 180.

Filtered solutions are sometimes passed through clarifying or settling vats before the dissolved metal or salt is recovered in order to correct imperfections of the filter. In such vats they may also be subjected to some chemical treatment to remove harmful impurities, *e.g.*, addition of H_2SO_4 to $AuCl_3$ before precipitation with $FeSO_4$; reduction of $Fe_2(SO_4)_3$ to $FeSO_4$ by SO_2 in copper solutions to be treated with Fe to save in precipitant.

Filter presses and suction filters are discussed in §§306 and 308.

243. **Precipitation.**—The precipitants employed are solids, liquids and gases. Thus, metals are frequently used, such as Fe and Cu for Ag, Fe for Cu, Zn for Au and Ag; charcoal and Met. S serve to recover Au from $AuCl_3$. Common liquids are H_2O for Ag_2SO_4 in conc. H_2SO_4 or $PbCl_2$ in NaCl; Na_2S or CaS_5 for $Ag_2S_2O_3$; $FeSO_4$ or $FeCl_2$ for $AuCl_3$; Na_2CO_3 for $NiCl_2$; $CaCl_2O_2$ for $CoCl_2$. Gases are sometimes pumped through solutions, as *e.g.*, H_2S for AuCl, SO_2 for $AuCl_3$ to reduce it to AuCl; in a few instances they are made to ascend in a tower down which trickles the solution. Stirring a solution by hand or mechanically or with compressed air often has a beneficent effect upon the precipitation and the character of the precipitate.

For drying the precipitate reverberatory and muffle furnaces are in common use; drying chambers supplied with superheated steam or air heaters are uncommon; occasionally compression by machinery followed by furnace-drying is in operation. Usually, in dealing with large amounts of base-metal precipitate, this is charged straight into the drying apparatus; small amounts of precious metal are placed on trays.

Electric precipitation is discussed in §251.

244. **Concentration (Evaporation) and Crystallization.**[1]—Solutions from which the metal is to be recovered in the form of a crystallized salt usually have to be brought to a certain degree of concentration and temperature in order that well-developed crystals may form. Generally part of the water has to be removed; frequently the first-formed crude crystals have to be redissolved to free them from impurities and then recrystallized after clarifying or filter-pressing the solution.

The evaporation of part of the water is effected by heating with fuel or steam. With fuel, over-heat or under-heat may be applied. With the former, the products of combustion pass over the liquid, contained in a pan covered with a brick arch, and come in contact with it; where this is not permissible, under-heat comes into play, in which case the gases travel underneath and perhaps along the sides of a pan. Over-heat is more efficient than under-heat. In order to utilize the heat of the fuel in a satisfactory degree, the concentration pans, usually shallow oblong vats, are placed in series and are terraced, the dilute liquor enters at the flue end and passes off slowly at the bridge end with the right degree of concentration.

[1] Hausbrand, E., "Verdampfen, Kondensieren und Kühlen," Springer, Berlin, 4th ed., 1912.
Sadtler, "Notes on the Theory and Practice of Evaporation," *J. Frankl. Inst.*, 1908, CLXVI, 291, 395.
Nagel, "Evaporation by Means of Steam," *Electrochem. Met. Ind.*, 1908, VI, 334.

Fire-heated concentrating pans are of sheet iron lined with lead, sometimes simply of heavy sheet lead, the bottom resting upon cast-iron plates.

Concentration by means of steam is carried on by passing steam through lead coils placed inside of wooden lead-lined pans, or through double-walled iron pans also lead-lined. In order to obtain a good evaporative efficiency, it is necessary that the coil be not too long, that the steam be free from air, and that the condensed water be removed continuously. Steam evaporation is more costly than fire evaporation, but is a cleaner mode of procedure. Evaporation *in vacuo* is not used in metallurgical works.

Crystallization is effected by running the solution of right concentration and temperature, *i.e.*, saturated for the given temperature, into a lead-lined wooden vat, covering this, and keeping the charge in absolute repose. The formation of large-size crystals is further assisted by suspending strips of lead upon which the crystals grow. If the solution is stirred while recrystallizing, small imperfectly developed crystals (crystal meal) will form. These have the advantage over large crystals of including less mother-liquor and of being more easily washed to free them from impurities. Stirring, however, is rarely resorted to in metallurgical works. The mother-liquor retains with the impurities more or less of the salt that crystallized out. By further evaporation more crystals may be recovered, but they are sure to be less pure than those that separated at first. Before marketing, the crystals have to be washed superficially with water to free them from mother-liquor; they are then dried either by placing them in thin layers on shelves in a heated room, the air of which is continuously changed, or by passing them through a centrifugal machine—a cylindrical basket, covered with perforated metal or wire gauze, which is fixed to a vertical shaft revolving at a high speed. The dry crystals are ready to be shipped in wooden barrels. Sometimes they are broken in wooden rolls either before washing or after drying.

Handling of liquors is discussed in §295 and following.

245. Amalgamation in General.—This is a wet metallurgical process in which Hg acts as a solvent. It is confined to the extraction of Ag and Au from its ores, and is therefore based upon the property Hg has of dissolving these metals from their ores, when suitably prepared, and to form a Hg-alloy or amalgam. In the process, the amalgam, separated from the gangue by washing and freed from excess Hg by straining, is distilled to recover the Hg; the residual precious metal, impure from the presence of some base metal which has a tendency to combine with Hg, is fire-refined and cast.

Amalgams[1] are formed by the direct union of Hg with metal at ordinary

[1] L. de Mortillet, "Les Amalgams," Bernard and Tignol, Paris, 1904.
Abegg, R. W. H., "Handbuch der Anorganischen Chemie," Hirtel, Leipsic, 1905, II², 569.
Kerp-Böttger, "Zur Kenntniss der Amalgame," *Zt. anorg. Chem.*, 1900, XXV, 1.
Dudley, "The Nature of Amalgams," *Proc. Am. Assoc. Adv. Sc.*, 1889, XXXVIII, 145.
Read, "The Amalgamation of Gold Ores," *Tr. A. I. M. E.*, 1906, XXXVII, 57.
Humphreys, *J. Chem. Soc.*, 1896, LXIX, 243, 1679.
Roberts-Austen, *Proc. Chem. Soc.*, 1896, LXIX, 219.
Puschin, *Zt. anorg. Chem.*, 1903, XXXVI, 201.
McPhail-Smith, *Am. Chem. J.*, 1906, XXXVI, 125.

temperature (K, Na, Ba, Sr, Ca, Mg, Zn, Cd, Sn, Pb, Bi, Ag, Au, Cu, Pd) or upon warming (Al, Sb). Other metals, such as Mn, Fe, Ni, Co, Cr, Ur, Pt and Pt-metals, can be amalgamated only indirectly, *e.g.*, by the electrolytic decomposition either of their salts using Hg as a cathode or of a Hg-salt with the metal as cathode; by placing Na–Hg in the neutral concentrated solution of the metal salt; by inserting the metal into Na–Hg in the presence of H_2O; the result is a superficial amalgamation. Metals which unite directly with Hg do this also indirectly. The formation of amalgam may be attended by an evolution (K, Na, Cd) or absorption (Zn, Sb, Sn, Bi, Pb, Ag) of heat. When heat is evolved, the amalgam becomes electronegative to the amalgamated metal; when absorbed, electropositive; thus in the couple (Cd, Cd–Hg), Cd will be the negative pole; in the couple (Zn, Zn–Hg), Zn the positive pole.

Amalgams are liquid when there is a large excess of Hg, solid when the metal prevails; warming softens solid amalgam. From the liquid amalgam a large part of the Hg can be removed by straining the alloy through a cloth or chamois-leather bag. The finer the pores of the filtering medium and the greater the pressure with which the bag is squeezed, the drier will be the residual amalgam. The filtered Hg, however, is not pure, but remains charged with metal (see below), and the amount increases with the temperature (hot-straining). There is little change in volume by the formation of amalgam; contraction has been observed with Cu, Ag, Sn, Pb and some other metals. The thermal conductivity of some solid amalgams is greater than that of either of the component metals; this is the case with Sn, Zn and Bi. Heat decomposes all amalgams; the Hg is volatilized leaving behind a fused or more or less spongy residue; it is often difficult to remove all the Hg in distilling; with Au and Ag only fusion expels it completely. Amalgams of the alkali metals decompose water.

Many researches along every practicable line have been made to study the constitution of amalgams, but the results on the whole have not been satisfactory. Many amalgams are held to be solutions of metal in Hg, others diffusions,[1] some chemical compounds, perhaps also a few eutectics.[2] With most amalgams the question of constitution is still open. Thus, *e.g.*, Puschin[3] examining freezing-point curves, electromotive forces and microstructures concludes that Zn, Cd, Bi, Pb and Sn do not form any chemical compounds with Hg, that Zn- and Bi-amalgams are mechanical mixtures, and that Sn-, Pb- and Cd-amalgams are solutions. On the other hand, McPhail-Smith[4] finds that amalgams are frequently solutions of the compound Met. Hg_m (1 atom amalgamated metal to the molecule) in Hg, and that this is the case with Zn, Cd, Bi, Pb and Sn.

In carrying out the amalgamation process, the Hg, usually in a finely divided state, is brought into intimate contact with the pulverized Ag or Au ore; occa-

[1] Humphreys, *J. Chem. Soc.*, 1896, LXIX, 243.
Roberts-Austen, *Proc. Chem. Soc.*, 1896, LXIX, 219.

[2] Tammann, *Zt. phys. Chem.*, 1889, III, 441.

[3] *Zt. anorg. Chem.*, 1903, XXXVI, 201.

[4] *Am. Chem. J.*, 1906, XXXVI, 125.

sionally the ore is made to pass over or through a bath of Hg. In order that amalgamation may be successful, the following should be observed:

(1) The mineral must be amalgamable; if this is not the case it has to be made so by a preliminary treatment in the dry or the wet way. This preparation is less common with Au ores than with silver ores, as Au occurs more frequently in the metallic state than Ag.

(2) The ore must be finely pulverized in order to liberate the disseminated mineral and thus make it accessible to the Hg. The coarsest screen through which the ore is made to pass is probably 30-mesh and the finest perhaps 150-mesh. The apparatus for fine-crushing are stamp mills, roller mills, ball mills, arrastras, pans and tube mills. All these devices serve for crushing and amalgamating; barrels serve for amalgamation alone. Whenever a comminuting apparatus has a decided grinding effect, crushing and amalgamating must be carried on as separate operations, as otherwise the Hg becomes "floured," *i.e.*, converted into finely divided globules resembling a grayish powder the particles of which have become coated.

(3) The Hg must be "quick" and "charged." It is quick or bright when it is free from Hg compounds (such as Hg_2Cl_2, HgO), from base metals and their oxides, carbonates, sulphides, sulphates, from clay, talc; and grease which coat the particles, dull them, make them inactive and prevent the globules from reuniting. Chemically pure Hg is a poor amalgamator; it gains in power when it is charged, *i.e.*, contains at ordinary temperature that amount of Au (0.10 per cent.),[1] or Ag (0.05 per cent.),[1] which cannot be separated by straining through chamois leather. Small amounts of base metal, such as Zn, Pb or Cu may serve a similar purpose; any excess has a harmful effect, as it "sickens" the Hg and causes it to break up into fine particles (to flour).

(4) The water must have a certain degree of purity.

(5) Heat must be supplied; this is especially the case with Ag ores difficult to amalgamate.

(6) The mechanical as well as the chemical loss must be reduced to a minimum. Mechanical loss is due to flouring and to the use of an excess of Hg or of H_2O; chemical loss to the sickening of the Hg because of the presence of too much base metal, or because the Hg is chemically attacked by the electronegative component of the ore.

[1] Thesis of C. I. Auer, No. 206, Mining Dept., Mass. Inst. of Technology, 1901.

CHAPTER X

ELECTROMETALLURGICAL PROCESSES AND APPARATUS[1]

246. Introduction.—Electrometallurgy may be defined as that branch of metallurgy which uses electric energy, wholly or in part, for the production and treatment of metals. The electric energy may be converted entirely into heat for the purpose of producing the temperature necessary for a process, or it may be used to decompose a compound by wet electrolysis where the amount of heat generated is relatively small, or by dry electrolysis where the amount of heat generated is necessarily large.

247. Practical Electrical Units.—Before discussing the principles underlying electrometallurgical processes, it is well to recall the practical units for measurement.

1. STRENGTH OF CURRENT.—The unit of current is the international ampere (amp.), that is, the unvarying current which, when passed through a solution of silver nitrate in water in accordance with certain specifications, deposits silver at the rate of 0.001118 g. per sec. It is usually measured by an ammeter placed in series with the main circuit between the dynamo and vat, or furnace, so that the whole current may pass through it and the total number of amperes be read off. The instrument in common use is the Weston.

Occasionally a voltameter, or coulometer, as it is now frequently called, is employed to determine the average current strength. A copper voltameter, *e.g.*, has a thick copper anode and a thin copper cathode; the electrolyte is made up of 150 g. $CuSO_4+5$ aq. cryst., 50 g. conc. H_2SO_4, 50 c.c. C_2H_6O in 1,000 g. H_2O. As 1 amp. should deposit 0.01971 g. Cu. per min. or 1.183 g. Cu per hr., the weight of the deposited Cu will give the average value of the current for the time of deposition, and from this the efficiency may be computed.

2. CURRENT DENSITY.—This is the numerical value of the current in amperes divided by the area of the electrode, either anode or cathode. Several units, depending upon the unit of surface chosen, are in common use, such as amp. per sq. ft., per sq. cm., per sq. dm., per sq. in. The usual range in wet electrolysis is 0.5 to 40.0 amp. per sq. ft. With electrothermic processes and with

[1] Borchers, W., and McMillan, W. G., "Electric Smelting and Refining," Lippincott, Philadelphia, 1904. Minet, A., "Traité théorique et pratique d'Électro-métallurgie," Béranger, Paris, 1901. Blount, B., "Practical Electrochemistry," Macmillan, New York, 1906. Standard Handbook of Electrical Engineers, McGraw Publ. Co., New York, 1908. Crocker, "Theory of Electrometallurgy," *School Min. Quart.*, 1895, XVI, 97; Electrochemistry and Electrometallurgy, *School Min. Quart.*, 1901, XXII, 119. Gore, G., "Electrolytic Separation of Metals," Van Nostrand, New York, 1892. Kershaw, J. B. C., "Electrometallurgy," Constable & Co., London, 1908. Ashcroft, E A., "A Study of Electrothermal and Electrolytic Industries, McGraw Publishing Co., New York, 1909. Billiter, J., "Die Electrochemischen Verfahren der Chemischen Gross-industrie, Vol. 1; Electrometallurgie Wässriger Lösungen," Knapp, Halle, 1909. Foerster, F., "Electrochemie Wässriger Lösungen," Barth, Leipsic, 1905.

fused electrolytes this notation is feasible, but usually not practicable, hence the total amperes are usually given, and these go into the thousands.

3. QUANTITY OF ELECTRICITY.—The unit of quantity is the international coulomb, that is, the quantity of electricity transferred by 1 international amp. in 1 sec.

4. RESISTANCE:—Resistance is the property materials have of opposing the flow of current. The practical unit of resistance is the international ohm, *i.e.*, the resistance of a column of pure Hg 106.3 cm. long, of uniform cross-section (1 mm.), and weighing 14.4521 g. at 0° C. (The resistance of 100 yards iron telegraph wire equals about 1 ohm.) The resistance of any substance is proportional to its length and its specific resistance, and inversely proportional to its cross-section.

Conductance is the reciprocal of resistance; a conductor with a resistance of 2 ohms is said to have a conductance of 1/2 mho. The specific conductivity of the leading metals was given in Table 13; alloys (§54) are poorer conductors than the component metals; liquids are poorer than metals, and gases under ordinary conditions are practically non-conductors. Some of the leading metal-bearing minerals have been arranged by Kiliani[1] according to their conductance, as shown in Table 165.

TABLE 165.—ELECTRIC CONDUCTANCE OF ORE-FORMING MINERALS

Metal	Good conductor	Inferior or non-conductor
Silver	Argentite, pyrargyrite, proustite.	
Copper	Chalcocite, chalcopyrite, bornite.	Cuprite, azurite, malachite, tetrahedrite, chrysocolla.
Lead	Galena	Cerussite, pyromorphite, crocoite, wulfenite, anglesite, bournonite.
Cobalt	Smaltite, linnæite, cobaltite.	
Nickel	Gersdorffite, niccolite, rammelsbergite.	
Tin	Cassiterite	Stannite.
Zinc		Blende, calamine, smithsonite.
Antimony		Stibnite.
Iron	Pyrite, pyrrhotite, magnetite.	Marcasite, hematite, siderite, limonite, menaccanite, blackband.

[1] *Berg. Hültenm. Z.*, 1883, XLII, 237.

Artificial sulphides conduct similarly as the corresponding minerals. Most oxides at ordinary temperature are practically non-conductors; their resistance decreases as the temperature rises (§59). The resistance of metals increases considerably with a rise of temperature; that of alloys does not show much change, that of carbon,[1] liquids and gases decreases. The increase of resistance in metals with rise of temperature is smaller than the decrease in solutions, hence in working with high current densities an increased resistance due to the warming of conductors is more than balanced by the decreased resistance of the warmed liquid.

The specific conductivities of aqueous solutions of acids, alkalis and salts vary with the concentration; frequently there is a concentration at which the conductivity has a maximum value.

Acid and alkali solutions have higher specific conductivities than the salts of the same equivalent concentrations; chlorides, bromides and iodides are better conductors than sulphates, nitrates and carbonates. The specific resistances of aqueous solutions are high in comparison with those of fused electrolytes. In metallurgical operations the resistance is usually not measured directly, but obtained indirectly by reading the current and the voltage, and calculating according to Ohm's law.

5. Electromotive force is that which causes electricity to flow. The legal unit is the international volt, and this is defined as the electromotive force (e.m.f.) which, applied to an international ohm, will produce a current of an international ampere. It is practically equal to $\frac{1000}{1434}$ of the e.m.f. of a Clark standard cell at 15° C. In wet electrolytic processes the voltage of the direct current is low; it is higher in dry electrolytic processes; in electric smelting processes the alternating current is usually of a high voltage. The drop in potential between two points is measured by a voltmeter. The instrument commonly used is the Weston direct-reading voltmeter.

6. Electric Energy.—The unit of electric work or energy is the joule (J.). This is equal to the energy expended by 1 coulomb of electricity passing through a difference in potential of 1 volt; it is therefore equal to 1 volt-coulomb.

7. Electric Activity.—The rate at which electric energy is expended in a circuit is proportional to the current and to its e.m.f.; the unit in which it is measured is the watt, and this is equal to 1 joule per sec., or 1 volt$\times$1 amp. $=$1 volt-amp. $=$1 watt. In metallurgical plants watts are usually not measured, although wattmeters exist; ordinarily they are calculated. As in practical work, 1 watt is a very small unit, 1,000 watts$=$ 1 kilowatt (kw.) is usually substituted. Thus a dynamo capable of developing a current of 1,000 amp. at 60 volt$=$60,000 watts, is referred to as a 60-kw. machine.

Two other useful practical units are the ampere-hour and the watt-hour (kilowatt-hour). An amp.-hr. is the quantity of electricity passing through a circuit in 1 hr. in which 1 amp. is flowing. Thus a storage battery of 60 amp.-hr. will furnish a current of 6 amp. for 10 hr. or 10 amp. for 6 hr. One

[1] Somerville, *Met. Chem. Eng.*, 1912, x, 485.

amp.-hr. $=60 \times 60=3{,}600$ coulombs. 1 coulomb $=0.0002778$ amp.-hr. Ampere-hours are counted by ampere-hour meters or voltameters; similarly watt-hours by watt-hour meters.

In mechanics activity is measured in foot-pounds per second. An English horse-power represents 550 ft.-lb. work per sec. or 33,000 ft.-lb. per min.; it is also equal to 746 watts, which is called the electric horse-power. In order to change kilowatts into horse-power, the value of the activity in kilowatts has to be multiplied by 1 1/3, as 1 h.p. $=0.746$ kw., or 1 kw. $=1$ 1/3 h.p. Similarly for units of power, 1 kw.-hr. $=$ 1 1/3 h.p.-hr.

248. Ohm's Law.—The relation that exists between the electromotive force, the resistance and the resulting current in a given conductor is defined by Ohm's law, which states that the strength of a current varies directly as the electromotive force and indirectly as the resistance of the conductor.

Current $=\dfrac{\text{Electromotive force}}{\text{Resistance}}$, I (amp.) $=\dfrac{E\ \text{(volt)}}{R\ \text{(ohm)}}$, or by transposition, $E=I.R$, and $R=\dfrac{E}{I}$

In the formula E represents the total e.m.f. and R the total resistance. The total e.m.f. E of a circuit containing an electric cell is made up of E_s, the direct e.m.f. of the dynamo, and e the opposing or counter e.m.f. of the cell. The total resistance R is made up of R_g the internal resistance of the generator; of R_1 the external resistance of the conductors connecting the generator with the cell, and of R_c the internal resistance of the cell itself. The above formula extended then has the following form

$$I=\frac{E_s-e}{R_g+R_1+R_c}$$

249. Heating Effect of Electric Current (Joule's Law).[1]—A current flowing through any resistance develops heat. A current of 1 amp. flowing through a resistance of 1 ohm develops therein per second 0.24 g. cal.; hence 1 Joule-sec. $=0.24$ g. cal., and 1 g. cal. $=4.24$ Joules.

The general relation between electrical and heat energy is given by Joule's law which states that the heat H evolved by the passage of a current through a conductor is proportional to the square of the strength of the current I, to the resistance of the conductor R, and to the time t (in sec.) that the current lasts; or expressed in g. cal., $H=I^2Rt \times 0.24$ g. cal. $=EIt \times 0.24$ g. cal (as according to Ohm's law $R=\frac{E}{I}\Big)$. The formula $H=EIt \times 0.24$ g. cal. is in common use, as E (volt) and I (amp.) are more easily measured than resistances. In metallurgical work the hour t is more convenient than the second, when $H=860$ EIt g. cal.

The rise in temperature T in time t produced in a conductor by a current is inversely proportional to its weight w (in g.) and its specific heat s, or $T=\dfrac{0.24\ E\ I\ t}{w\ .\ s}$ presupposing that there is no loss in heat by radiation or conduc-

[1] Askenasy, P., "Electrothermie," Vieweg, Brunswick, 1910.

tion. This is approximately true, if the passage of the current lasts only a few seconds or if the body to be heated is enclosed in a non-conductor of heat. In practice there is a considerable loss of heat by radiation and conduction; the temperature rises, quickly at first and then more slowly until the heat produced is balanced by that which is lost, when an equilibrium will have been established and the amount of heat lost will be equal to that received.

1. APPLICATIONS.—The heat developed by the electric current has found application[1] in welding, annealing, heating for metal-working, smelting and other purposes. Laboratory apparatus,[2] such as tubes, muffles and crucibles, used in research work are heated by a current passing through a wire or ribbon of metal of high resistance (Ni, Pt . . .) wound around them, because of the perfect control of temperature they permit, and the ease with which they can be operated.

2. WELDING.[3]—There are two systems of electric welding. In the *Thomson System*[4] (sometimes called incandescent or resistance welding) metals which otherwise have no welding power can be readily united. The two pieces to be welded are held in a heavy copper-clamp vise and pressed end to end; an alternating current, taken from a step-down transformer in which a voltage of 250–1,250 volts is transformed down to 2–3 volts, passes through the joint, heats the abutted ends (by the resistance that the uneven surfaces offer) to the working temperature at which the metals either soften or fuse. The ends are usually made convex and softening under the heat become slightly upset. A little borax added to the joint slags any oxide that may have formed. The time necessary for welding is less than a minute; the pressure required for copper is 600, for iron 1,200, and for steel 1,800 lb. per sq. in.

In the *Coffin* or *Arc-welding System* an electric arc fuses, between the ends to be united, chips of metal of the same composition as that which is to be welded. It is really a soldering process in which the pieces to be welded form the negative and a carbon pencil the positive pole of an electric circuit carrying 200–300 amp. at 110 volts (Bernardo), or the pieces are wholly without the circuit, and an arc 2–2.5 in. is struck between two carbon pencils (Zerener). Arc-

[1] Richards, *Alum. World*, 1898, V, 105, 126, 149.

[2] Norton, *Elect. World and Eng.*, 1900, XXXVI, 951.

[3] Bramwell, *Proc. Inst. Civ. Eng.*, 1889–90, CII, 2.
Thomson *Tr. A. I. M. E.*, 1890–91, XIX, 877; *Cass. Mag.*, 1904, XXVI, 225.
Burr, *Iron Age*, 1892, XLIX, 149.
Royce, *op. cit.*, 1892, L, 977.
Editor, *op. cit.*, 1908, LXXXI, 1762.
Diegel, *Verh. Ver. Beförd. Gewerbefl.*, 1908, LXXXVII, 323, 355, 441.
Groth, L. A., "Welding and Cutting of Metals by Aid of Gases or Electricity," Van Nostrand, New York, 1913.
Hart, R. N., "Welding, Theory, Practice, Apparatus and Tests," McGraw-Hill Book Co., New York, 1910.
Loewenherz, *Zt. Verein. deutsch. Ing.*, 1911, LV, 1666; *Rev. Mét. Extr.*, 1912, IX, 403.

[4] Editor, *Eng. Min. J.*, 1888, XLVI, 413; 1889, XLVII, 136; 1890, L, 309.
Wood, *Tr. A. I. M. E.*, 1891, XX, 249.
Auel, *Proc. Eng. Soc. West. Pa.*, 1909, XXV, 213; *Iron Age*, 1909, LXXXIII, 1992.

welding has not been generally adopted on account of the high temperature of the arc which, *e.g.*, cannot but have a harmful influence upon steel, oxidizing C, Mn and Si, or adding C to the joint which is deposited from the positive to the negative pole.

3. ANNEALING.—Medium-C steel plates superficially carburized (skin-hardened) by case-hardening have been annealed by the Thomson system in places where holes are to be bored for rivets. Two copper poles about 2 in. sq. are placed on the smooth plate leaving a distance of 1–2 in. between them, a current of 3,500 amp. at 4 volts is turned on gradually and increased to 6,000 amp.; the space between the poles is brought to the temperature necessary for effecting the structural change, held there for some time and the current strength gradually lowered for the temperature of the steel to sink below the critical range.

Annealing wire between drawings is easily accomplished by making it pass around two isolated wheels connected with the poles of a circuit, while it is being unrolled from one coil and rolled up on another. The wire is heated to the desired temperature while it travels between the two wheels.

4. HEATING FOR METAL-WORKING.—The Burton Water-forge (the Lagrange and Hoho Systems are similar) may serve as an example. A bucket is filled with a solution of borax and soda, a large lead plate inserted forms the positive pole, a small piece of bent sheet copper connected with the negative pole is hooked over the edge of the bucket but does not touch the solution; a bolt, *e.g.*, to be heated is gripped with tongs, plunged into the solution, the tongs toward the middle resting on the negative copper pole, the circuit with a tension of 50 to 100 volts (25 to 35 volts is the minimum to start the arc) is thereby closed and the solution decomposed. H collects on the cathode, offers sufficient resistance to the current for an arc to form which quickly brings the metal to a welding heat, when it is withdrawn. The H protects the metal from oxidation, and the large surface of lead allows the O to pass off readily so that no arc is formed there. The passage of the current heats the solution, but only to a small extent. This method of heating is desirable when it is essential that the metal be kept perfectly clean.

Another method of heating is that of the General Electric Co., Schenectady, N. Y.[1] It consists in immersing the articles to be heated in a bath of fused metallic salts as, *e.g.*, equal portions of $BaCl_2$ and KCl. This is contained in a box of fire-brick surrounded by asbestos enclosed by a sheet-iron casing; it is fused by means of an alternating current which passes through two electrodes connected with the low-tension side of a transformer and placed on opposite sides of the furnace in direct contact with the contents. In starting the furnace, an arc is struck across the broken flux by means of an auxiliary electrode which quickly melts the salts. The temperature of the bath is easily raised or lowered by increasing or decreasing the current passing through it. Electrically heated

[1] *Bull.* No. 4737, May, 1910.

oil-tempering baths[1] ranging in temperature from 220 to 356° C. are similar in principle, but differ greatly in the manner of heating.

5. SMELTING.—In electric smelting, the high temperature of the arc (3,700° C.) is utilized to reduce by means of C metallic oxides which at the lower temperatures of carbon-heated furnaces (1,600° C.) would not give up their O. Mabery[2] in 1885 was the first to state that all metallic oxides could be reduced by C at the temperature of the electric arc; Moissan[3] the first[4] to demonstrate the general proposition by extended experiments. The reduced metal will be carburized if it has any affinity at all for C.

The total amount of heat developed by the electric current is that due to the electrical energy plus or minus that due to the chemical changes that take place. Thus $H=0.24\ EIt-w.h+w_1h_1-$(latent heat, radiation, conduction, etc.); wherein w=g. of metal reduced, h=heat of combination per g., w_1=g. C required for reduction, h_1=heat of combination per g.

On account of the high cost, electric smelting is practised under normal economic conditions only when high temperatures are absolutely necessary (as in the C-reduction of Al_2O_3); in special cases it is used to supply heat when the cost of fuel is abnormally high and water-power is available.

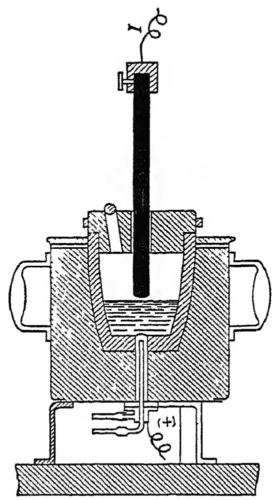


FIG. 323.—Siemen's electric-arc crucible-furnace.

The first to apply the electric arc to fusion was W. Siemens[5] who fitted into the bottom of a crucible, to receive the charge, at first an electric-light carbon and later a water-cooled copper casing, Fig. 323, to form the positive pole. The negative pole was formed by a carbon rod suspended centrally in the crucible by a solenoid arrangement which automatically was to maintain constant the distance between the two poles. The current entering through the charge, crosses the air-gap to the negative pole, forms an arc and fuses the metal quickly. This method of melting has not been applied commercially. While the efficiency of the furnace was high,[6] the water-cooled pole wore through quickly and became dangerous; the regulation of the current was imperfect, the crucible became highly heated by the current passing through it and therefore short-lived, and the melted Fe absorbed C and Cu.

In 1885 Cowles Bros. of Cleveland, O., began to produce Al–Cu and Al–Fe alloys by arc-smelting, and later at Lockport, N. Y., manufactured other alloys

[1] Fulwider, *General Electric Rev.*, Nov., 1910, through *Metallurgie*, 1910, VII, 157; *Bull. General Electric Co.*

[2] *Am. Assoc. Adv. Science*, 1885, XXXIV, 136.

[3] Moissan, H., and Lehner., V., "The Electric Furnace," Easton, Pa., 1904.

[4] Priority Claim of Borchers, *Eng. Min. J.*, 1893, LVI, 3.

[5] Siemens-Huntington, *Rep. Brit. Assoc.*, 1882, 496.
Siemens, Sir C. W., "Collected Works," Murray, London, 1889, II, 230.

[6] Richards, *J. Frankl. Inst.*, 1907, CLXIV, 446.

from oxides difficult of reduction. In the iron and steel industry in recent years electric smelting has been applied in the reduction of iron ore, in the production and refining of steel and other ferro-alloys. There is an opening for it in foundry work.[1] Electric smelting of mixed zinc ores promises well. The leading features of modern electric furnaces have been given in §167.

250. Electrolytic Work.—The first to apply the principle of electrolysis to the treatment of ores was Becquerel[2] who in 1836 gave sulphide ore a sulphatizing or chloridizing roast, dissolved the sulphates or chlorides formed and electrolyzed the solutions using insoluble anodes. This method, proving too slow and expensive, was not used until in recent years when it has been taken up again in a few instances.

The direct electrolytic treatment of ores without any preliminary treatment was undertaken by Clinton (1840) and Werdeman (1880) in England, and by Blas and Miest[3] (1880) in Belgium, but without lasting success. All minerals, excepting metals and sulphides, are poor conductors of electricity and even with artificial sulphides, such as mattes, it has not been possible to recover one metal alone in the presence of others. When, however, only one metal is present, as *e.g.*, in 80 per cent. copper matte, the matter assumes a different aspect (Günther process). Elkington in 1865[4] was the first to desilverize and refine electrolytically impure metallic copper. The refining of impure metals by means of the current has become the leading metallurgical application of wet electrolysis.

In 1852 Bunsen[5] decomposed fused $MgCl_2$ by electrolysis, and in 1854 he[6] and St. Claire Deville[7] separately published their results on the electrolytic reduction of $AlCl_3$. To-day all Al is produced by dry electrolysis. Other applications have hardly passed the experimental stage.

251. Wet Electrolysis, General.—When a continuous or direct current passes through a metallic conductor (conductor of the first class) no chemical action takes place; when it passes from a conductor of the first class into an aqueous solution of an acid, base or salt (conductor of the second class or electrolyte) a chemical change takes place in the latter. Conductors of the first class through which the current enters and leaves the electrolyte are called electrodes. The electrode through which the current enters the electrolyte is the anode; the electrode through which it leaves, the cathode. The charged atoms or atomic groups into which the electrolyte is dissociated[8] and which carry the current are

[1] Richards, *Met. Ind.*, 1910, VIII, 456.
Weeks, *Met. Chem. Eng.*, 1911, IX, 363.
Clamer-Hering, *Brass World*, 1912, VIII, 355.

[2] Becquerel, "Eléments d'Électro-chimie," Didot, Paris, 1864, 528.

[3] Blas, C., and Miest, E., "Essai d'application de l'Électro-analyse," Gauthier, Louvain-Paris, 1882.

[4] Engl. Patent No. 2838, Nov. 3, 1865.

[5] *Liebig Ann.*, 1852, LXXXII, 137.

[6] *Pogg. Ann.*, 1854, XCII, 648.

[7] *Ann. Chim. Phys.*, 1854, XLIII, 27.

[8] Stevenson, R., Chemical Reactions at Electrodes, *Met. Chem. Eng.*, 1910, VIII, 240.

called ions. They are liberated at the electrodes and may either remain uncombined, or react with the water, the electrode, or the electrolyte, depending upon the nature of the ion liberated. The negatively charged ions which wander toward the anode (acid radicals or hydroxyl) are called anions; the positively charged ions which wander toward the cathode, kations (H, metals, basic radicals).

Cathode		Compound		Anode
H_2	$\leftarrow$	$2HCl$	$\rightarrow$	Cl_2
Cu_2	$\leftarrow$	$CuCl_2$	$\rightarrow$	Cl_2
H_2	$\leftarrow$	H_2SO_4	$\rightarrow$	SO_4
Na_2	$\leftarrow$	$2NaOH$	$\rightarrow$	$(OH)_2$

The metal Na set free at the cathode cannot exist in contact with water, as it decomposes it: $Na_2+2H_2O=2NaOH+H_2$, hence H_2 will be liberated there. The hydroxyl $(OH)_2$ set free at the anode is split thus: $2(OH)_2=2H_2O+O_2$, so that at the anode there is liberated O. The acid radical (SO_4) set free at the anode acts upon water: $2SO_4+2H_2O=2H_2SO_4+O_2$, so that here also O will appear at the anode.

252. Dissociation and Conductivity.[1]—Before the conditions under which substances existed in solution were understood, it was believed that acids, bases and salts in aqueous solution remained in their molecular conditions, that a current passing through a solution caused the molecules pointing haphazard in any direction to arrange themselves in chains (Grotthus chains) with their positive constituents pointing toward the cathode and their negative constituents toward the anode, and that molecules were disrupted when the applied e.m.f. was stronger than the force that bound them together, the cation being set free at the cathode, the anion at the anode; that this was followed by an interchange all along the chain, a positive constituent combining with a negative constituent of the adjacent molecule and *vice versa*, thus forming by a double migration of positive and negative constituents new molecules which were again decomposed at the poles.

Modern conceptions of electrolysis are based upon the Arrhenius theory of electrolytic dissociation which states that many compounds when dissolved are dissociated more or less into electrically charged positive and negative ions distributed uniformly throughout the solution. When an e.m.f. is applied at two electrodes placed in such a solution, the positive pole attracts the negative ions which begin to migrate toward it, and the negative pole the positive ions; the ions give up their electric charges at the electrodes and become electrically neutral atoms or complexes which, in the absence of a substance with which

[1] Lehfeldt, R. A., "Electrochemistry," Longmans, Green & Co., London, 1904.

Abegg, R., and von Ende, C. L., "The Electrolytic Dissociation Theory," Wiley, New York, 1907.

Le Blanc, M., Whitney, W. R., and Brown, J. W., "Text-book of Electrochemistry," Macmillan, New York, 1910.

they can form a chemical compound, combine to form less active molecules; since the potential at the electrodes is maintained from an extraneous source, the electrolysis is a continuous process.

The actual weight of an ion is not known; its composition, however, is given by the chemical formula, and calculations are based upon its gram equivalent, *i.e.*, the weight of the gram ion divided by the valency; the weight of a monovalent silver ion is equal to its atomic weight in g. or 108. The electric charge carried by a monovalent ion is 96,660 coulomb, *i.e.*, the quantity of electricity necessary to deposit 1 g. H' or 108 g. Ag'; the charge carried by a divalent ion is $2 \times 96{,}540$ coulomb, etc. The leading monovalent anions are: Br', Cl', I', F', OH', BrO_3', ClO_3', IO_3', NO_3', CN'; the bivalent anions: S'', Se'', Te'', SO_4'', SeO_4'', SiO_3'', CO_3'', C_2O_4'', CrO_4''; the leading monovalent cations: $H^{\cdot}$, $K^{\cdot}$, $Na^{\cdot}$, $Ag^{\cdot}$ and the -ous compounds of $Cu^{\cdot}$ and $Hg^{\cdot}$; the leading bivalent cations: $Zn^{\cdot\cdot}$, $Pb^{\cdot\cdot}$, $Cd^{\cdot\cdot}$, $Co^{\cdot\cdot}$, the -ous compounds of $Fe^{\cdot\cdot}$, $Mn^{\cdot\cdot}$, $Ni^{\cdot\cdot}$, $Sn^{\cdot\cdot}$, and the -ic compounds of $Cu^{\cdot\cdot}$ and $Hg^{\cdot\cdot}$; the leading trivalent cations: $Al^{\cdot\cdot\cdot}$ and the -ic compounds of $Cr^{\cdot\cdot\cdot}$, $Fe^{\cdot\cdot\cdot}$, $Au^{\cdot\cdot\cdot}$, $Mn^{\cdot\cdot\cdot}$ and $Ni^{\cdot\cdot\cdot}$, the leading tetravalent cation: $Sn^{\cdot\cdot\cdot\cdot}$ in stannic compounds.

All molecules of a substance in solution are not dissociated. The degree of dissociation for specific dilutions varies with the substance. As the conductivity depends in part upon the number of electrically charged ions, the greater the dissociation the larger the conductivity. The conduction, however, further depends upon the rapidity of motion and the quantity of electricity carried by each ion. Dissociation increases with increasing dilution, and is theoretically complete only at infinite dilution. On the other hand, the larger the number of ions (*i.e.*, the more concentrated the solution), the greater the conduction. This explains the fact that solutions of HCl, HNO_3, H_2SO_4, KOH, $NaOH$, etc., show the greatest conductivity when they are moderately concentrated.

A few examples of dissociations are given in Table 166.

TABLE 166.—DISSOCIATION OF SOME ELECTROLYTES

Concentration $1/v$	Percentage dissociation						
	HCl	KCl	$KC_2H_3O_2$	$\frac{1}{2}$ $BaCl_2$	$\frac{1}{2}$ H_2SO_4	$\frac{1}{2}$ $ZnSO_4$	$\frac{1}{2}$ $CuSO_4$
0 0001	....	99.2	99 0	99.2	99.2	98 9	95.1
0.001	99.2	97 9	97.3	96 2	95.9	89.0	93.3
0.01	97.4	94 1	93.1	88 6	87.9	66.4	60.7
0.1	92.4	86.1	83.0	75.9	71.3	41.8	37.8
1.0	79.2	75.5	62.8	57.9	53.4	24.1	21 7

v = number of liters in which 1 equivalent weight is dissolved.

Ions, as shown by Hittorf, migrate through the electrolyte at different rates of speed. The result is that, as equivalent quantities of anions and kations are

set free at the anode and cathode, the concentration of the electrolyte at the anode and cathode becomes changed. This explains the fact that, *e.g.*, in electrolyzing a $CuSO_4$-solution, the electrolyte becomes impoverished in Cu at the cathode and enriched in SO_4 at the anode, and requires stirring, by some means or other, to obtain a uniform concentration and thereby a uniform deposition.

253. General Rules for Electrolysis. 1. CONDUCTION.—Electrolysis cannot take place unless the solution be a conductor, and conduction is effected by means of the movement of electrically charged ions.

2. CHEMICAL ACTION IN SINGLE CIRCUIT.—Equivalent amounts of substances are liberated at the electrodes, as the current in a circuit is equal at all parts. There is chemical action only at the electrodes. In Fig. 324 representing a single circuit, C = cell, $VV'V''V'''$ = copper voltameters arranged in series. The weight of Cu deposited on each cathode in the same time is the same.

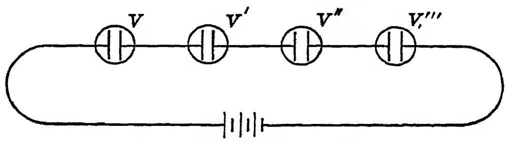


FIG. 324.—Single circuit.

3. FARADAY'S FIRST LAW.—The amount of an ion liberated by an electric current in a given time is proportional to the quantity of electricity passing through the electrolyte. It is, therefore, independent of the strength or temperature of the electrolyte, of the size and distance apart of the electrodes. In Fig. 325 representing a divided circuit, C = cell, $VV'V''V'''$ = voltameters. In the two branches V and V''' the weights of Cu will be equal and the same as in the single circuit of Fig. 324, as the same number of amperes pass through them in a given time; the weights of Cu deposited in V' and V'' will also be equal, but only one-half of those of V and V''', as only one-half of the number of amperes passes through them in a given time.

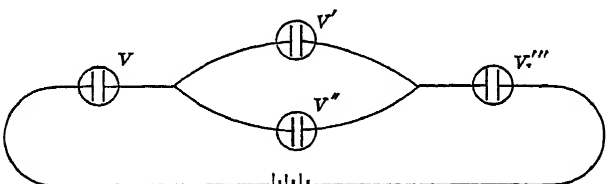


FIG. 325.—Divided circuit.

4. FARADAY'S SECOND LAW.—When a current of electricity passes through a number of electrolytic cells connected in series, *electrochemically equivalent quantities* of substances are set free. 1 coulomb (=1 amp. per 1 sec.) liberates 0.000010384 g. H., which is the e.c.e. of H.; a current of I amp. liberates $I \times$ 0.000010384 g. H. in a second. In order to find the e.c.e. of any other element, multiply its chemical equivalent $\left(i.e., \frac{\text{atomic weight}}{\text{valency}}\right)$ by 0.000010384. Thus,

Ag $\frac{108}{1} \times 0.000010384 = 0.0011180$ g.

Cu from cupric salt $\frac{63.5}{2} \times 0.000010384 = 0.000328$ g.

Cu from cuprous salt $\frac{63.5}{1} \times 0.000010384 = 0.000657$ g.

In order to find the weight of an element deposited by electrolysis, multiply the e.c.e. by the strength of current (amp.) and the time (sec.) during which it flows. A current of 1 amp. flowing for 1 hr. through a vat containing divalent $CuSO_4$ will deposit

$$Cu = \underset{\text{e.c.e.}}{0.000328} \times \underset{\text{amp.}}{1} \times \underset{\text{sec.}}{(60 \times 60)} = 1.1832 \text{ g.}$$

This theoretical figure is, however, never reached in practice. The efficiency of deposition in refining is rarely much higher than 85 per cent. It is less at the anode than at the cathode on account of the impurities in the former.

TABLE 167.—ELECTROCHEMICAL EQUIVALENTS AND GRAMS METAL DEPOSITED PER AMPERE-HOUR[1]

Anion	Electrochemical equivalent	g. per amp.-hr.	Cation	Electrochemical equivalent	g. per amp.-hr.
Br'	0.828	2.982	$Ag^{\cdot}$	1.1181	4.025
BrO_3'	1.326	4.772	$Al^{\cdot\cdot\cdot}$	0.0936	0.3369
Cl'	0.3672	1.322	$As^{\cdot\cdot\cdot}$	0.2590	0.932
ClO_3'	0.864	3.112	$Au^{\cdot\cdot\cdot}$	0.681	2.451
CHO_2'	0.4662	1.678	$Ba^{\cdot\cdot}$	0.712	2.562
$C_2H_3O_2'$	0.611	2.201	$Ca^{\cdot\cdot}$	0.2077	0.748
CN'	0.2694	0.971	$Cd^{\cdot\cdot}$	0.582	2.096
CO_3''	0.3108	1.119	$CO^{\cdot\cdot}$	0.3056	1.100
CrO_4''	0.4558	1.641	$Cr^{\cdot\cdot\cdot}$	0.1799	0.648
CrO''	0.601	2.165	$Cu^{\cdot}$	0.659	2.372
F'	0.1968	0.709	$Cu^{\cdot\cdot}$	0.3294	1.186
I'	1.315	4.735	$Fe^{\cdot\cdot}$	0.2895	1.042
IO_3'	1.813	6.525	$Fe^{\cdot\cdot\cdot}$	0.1930	0.695
NO_3'	0.642	2.312	$H^{\cdot}$	0.01044	0.03759
O''	0.0829	0.2983	$Hg^{\cdot\cdot}$	2.072	7.46
OH'	0.1762	0.634	$K^{\cdot}$	0.4056	1.460
SiO_3''	0.3957	1.425	$Li^{\cdot}$	0.0728	0.2622
S''	0.1661	0.598	$Mg^{\cdot\cdot}$	0.1262	0.4542
Se''	0.4102	1.477	$Mn^{\cdot\cdot}$	0.2849	1.025
SO_4''	0.4975	1.791	$Na^{\cdot}$	0.2388	0.860
Te''	0.661	2.379	$Ni^{\cdot\cdot}$	0.3040	1.094
....	....	....	$NH_4^{\cdot}$	0.1869	0.673
....	....	....	$Pb^{\cdot\cdot}$	1.072	3.858
....	....	....	$Sb^{\cdot\cdot\cdot}$	0.415	1.494
....	....	....	$Sn^{\cdot\cdot}$	0.616	2.219
....	....	....	$Sn^{\cdot\cdot\cdot\cdot}$	0.3082	1.109
....	....	....	$Sr^{\cdot\cdot}$	0.4537	1.633
....	....	....	$Te^{\cdot\cdot\cdot\cdot}$	0.3305	1.190
....	....	....	$Tl^{\cdot}$	2.114	7.61
....	....	....	$Zn^{\cdot\cdot}$	0.3387	1.219

[1] Standard Handbook of Electrical Engineers, McGraw Hill Book Company, New York, 1908, p. 968. See also Hering, *Electrochem. Ind.*, 1903, 1, 170.

5. Electrolytic Solution and Osmotic Pressure.—In order to explain the solution of the different metals in the anodes used in electrolytic refining processes, and the deposition of metals upon the cathodes, it is necessary to give a brief statement of the Nernst theory of the potential of "reversible metallic electrodes." By reversible metallic electrode is understood a metal dipping in a solution of one of its salts. According to this theory the e.m.f. between the solution and the metal is given by the equation:

$$e=\frac{RT}{n}\log.\frac{P}{p}=\frac{0.058}{n}\log_{10}.\frac{P}{p}$$

for room temperature, in which R=gas constant, T=absolute temperature, n= valence of metal, P=electrolytic solution pressure, p=osmotic pressure of metallic ions. When a piece of metal is dipped into such a solution, one of three things may happen according as $P>p$, $P<p$, $P=p$.

If $P>p$, some of the metal goes into solution, the ions charging the solution positive and the metal negative. This dissolving continues until the electrostatic attraction between the electrode and solution plus the osmotic pressure are sufficient to balance the electrolytic solution pressure.

If $P<p$, some of the metallic ions in solution deposit upon the electrode charging the metallic electrode positively and the solution negatively. The electric force tending to send the metal into solution is now in the opposite direction from that in the former case, it is added to the value of P; and when the sum of this force and of P equals p, equilibrium is again established.

If $P=p$, nothing happens, and e.m.f.=zero.

The quantity P refers to the metal in the pure state; if this is made less concentrated by diluting it with some other more electronegative metal the value is changed; the same is the case if the metal is deposited upon another metal in a very thin film.

To consider separately the action at the anode and at the cathode:

The Anode.—A pure metal dipping in a solution of one of its salts may be dissolved or deposited, if the potential difference between it and the solution is changed the slightest amount from the potential which it exhibits in the solution. It will be dissolved if this small externally applied electromotive force is directed from the metal to the solution; it will be deposited if the electromotive force applied is oppositely directed. An impurity in an anode will always dissolve if it is more electropositive than the principal metal of the anode; it may be immediately precipitated by some salt contained in the solution. Metals more electronegative than the principal metal of the anode will not dissolve; if in solution they will be precipitated by the principal metal of the anode. Whether an impurity is more electropositive or more electronegative than the principal metal depends upon the value P, its electrolytic solution pressure, and the value p, the osmotic pressure of its ions in solution. The value P of a pure metal changes when this contains a small amount of another metal as impurity; the value of p for the impurity is much smaller than for the principal metal. It may happen that when the solution is fresh and does not contain much impurity, a

given impurity will be electropositive to the principal metal, while after it has accumulated in solution, it will become electronegative.

THE CATHODE.—Here the same considerations apply. If an impurity is relatively electronegative, it will be deposited; if electropositive, it will not. But as above, its behavior depends upon its concentration. When dilute, it may be relatively electropositive and will not be deposited; upon becoming more concentrated, it may have the same potential difference as the principal metal and be deposited with it.

At the cathode there has also to be considered the liberation of H. The e.m.f. at which H is liberated on a metal cathode depends upon the character of the metal and finds expression in $e=e'+RT \log. \frac{P}{p}$, in which $P=$ electrolytic solution pressure of H, $p=$osmotic pressure of H-ions, and e' the "over-voltage" for metal of which the cathode consists. If the potential drop from the solution to the cathode is equal to or greater than $e'+RT \log. \frac{P}{p}$, then H will be deposited; if less, it will not.

In the above discussion it has been assumed that all the metals form soluble salts with the anion of the electrolyte. Any metal that forms an insoluble salt with the anion cannot exist, of course, in solution to any appreciable extent. If it is attacked by the current, it will adhere to the anode or drop off into the anode mud.

In the case of two electrodes of the same metal dipping into a solution of one of its soluble salts, as much energy is used at one electrode as at the other when a current is passed. This explains why the smallest e.m.f. sends a current through such a cell.

The leading metals arranged in the order of their electrolytic solution pressures give the following series:

ELECTROPOSITIVE: Na, Ca, Mg, Al, Mn, Zn, Cd, Fe, Co, Ni, Sn, Pb, H, Bi, Sb, As, Cu, Hg, Ag, Pd, Pt, Au: ELECTRONEGATIVE.

6. CALCULATION OF ELECTROMOTIVE FORCE (THOMSON RULE).—The basis of the relation between electrical and chemical energy, as shown, is that 1 coulomb deposits 0.011181 g. of Ag or liberates 0.000010384 g. of H. In order to set free 1 g. H or 1 gram-equivalent of any other element, an expenditure of $\frac{1}{0.000010384}=96{,}540$ coulomb is required; this quantity of electricity is called a Faraday and is usually denoted by the letter F.

It is often desired to calculate the e.m.f. necessary to decompose a given chemical compound, or to find the e.m.f. that would be produced in a battery from the union of given chemicals. This may be done approximately from the heats of combination. It was formerly supposed that the heat of combination and the electrical energy were equal to one another; or if $Q=$the heat energy of the union of one molecular weight of a substance, then the e.m.f. could be

calculated from the equation $nEF=Q\times 4.19$, in which $n=$ the number of equivalents in one molecular weight. Since $F=96,540$, we obtain $E=\frac{Q}{n\times 23040}$. This is known as Thomson's Rule. The rule is incorrect, as the true relation between heat and electrical energy is given by the Gibbs-Helmhotz equation $nEF=Q+T\frac{dE}{dT}$, in which $T=$ absolute temperature, and $\frac{dE}{dT}$ the temperature coefficient of the e.m.f. As this coefficient is generally not large, Thomson's rule is used sometimes to give an approximate value.

With a compound, consisting of two elements, the products of decomposition will be the original components:

$$H_2+O=H_2O+69,000\text{, liq., g. cal., } n=2 \text{ valencies,}$$

$$\text{hence } V=\frac{69000}{2\times 23040}=1.49=1.5 \text{ volts.}$$

$$Cu+Cl_2+aq.=62,500 \text{ g.cal., } n=2 \text{ valencies,}$$

$$\text{hence } V=\frac{62500}{2\times 23040}=1.36 \text{ volts.}$$

With a compound consisting of a basic element and an acid radical, the voltage may be calculated, (1) for decomposition into ultimate constituents, or (2) for the actual products obtained in electrolysis. Thus, (1) $Cu+S+O_4+aq.=197,500$ g. cal., $n=2$, $V=\frac{197500}{2\times 23040}=4.30$ volts. This calculation has, however, no practical interest, as $CuSO_4$ cannot be broken up into Cu, S and O_4 by electrolysis.

$$(2)\ CuSO_4+H_2O=Cu+H_2SO_4+O.$$
$$-197,500-69,000+210,000=-56,300 \text{ g. cal.}$$

$$\frac{56300}{2\times 23040}=1.22 \text{ volts.}$$

Table 167*a* by Richards[1] gives for one chemical equivalent the thermochemical constants for basic and acid elements, and the corresponding drops of energy in their decomposition; H has been selected as basic reference and has received the value zero.

In the formation of $CuCl_2$ the data in the table are $-7,900$ Cu, $+39,400$ Cl_2 $=31,500$ g. cal. for 1 chemical equivalent; the corresponding drop in voltage is -0.34 Cu, $+1.71$ $Cl_2=1.37$ volts, which agrees with the previous calculation.

The order in which the elements are placed gives also the order in which they will be deposited one after another by decreasing voltages. A second table by Richards in the paper quoted gives the data for decomposing compounds of basic elements and acid radicals into their ultimate constituents.

[1] *J. Frankl. Inst.*, 1906, CLXI, 131, 162.

TABLE 167a.—THERMOCHEMICAL CONSTANTS PER CHEMICAL EQUIVALENT WITH CORRESPONDING VOLTAGES[1]

Basic elements						Acid elements			
Element	Per chemical equivalent, g. cal.	Corresponding voltage	Element	Per chemical equivalent, gr. cal.	Corresponding voltage	Element	Per chemical equivalent, g. cal.	Corresponding voltage	Salt
$Li^{\cdot}$	+62,900	+2.73	$Cd^{\cdot\cdot}$	+ 9,000	+0.39	F''_2 (gas)	+52,900	+2.30	Fluoride.
$Rb^{\cdot}$	+62,000	+2.69	$Co^{\cdot\cdot}$	+ 8,200	+0.36	Cl''_2 (gas)	+39,400	+1.71	Chloride.
$K^{\cdot}$	+61,900	+2.69	$Ni^{\cdot\cdot}$	+ 7,700	+0.33	Br''_2 (gas)	+32,300	+1.40	Bromide.
$Ba^{\cdot\cdot}$	+59,950	+2.60	$Fe^{\cdot\cdot\cdot}$	+ 3,230	+0.14	Br' (liquid)	+28,600	+1.20	Bromide.
$Sr^{\cdot\cdot}$	+58,700	+2.55	$Sn^{\cdot\cdot}$	+ 1,900	+0.08	Br' (solid)	+27,300	+1.18	Bromide.
$Na^{\cdot}$	+57,200	+2.48	$Pb^{\cdot\cdot}$	+ 400	+0.02	I''_2 (gas)	+20,000	+0.87	Iodide.
$Ca^{\cdot\cdot}$	+54,400	+2.36	$H^{\cdot}$	0	0	I' (liquid)	+14,600	+0.63	Iodide.
$Mg^{\cdot}$	+54,300	+2.36	$Tl^{\cdot\cdot}$	− 900	−0.04	I' (solid)	+13,200	+0.57	Iodide.
$Al^{\cdot\cdot\cdot}$	+40,100	+1.74	$Cu^{\cdot\cdot}$	− 7,900	−0.34	S'' (solid)	− 5,100	−0.22	Sulphide.
$(N+H_4)^{\cdot}$	+33,400	+1.45	$Hg^{\cdot\cdot}$	−14,250	−0.62	Se'' (met.)	−17,900	−0.78	Selenide.
$Mn^{\cdot\cdot}$	+24,900	+1.08	$Pt^{\cdot\cdot\cdot\cdot}$	−19,450	−0.84	. . .	.	. . .	. . .
Zn.	+17,200	+0.75	$Ag^{\cdot}$	−25,200	−1.10	. .		. .	. . .
$Fe^{\cdot\cdot}$	+19,900	+0.47	$Au^{\cdot\cdot\cdot}$	−30,300	−1.32	. . . .	. . .	. .	. . . .

The calculation of potential according to the formulæ of Nernst from the electrolytic solution and osmotic pressures is so far not available for metallurgical processes, as the experimental data are missing. Some decomposition voltages which have been determined experimentally[2] are assembled in Table 168.

TABLE 168.—DECOMPOSITION VOLTAGES OF VARIOUS SOLUTIONS

ACIDS: H_2SO_4 1.67, HNO_3 1.69, H_3PO_4 1.70, HCl 1.31, $H_2C_2O_4$ 0.95, HBr 0.94, HI 0.52.
BASES: NaOH 1.69, KOH 1.67, NH_4OH 1.74.
SALTS: $ZnSO_4$ 2.35, $ZnBr_2$ 1.80, $NiSO_4$ 2.09, $NiCl_2$ 1.85, $Pb(NO_3)_2$ 1.52, $AgNO_3$ 0.70, $Cd(NO_3)_2$ 1.98, $CdSO_4$ 2.03, $CdCl_2$ 1.88, $CoSO_4$ 1.92, $CoCl_2$ 1.78.

7. CHARACTER OF DEPOSIT.—The character of the deposit depends to a certain extent on the current density. With values of the latter that are too large the deposit is generally spongy; if too small it often does not come down well. In practice there is no danger of erring on this side. Nodules at the corners and edges are due to the greater current density at these places than at others. From mixed solutions, alloys may be deposited by a current of sufficient e.m.f. to decompose the salt of the more electropositive metal as long as the e.m.f. of the two metals are not too far apart; thus brass is deposited from Zn–Cu cyanide solutions but not from sulphate solutions.

8. GROUPING OF CELLS OR VATS.—Electrolytic cells or vats are arranged in series or in parallel or in a combination of the two systems. The arrangement

[1] The normal number of equivalents of an element are designated by a dot (˙) representing a negative, and by a stroke (′) a positive bond.

[2] Standard Handbook for Electrical Engineers, McGraw Publishing Co., New York, 1908, p. 975.

of cells is governed by the voltage of the dynamo. With a given voltage enough vats must be connected in series for their combined resistances to furnish, according to Ohm's law, the desired current. Then as many vats as the dynamo can supply with the current are grouped in parallel.

In technical electrolytic work, both the multiple and series systems are in operation. In the former, which is the more common, the anodes and cathodes of a vat are coupled in parallel and the vats in series, and the number of vats is increased to fit the voltage of the dynamo; in the latter, the electrodes are placed in series, while a number of vats are connected in parallel.

9. INSOLUBLE AND SOLUBLE ANODES.—In electrolyzing a $CuSO_4$-solution using an insoluble anode, Cu is deposited on the cathode and O-bubbles are set free at the anode: $CuSO_4 = Cu + SO_4$ and $SO_4 + H_2O = H_2SO_4 + O$. As the solution becomes poorer in Cu and richer in acid, H-bubbles are set free at the cathode. Energy is wasted, because useless O and H are generated; the current is weakened by the increased resistance caused by the badly conducting films of gas which cover the electrodes, and because an opposing e.m.f. is set up by the H at the cathode. Insoluble anodes are therefore the exception in metallurgical electrolytic work. Siemens and Halske have utilized the anode energy obtained from electrolyzing with insoluble anodes a solution of mixed blue and green vitriol for oxidizing $FeSO_4$ to $Fe_2(SO_4)_3$, viz.: $2CuSO_4 + 4FeSO_4 = Cu_2 + 2Fe_23SO_4$; they thus regenerate their solvent for dissolving roasted copper-iron sulphides. Hoepfner in his process has electrolyzed one-half of a solution of Cu_2Cl_2 in brine and perchloridized the other half with the Cl_2 set free, viz.: $Cu_2Cl_2 = Cu_2 + Cl_2$ and $Cu_2Cl_2 + Cl_2 = 2CuCl_2$; he thus also regenerates his solvent for treating silver-bearing roasted copper-iron sulphides.

In electrolyzing a $CuSO_4$-solution using a soluble anode of Cu, Cu is again deposited on the cathode, but the anode energy (SO_4) is utilized to dissolve the anode Cu. Approximately the same amount of energy is consumed at the cathode by decomposition, $CuSO_4 \rightleftarrows Cu + SO_4$, as is generated at the anode by formation of $CuSO_4$. No bubbles of O are set free at the anode, and, as a lower e.m.f. is required than for the decomposition of H_2O, no H-bubbles are liberated at the cathode.

The anode in electrolyzing $CuSO_4$ need not be Cu; it can be a copper compound, *e. g.*, matte $5CuSo_4 + (2Cu_2S + 4FeS) = 9Cu + 2FeSO_4 + Fe_2(SO_4)_3 + 6S$, or $6CuSO_4 + (2Cu_2S + 4FeS) = 10Cu + 2Fe_2(SO_4)_3 + 6S$, but since more Cu is deposited on the cathode than dissolved at the anode, the electrolyte will have to be quickly circulated and brought to the normal percentage of Cu by being made to act chemically upon fresh matte, if gas bubbles are to be avoided.

10. PRIMARY AND SECONDARY REACTIONS.[1]—In electrolytic work a metal may be deposited directly by a primary, or indirectly by a secondary reaction. In a primary, the current directs the ions toward the electrodes and liberates them there as ions, *e.g.*, cathode ← Cu | SO_4 → anode. In a secondary reaction the ions react upon themselves, upon the electrodes, or the electrolyte, causing

[1] Richards, *J. Frankl. Inst.*, 1901, CLII, 201.
Stevenson, *Met. Chem. Eng.*, 1910, VIII, 240.

substances to form at the electrodes other than the ions, *e.g.*, primary: cathode $\leftarrow K \mid Ag(CN)_2 \rightarrow$ anode (Ag); at the cathode, secondary: $K+KAg(CN)_2=2K(CN)+Ag$ deposited; at the anode, secondary: $Ag(CN)_2+Ag=2Ag(CN)$; now $2K(CN)+2Ag(CN)=2KAg(CN)_2$ regenerates the original solution.

254. The Electrolytic Plant and its Management.—In order to obtain the data required for the erection of a new plant, one can follow precedent or carry on systematic investigations of a process.[1]

After the voltage required has been acertained, data must be found for the concentration and acidity, or alkalinity of the electrolyte, for the strength of current and distance of electrodes. In experiments, the cells used must not be too small (not <4 in. sq. for Cu), as otherwise the results will be unreliable when interpreted for a working scale. These main data having been ascertained the following problems[2] should be studied.

(1) Required output of metal per day; (2) required purity of anode and of electrolyte which governs; (3) rate of deposition (impure anodes require slower work than pure for same quality of cathode metal; with expensive power, slow work may be cheaper than quick); (4) size of electrodes and total cathode area (large electrodes are inconvenient to handle, small electrodes increase number of contacts); (5) vats, their size and number of electrodes per vat, their number and necessary floorspace, their material of construction (wood, lead-lined, tarred, rubber-painted; stone; glazed earthenware); their arrangement on level or terraced plane; (6) dynamo, kind, capacity, and e.m.f.; (7) power, kind and amount.

An electrolytic plant requires constant supervision in order to obtain a high efficiency. (1) The current must be kept uniform (constant measurement of amperage and periodical measurement of voltage). Thus short-circuiting, damage to dynamo, defective contacts, changes in electrode distances, badly conducting coatings on anodes, and inferior cathode deposits must be avoided.

(2) The electrolyte must be kept moving and frequently analyzed. A solution that is not stirred or circulated becomes poorer in metal at the cathode and richer at the anode, the heavier liquor sinks and forces up the lighter one (layering), and the current goes mainly through the heavier one, or through a smaller electrode area and changes values. The electrolyte must be tested for metal content and acidity, or alkalinity, and its temperature taken. An excess of H_2O increases the resistance, a lack of it causes salts to settle upon the anode; an excess of acid or alkali may attack the cathode, a lack of it may cause oxide to settle upon the anode; accumulation of impurity affects the cathode deposit; floating matter does the same.

(3) The anode mud must be removed at fixed intervals; as certain impurities in the mud, as shown above, are sure to go into solution if left even a short time in the vat, and as the mud, growing too thick, may cause short-circuiting or may be stirred up in exchanging electrodes.

[1] Oettel, F., and Smith, E. F., "Introduction into Electrochemical Experiment," and "Practical Exercises in Electrochemistry," Blakiston, Philadelphia, 1897.

[2] Gore, *op. cit.*, p. 224.

(4) The anodes must be replaced at stated intervals as well as the cathodes, and the anodes may have to be brushed off to remove coatings, etc.

255. Dry Electrolysis.[1]—The electrolysis of fused electrolytes has been practiced so far in metallurgical plants only in the reduction of Al_2O_3. Many processes for the treatment of other ores and for refining metals have been proposed. In all of them either the ore itself (carnallite, $KMgCl_3+6aq.$) forms the electrolyte, or the ore is held in igneous solution by a salt which requires a higher potential for decomposition than the ore (Al_2O_3 in $3NaF.2AlF_3$); such salts are mainly chlorides and fluorides.[2] In both cases the anode may be attacked by the electronegative component. As a rule, the resistance of the fused electrolyte to the passage of the current, considering the size of the vat, is sufficiently high to keep the bath fluid without having to resort to an extraneous source of heat. The temperatures attained rarely exceed 1,000° C. Part of the energy of the current is required for the chemical work; the rest $(E-e)$ is converted into heat to keep the bath fluid, thus $H=0.24.(E-e).I.t.$ The specific resistance of fused electrolytes is low, *e.g.*, that of $PbCl_2$ is 1/5 that of the minimum value of H_2SO_4. Hampe[3] classed as good conductors: $(Ba.Na_2.K_2)Cl_2.Br_2.I_2$; $(Sr.Ca.Mg.Zn.Cd)Cl_2.Br_2$; $(Zn.Cd.Pb)I_2$; $(Sn\ Cu_2)Cl_2$; $BiCl_3$; as inferior conductors: $HgCl_2$, $SbCl_3$; as non-conductors: $AlCl_3$, $AlBr_3$, $SnCl_4$, $AsCl_3$. An electrolyte should have the following properties:[4] low melting-point, stability at temperature above the melting-point, low vapor tension, good conductivity, fluidity and homogeneity, low dissolving power for cathodic metal, low voltage of decomposition, and its anion little corrosive effect upon anode or vessel.

The principles outlined for wet electrolysis hold good for dry electrolysis. It is assumed that all the molecules of a fused salt participate equally in the transport of electricity and not only the solute as is the case with wet electrolysis. One of the difficulties encountered in dry electrolysis is the formation of a mist of finely divided metal which permeates the electrolyte and disturbs the regular passage of the current; another, the formation at the anode of secondary products which coat it and stop the process. It is essential to keep the composition of the electrolyte constant as well as its temperature. Changes in composition cause variations in current density; a rise in temperature consumes power and causes solution of deposited metal and its possible oxidation at the anode. The anode, usually C, sometimes water-cooled metal, has to be adapted to the process. The vat, steel or cast-iron, lined with refractory argillaceous or carbonaceous material, does not last very long.

256. Source and Cost of Power.—Galvanic, storage and thermo-electric batteries are used only in laboratory experiment. The apparatus for generat-

[1] Lorenz, R., "Die Electrolyse Geschmolzener Salze," Knapp, Halle, 1905–06.

[2] Melting-points of mixtures: Kern-Auerbach, *School Min. Quart.*, 1907, XXIX, 77, 78.

[3] *Chem. Z.*, 1887, XI, 816; 1888, XII; *J. Chem. Soc.* (Engl.), 1888, XLIV, 211 (abstracts).

[4] Kern-Auerbach, *School Min. Quart.*, 1907, XXIX, 67.

Houben: Solubility of Met.O and Met.S in fused NaCl, *Metallurgie*, 1912, IX, 592.

ing electric currents to be used in metallurgical work is the dynamo-electric machine[1] which converts mechanical into electrical energy.

Alternating-current (A.C.) generators are usually employed for electro-thermic work. Direct-current (D.C.) generators have to be used for electrolytic work. Table 169 gives the characteristics of some of the dynamos used in metallurgical plants.

TABLE 169.—DYNAMOS OF SOME METALLURGICAL AND CHEMICAL PLANTS[2]

Generator	Poles	Kw.	Rev.	Amp.	Volt	Application
Single-phase	20	500	150	......	110	Electric steel smelting.
Single-phase	12	1,000	400	.....	45	Calcium carbide manufacturing.
Direct-current	10	162	410	....	54	Bleaching liquor manufacturing.
Direct-current	8	300	600	......	25–75	Electrolytic refining.
Direct-current	14	2,100	225	......	700	Aluminum smelting.
Direct-current	18	2,200	140	....	500	Aluminum smelting.
Direct-current	18	3,540	300	....	650	Aluminum smelting.
Direct-current	14	1,500	214	....	250	Copper refining.
Direct-current	.....	1,050	100	10,000	.....	Copper refining.
Direct-current	....	300	150	2,400	.....	Copper refining.
Direct-current	......	520	110	4,500	.....	Copper refining.
Direct-current	......	510	125	1,700	......	Copper refining.

The cost of an electro-metallurgical operation depends primarily upon the cost of current, and this upon the cost of mechanical energy and the loss in converting mechanical into electrical energy.

The cost per h.p.-year varies with the source of power (steam, water, gas, oil, engine), the size of plant (large cheaper than small), the manner of application (working full- or half-time; full-load, partly loaded). Emery[3] estimates for ordinary electric-lighting stations, running 10 hr. a day with steam-power and coal at $3 per long ton, the cost per 1 h.p.-hr. is 1 cent or the h.p.-year $30.80. Water-power under favorable conditions has been developed in this country at $8 to $12 per h.p.-year. High rentals and other unfavorable conditions have

[1] Franklin, W. S., and Esty, Wm., "Direct-current Machines, Electric Distribution and Lighting," Macmillan, New York, 1906; Alternating Currents, 1911.

Sheldon, S., and Mason, H., "Dynamo-electric Machinery: Its Construction, Design and Operation," Van Nostrand, New York; Direct-current Machines, 1909; Alternating-current Machines, 1904.

Thompson, S. P., "Dynamo-electric Machinery," Spon, London-New York; Continuous-current Machines, 1904; Alternating-current Machinery, 1905.

[2] See also: General Electric Co., Schenectady, N. Y., *Bull.* 4860, Sept., 1911.

[3] *J. Frankl. Inst.*, 1896, XLII, 165 to 186.

Doherty, *Am. Inst. Elect. Eng.*, Dec. 16, 1909; *Met. Chem. Eng.*, 1910, VIII, 116.

Crabtree, *Met. Chem. Eng.*, 1910, VIII, 337.

placed it as high as steam-power. Emery believes (1) that a very large water-power may be developed and the power transmitted locally with advantage in comparison with steam-power even where fuel is cheap, if the costs of construction are kept within reasonable limits; (2) that transmission of power 20 miles[1] from waterfall is practicable even in competition with cheap fuel, if the power is utilized 24 hr. per day or in large units, or delivered in high tension on the premises of the manufactory and distributed under conditions less exacting than those which obtain in large cities; (3) that when fuel is $3 a ton or over, electric transmission is generally practicable.

In converting mechanical into electrical energy, it is advisable to assume a loss of 10 per cent. giving a gross efficiency of 90 per cent. Of the energy delivered by the dynamo it can be assumed that only 75 per cent. does actual work (net or commercial efficiency), the remaining 25 per cent. being lost by heating of conductors and by leakage.

[1] In California a hydroelectric power station transmits power 250 miles at 100,000 volts. In the East a number of stations use between 30,000 and 60,000 volts; the common ratios for transformers are 10:1, 100:1, 1,000:1

CHAPTER XI

MECHANICAL METALLURGICAL OPERATIONS

257. Introduction.—Metallurgical processes necessitate mechanical operations which have to deal with solid, liquid or gaseous materials. Thus coarse ores may have to be crushed, fine ores and dust concentrated and briquetted; metals and alloys are frequently fashioned by rolling, forging, pressing and drawing for use in the arts; solutions have to be handled in wet processes, and solids separated from liquids by filter-pressing or other mechanical means; smelting operations require forced draft or blast, and the air may have to be heated or cooled before it is made to act upon fuel or charge; the vapors and gaseous products of a process have to be withdrawn from the apparatus in which they are produced, and perhaps some of the values recovered as flue dust before the gases are allowed to pass off into the open. These and similar operations, common to many metallurgical processes, are passed in review in this chapter.

A. ORES

258. Crushing Ores in General.[1]—Ores, fluxes, fuels and some intermediary metallurgical products, such as speise and matte, which are too coarse for a process have to be reduced to suitable sizes.

Crushing machinery is conveniently classed according to the size of the product as coarse-crushing, furnishing material 2 1/2 to 3/4 in. diam.; medium-size crushing, 3/4 in. to 30- and 40-mesh, and fine-crushing 30- and 40-mesh and smaller. The sizes from 1/4 in. (or 1/2 in.) upward are designated by the opening in the screen measured in inches; sizes smaller than 1/4 in. are designated by the number of meshes to the linear inch. Such a screen, having four holes to the linear inch, will be a 4-mesh screen and will have 16 holes to the square inch. This approximation becomes an accurate definition only if the diameter of wire is added, but this addition has become customary only in recent years.

The simplest way to designate a screen size is to give the size of the opening in millimeters. To this should be added (1) the percentage of area of holes in a given surface to show the freedom of discharge and strength of screen, and (2) the thickness of wire or metal.[2]

Table 170 of the W. S. Tyler Co., Cleveland, O., for mining wire-cloth gives the customary number of meshes per linear inch and adds the sizes of iron and

[1] Richards, R. H., "Ore Dressing," McGraw-Hill Book Co., New York, 4 volumes, 1903–1909.

Richards, R. H., "A Text-book of Ore-dressing," McGraw-Hill Book Co., New York, 1909.
Louis, H., "The Dressing of Minerals" Longmans, Green & Co., London, 1909.
Naske, C., "Zerkleinerungsvorrichtungen und Mahlanlagen," Spamer, Leipsic, 1911.

[2] See Definitions of Institute of Mining and Metallurgy, *Tr. I. M. M.*, 1909–1910, XIX, 486 (Hoover), 1911–12, XXI, 738; and Standard Screens (Richards), *Met. Chem. Eng.*, 1913, XI, 168; *Min. and Sc. Press*, 1913, CVI, 312.

steel wire in the gauge of Washburn and Moen, and of copper and brass wire in the old English gauge. It thus defines and shows the actual sizes of holes in the regular screens made by the firm. The openings of screens made by other manufacturers will deviate slightly from the values given if they use a wire of different size.

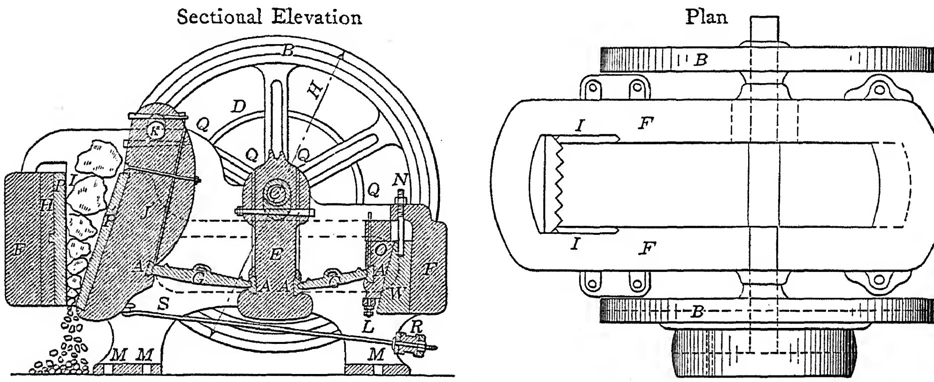
The designation by "needles" for plate in stamp-mill screens is given in §263.

TABLE 170.—W. S. TYLER CO.'S WIRE-CLOTH

Iron or steel				Brass or copper			
No. of meshes per linear inch	Washburn and Moen gauge	Size of opening		No. of meshes per linear inch	Old English gauge	Size of opening	
		Inch	Millimeter			Inch	Millimeter
2	8	0.338	8.59	2	8	0.335	8.51
3	10	0.198	5.03	3	10	0.199	5.05
4	11	0.130	3.30	4	11	0.130	3.30
4	12	0.145	3.68	4	12	0.141	3.59
5	13	0.108	2.74	5	13	0.105	2.67
6	14	0.087	2.21	6	14	0.082	2.08
7	15	0.071	1.80	7	15	0.071	1.80
8	16	0.062	1.57	8	16	0.060	1.52
9	17	0.057	1.45	9	17	0.053	1.35
10	18	0.053	1.35	10	17	0.042	1.07
12	19	0.042	1.07	10	18	0.051	1.30
14	20	0.036	0.91	12	18	0.034	0.86
16	22	0.034	0.86	12	19	0.043	1.09
18	23	0.0305	0.77	14	19	0.031	0.79
20	24	0.027	0.69	14	20	0.036	0.91
22	24	0.022	0.56	16	20	0.0275	0.70
22	25	0.025	0.64	16	21	0.031	0.79
24	25	0.021	0.53	18	22	0.026	0.66
24	26	0.023	0.58	20	23	0.023	0.58
26	26	0.0205	0.52	22	24	0.0204	0.52
26	27	0.0215	0.55	24	25	0.0186	0.47
28	27	0.0187	0.47	26	26	0.0179	0.45
30	28	0.0173	0.44	28	27	0.0169	0.43
35	30	0.0145	0.37	30	27	0.0185	0.37
40	31	0.0115	0.29	35	28	0.0120	0.30
45	33	0.0112	0.28	35	29	0.0131	0.33
50	34	0.0100	0.25	40	30	0.0112	0.28
55	35	0.0086	0.22	45	31	0.0100	0.25
60	35	0.0072	0.18	50	32	0.0083	0.21
70	37	0.0058	0.15	60	35	0.0077	0.20
80	40	0.0053	0.14	70	37	0.0078	0.20
90	42	0.0044	0.11	80	38	0.0068	0.17
......	......	......	......	90	39	0.0061	0.15

259. Coarse-crushing Breakers.—This work is usually accomplished by machines which crush by the pressure of approaching jaws. The largest size lump fed to them is about 24 in. in diam., the usual size not over 9 in. The intermittent jaw-breakers of Blake and Dodge, and the continuous spindle or gyrating breaker of Gates are the leading types.

The Blake eccentric breaker is represented by Figs. 326 and 327 in vertical longitudinal section and plan. A heavy cast-iron or cast-steel frame *F* resting on four legs *M* with bolt-holes carries swing-jaw shaft *K* and eccentric shaft *C*. At the rear, a horizontal web forming part of frame *F* supports wedge-block *W* by means of adjusting stud *N* and toggle block *O* with its adjusting bolt to regulate the width of the throat or discharge end; at the front, the fixed jaw *H*



LEGEND

AA'	Bearings Toggle	*H*	Fixed Jaw	*O*	Toggle Block
B	Flywheel	*I*	Cheek Plates	*PP'*	Jaw Plates
C	Eccentric Shaft	*J*	Swing Jaw	*R*	Rubber Spring
D	Driving Pulley	*K*	Swing Jaw Shaft	*S*	Rod
E	Pitman	*L*	Set Screws for Toggle Block	*W*	Wedge Block
F	Frame	*M*	Legs with Bolt Holes		
G	Toggle	*N*	Wedge Adjusting Stud		

Figs. 326 and 327.—Blake eccentric breaker.

bedded with 1/4 in. of zinc against frame *F* carries jaw-plate *P*, usually of corrugated chilled cast iron and held in place by check-plates *I*. The swing jaw *J*, pivoted above from jaw shaft *K*, carries at the front jaw plate *P'*, and at the back toggle bearing *A* and eye for spring-rod *S*. Steel pitman *E*, lifted by the eccentric on shaft *C*, straightens out toggles *G* which force swing jaw *J* to approach fixed jaw *H*, and between the two the rock fed at the mouth is crushed. When the pitman is lowered, the toggles are unlined, and the swing jaw is made to recede by rod *S* and rubber spring *R*, which opens the throat for the discharge of crushed rock. Details of some of the common sizes of this crusher are given in Table 171.

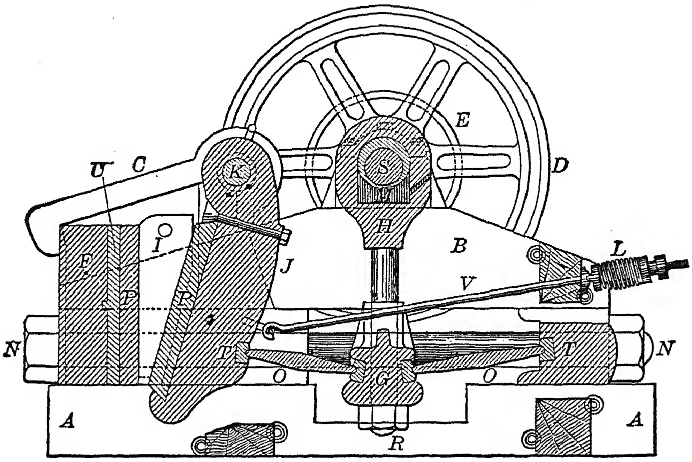
The Blake challenge rock breaker, a modification of the Eccentric, is shown in Fig. 328; it is lighter and stronger than the original. The heavy cast-iron frame *F* of Figs. 326 and 327 has been replaced by timber frames *A* and *B*, the tensile strain is taken up by wrought-iron bolts *N*, and the upper ends of the jaws are held together by the steel straps *C*.

TABLE 171.—SIZES OF BLAKE EXCENTRIC BREAKERS[1]

No.	Mouth, size, inches	Capacity, tons rock in 24 hr. to sizes stated			Outside dimensions			Pulley size, inches	Rev. per minute	Horse-power required	Total weight, pounds	Weight of heaviest piece
		Tons. In.	Tons. In.	Tons. In.	Length, Ft. In.	Breadth, Ft. In.	Height, Ft. In.					
2	6×2	24—2	10—1	3½—½	3—2	2—6	2—3	11×4	275	2	1,200	560
3	10×4	84—2	60—1½	36—1	4—9	3—8	3—11	20×6½	275	5	4,660	2,700
4	10×7	120—2	96—1½	60—1	6—0	4—2	4—5	30×7½	275	8	8,200	4,450
6½	15×9	240—2½	190—2	130—1½	7—0	5—3	5—3	30×9	275	12	15,000	7,500
8	20×10	420—3	360—2½	300—2	7—8	5—10	5—1	30×12	275	20	18,300	10,300
9	24×15	600—3	480—2½	420—2	7—10	7—2	5—9	30×12	275	25	26,000	12,000
10	30×15	960—4½	840—3½	....	8—6	8—2	6—4	30×12	275	30	37,600	22,000

[1] From Farrel Foundry & Machine Co., Ansonia, Conn.

The Dodge Breaker, Fig. 329.—A cast-iron frame, 2, carries at one end the box of fulcrum pin 4, at the other that of eccentric shaft 8; fixed jaw-plate 3 is bolted to frame 2, movable jaw-plate 3′ to jaw 1′ which is the short arm of lever 1 oscillating on fulcrum pin 4; the lost motion of the latter is taken up by a spring not shown. The crushing movement is imparted to arm 1″ by eccentric



A. Lower timber frame.
B. Upper " "
C. Clamps.
D. Fly-wheels.
E. Pulley.
F. Fixed jaw block.
G. Pitman toggle block.
H. Pitman half-box.
I. Cheeks.
J. Swing jaw.
K. Jaw shaft.
L. Spring.
M. Oil chamber.
N. Main tension bolts.
O. Toggles.
P. Jaw plates.
R. Pitman rod nuts.
S. Main eccentric shaft.
T. " toggle block.
U. Fixed jaw back.
V. Spring rod.

Fig. 328.—Blake challenge rock breaker.

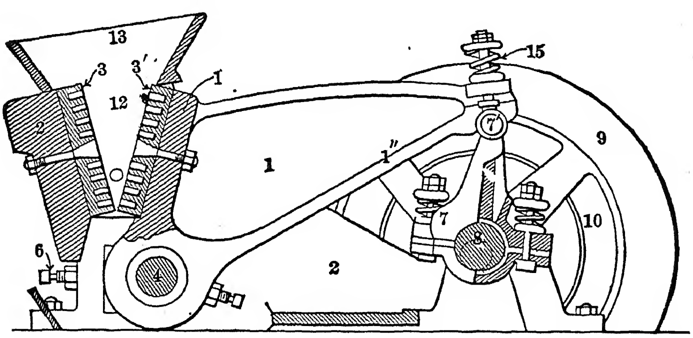


Fig. 329.—1, lever with short arm 1′, and long arm 1″; 2, cast-iron frame; 3, fixed jaw-plate 3′, movable jaw-plate; 4, fulcrum pin; 6, set-screw; 7, connecting rod; 7′, projecting pin; 8, eccentric shaft; 9, flywheel; 10, pulley; 12, throat; 13, hopper; 15, spring.

8, through connecting-rod 7 and pin 7′ which in their turn are joined to the arm by connections and springs 15. The width of the throat is adjusted by plates (not shown) held by set-screw and lock-nut 6. The power, applied by pulley 10, is consumed for little less than half a revolution and absorbed in fly-wheel for the remainder.

TABLE 172.—SIZES OF DODGE CRUSHER[1]

No.	Capacity, tons rock in 24 hr. to nut size	Pulley-size, inches	Rev. per minute	H.P. required	Total weight, pounds	Mouth size, inches
1	½–1	20×4	275	2–4	1,200	4×6
2	1–3	24×5	235	4–8	4,300	7×9
3	2–5	30×6	220	8–12	5,600	8×12
4	5–8	36×8	200	12–18	12,000	10×16

Comparing the Blake and Dodge types, it will be seen that with the former the swinging jaw is pivoted above and has the greatest motion on the smallest lump, while with the latter the jaw is pivoted below and has the greatest motion on the largest lump. Hence it is commonly held that the Blake type has the largest capacity, and the Dodge the evener product, but it is also known that the Dodge breaker makes more fines than the Blake, and that it shows a tendency to become choked.

THE GATES CRUSHER, STYLE K, Fig. 330.—The crushing mechanism consists of a vertical spindle 25, of wrought iron or forged steel with chilled-iron crushing head 18, usually corrugated, gyrating in a top shell 3, lined with smooth chilled-iron dies or concaves 19. The ore, fed through hopper 7, is crushed between plates 18 and 19 as the head 18 approaches or recedes from concaves 19, and discharged over wearing plate 22 and chute 32. The spindle 25, a lever having its fulcrum in spider 6, has a bushing 45 fastened to it by a jib-key and is suspended by this from a second bushing 44 fastened to the spider; the spindle can be raised and lowered as much as 7 in. by removing the dust cap 17, loosening a split-nut (not shown) on the shaft and screwing it up or down. The lower end of the spindle is journaled in the eccentric hub 8 of bevel gear 9, which is driven by the bevel pinion 11 on shaft 31 through break-pin hub 13 and break-pin 14, and the shaft rotated by pulley 12. Oil bonnet 15, with oil-hole and dust-ring 16, make dust-tight joints.

TABLE 173.—COMMON SIZES OF GATES CRUSHER, STYLE K

No.	Size of each of the two receiving openings, inches	Capacity, tons rock per hour to size of ring in inches								Pulley size, inches	Rev. per min.	Horse-power required	Total weight, pounds
		1½	1¾	2	2½	3	3½	4	5				
4	8×30	15	20	25	30	40	.....	....	....	32×12	400	14–21	20,900
5	10×38	...	30	40	50	60	70	.....	. .	36×14	375	22–30	31,200
6	12×44	....	....	50	70	80	90	....	...	40×16	350	28–45	45,500
7½	14×52	....	....	....	80	90	100	120	. .	44×18	350	50–75	64,800
8	18×68	....	....	....	....	....	130	150	....	48×20	350	70–110	100,000
9	21×76	....	. .	....	....	....	.....	250	300	56×20	300	100–150	153,000

[1] Fraser and Chalmers, Catalogue No. 4.

Gates crushers[1] are used generally in large sizes and for a large production a whole car of ore being dumped into the hopper; for small capacities they are expensive. The large pieces of rock are received where the motion is smallest

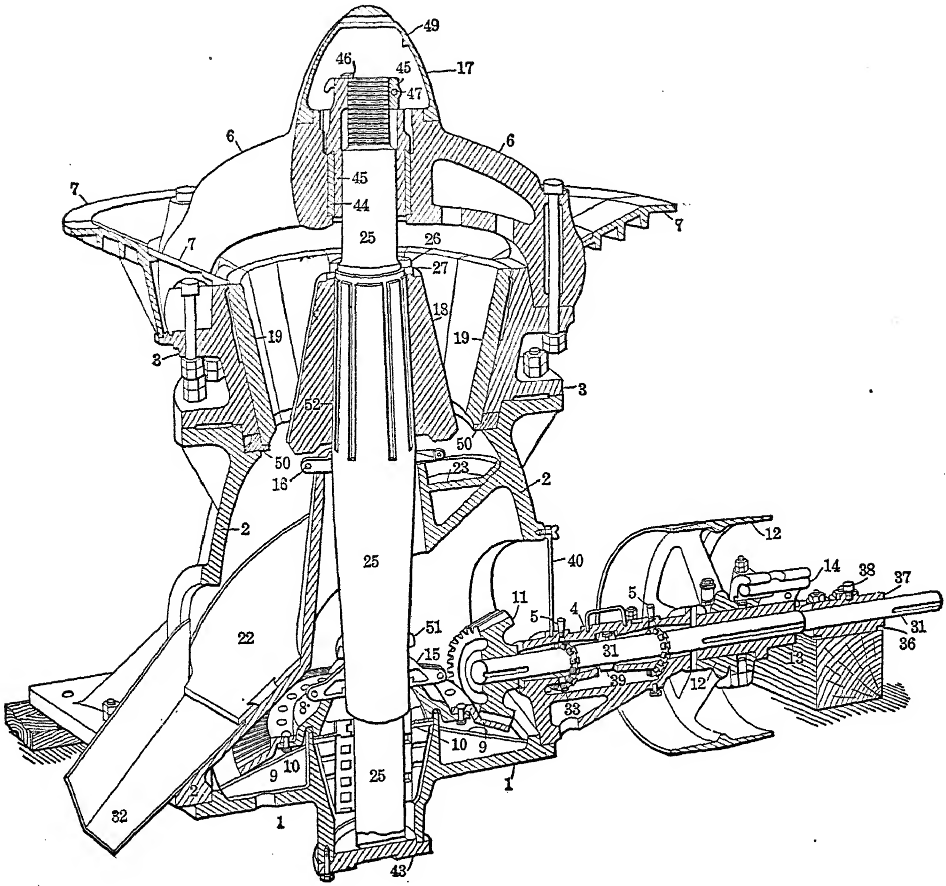


FIG. 330.—1, Bottom plate; 2, bottom shell; 3, top shell; 4, bearing cap; 5, oil cellar cap; 6, spider; 7, hopper; 8, eccentric; 9, bevel wheel; 10, wearing ring; 11, bevel pinion; 12, pulley; 13, break-pin hub; 14, break-pin; 15, oil bonnet; 16, dust ring; 17, dust cap; 18, head; 19, concaves; 22, chilled wearing plates; 25, spindle; 26, upper ring nut; 27, lower ring nut; 31, counter shaft; 32, chute; 33, oiling chain; 36, base of box; 37, cap of box; 38, oil cup; 39, collar; 40, housing; 43, bottom plate; 44, bushing attached to spider hub; 45, bushing keyed to spindle 25 and resting on 44; 46, gib-key for 45; 47, screw to tighten split-nut; 49, plugged oil-hole; 50, ring support for liners; 51, steel collar over oil bonnet to deflect dust; 52, rib for crusher head.

and the leverage greatest, hence the power required is relatively small. While the movement at the throat is greatest, the total amount is so small that the product is pretty uniform in size.

260. Medium-size Crushing.—DRY-CRUSHING ROLLS.—Raw material that

[1] Wollenberg, Comparison of Jaw and Gyratory Crushers, *Eng. Min. J.*, 1910, xc, 509.

has been crushed in one or more crushers and has to be reduced to medium size, say 1 1/2- to 2-in. cubes, is usually passed through dry-crushing rolls, as for example, in sampling ores or in preparing them for roasting, leaching by percolation, etc.[1]

In rolls, the two iron cylinders *A*, Fig. 331, with parallel axes *B* in one horizontal plane revolve in opposite directions, act radially upon a piece of ore *C*, and, drawing it in toward the narrowest space, break it by compression. Rolls

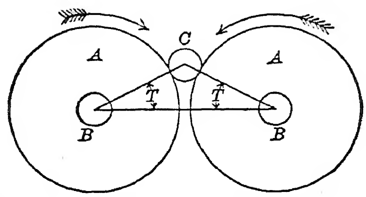


FIG. 331.—Principle of crushing-rolls.

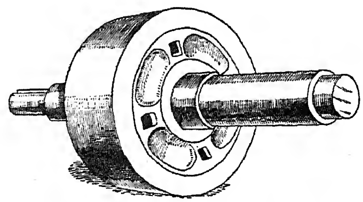


FIG. 331*a*.—Shaft, core and shell of roll.

make a small percentage of fines, as any particle smaller than the distance at which the rolls are set can drop between them without being crushed. This makes them valuable for most metallurgical work.

A roll, Fig. 331*a*, consists of a shaft of mild steel upon which is permanently mounted a core of soft cast-iron; the latter is enclosed by an exchangeable shell of hard iron (deeply chilled cast-iron or rolled carbon-steel) which forms the crushing surface. The two shafts, Fig. 332, of a pair of rolls are held in position

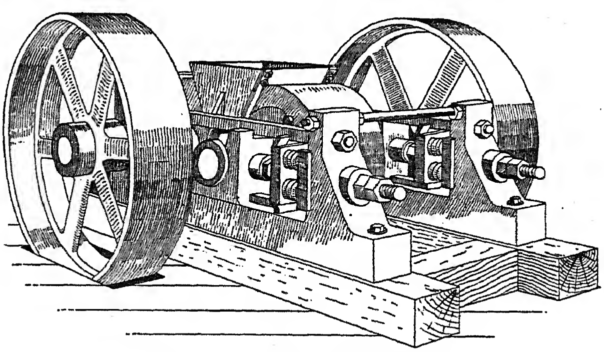


FIG. 332.—Belt-driven rolls.

by their journals and these are set in a heavy cast-iron frame. In the common rolls, one has a fixed box, Fig. 332, forming part of the frame, and the other a movable one; the latter is held up against the former by rubber or by steel springs; the distance between the two is regulated by shimes inserted between the boxes. In rigid rolls, Fig. 333, the boxes are held firmly in place, and the

[1] Ball-, Huntington-, Chilean-mills, etc., which crush as coarse as 20-mesh might do the work of rolls; usually, however, they are employed for crushing finer than 30-mesh and are therefore treated under fine-crushing.

shafts are protected against breaking by shearing bolts. Rigid rolls are simple in construction; as their action is positive, they make little oversize and thus save much of the necessary re-treating of ore so usual with common rolls.

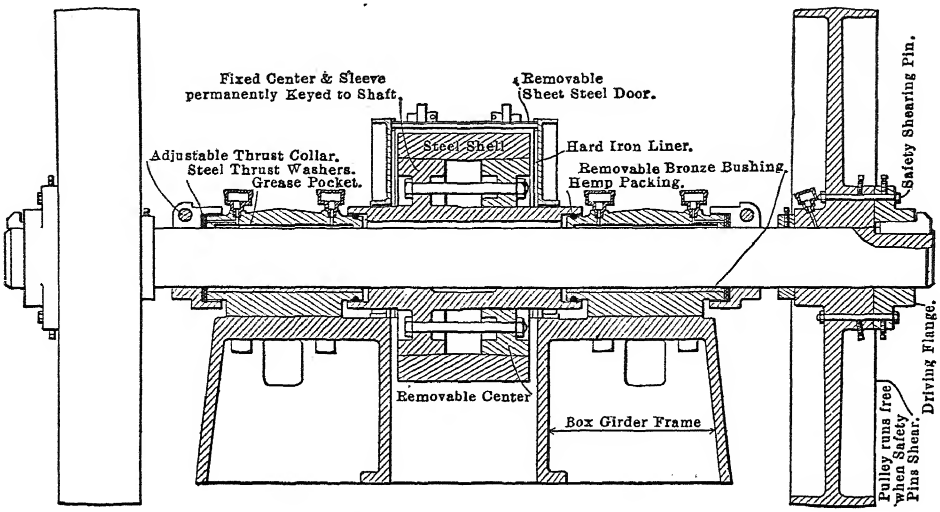


FIG. 333.—Rigid rolls.

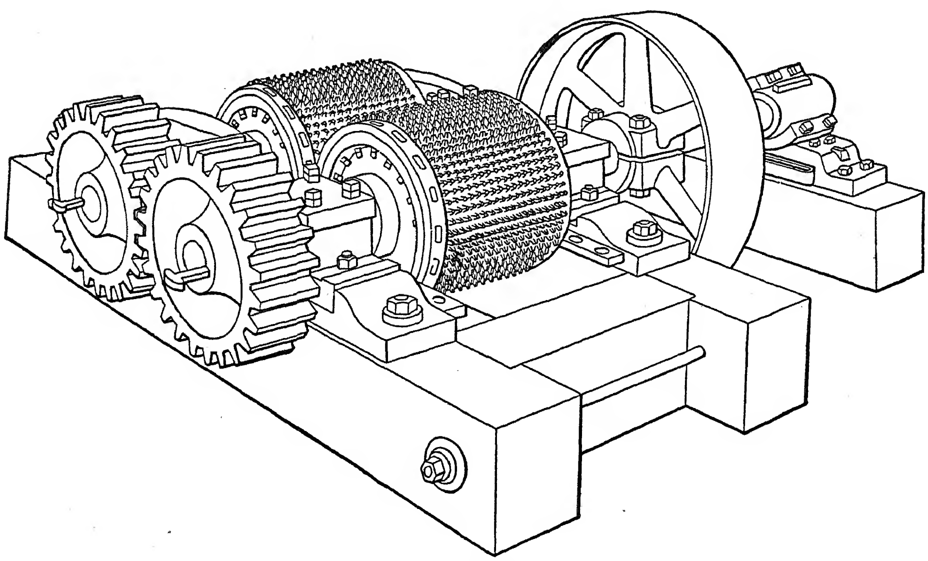


FIG. 334.—Toothed rolls.

The shafts of rolls are driven, either by gears and pulleys in "geared rolls," Fig. 334, suited mainly for slow-speed coarse-crushing with less than 50 r.p.m., or by belts alone in "belted rolls," Fig. 332, suited mainly for high-speed fine-crushing with over 50 r.p.m. In dry-crushing geared rolls are little used at

present excepting in fluted and toothed (Fig. 334) rolls common for crushing coal[1] and clay.

The width and diameter of rolls stand in certain relations to one another. In general, a wide roll, characteristic of the so-called Cornish rolls, has good capacity, but also causes a great stress on the frame; hence such rolls have a small diameter and are run slowly. They are better suited for producing coarse than fine material. They are difficult to keep true. Rolls with a large diameter and small width, if run at a high speed, also offer a large crushing surface. As their stress on the frame, with the corresponding loss of power

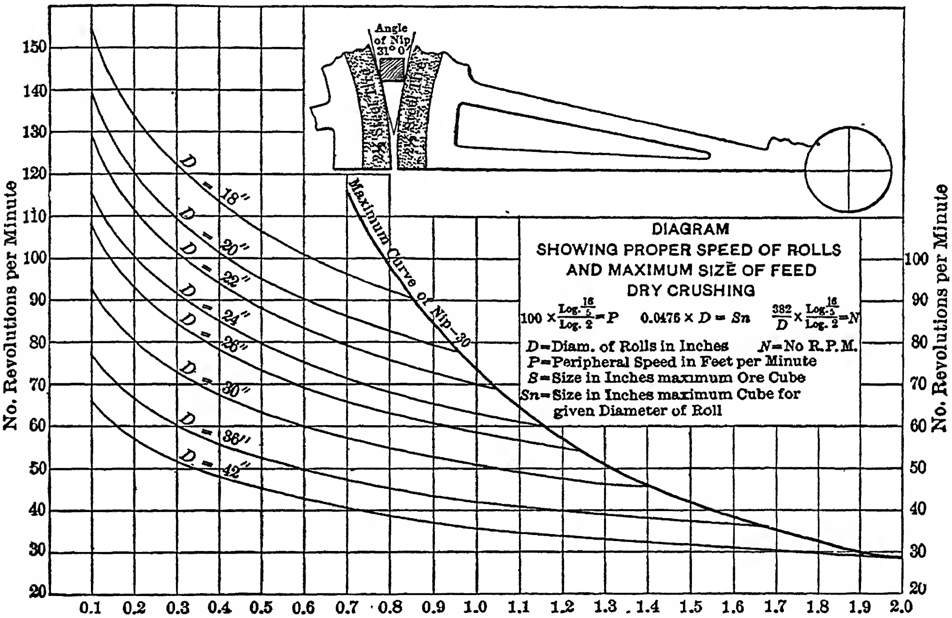


FIG. 335.—Feed, speed and capacity of rolls in dry-crushing.

in friction, is small, they can be and are run at a high speed. Further, the larger the diameter the smaller the angle of nip, Fig. 335 (the angle, formed by tangents to rolls drawn on points of contact of piece with rolls), or the better the bite of the rolls and the less the dancing of ore on the roll surface. A large-diameter roll can bite a larger piece of rock (work coarser material) than one of small diameter; it also makes a greater reduction in size with one passage, hence this class of rolls is used for finishing rather than for roughing.

Argall[2] states that the field of work for dry rolls begins with sizes of 1 1/2- and 2-in. cube, and ends with 30- to 40-mesh pulp; also, that for a single reduction the ratio ought not to exceed 4 : 1, *e.g.*, 2-in. : 1/2-in., 1/2-in. : 1/8-in. Fig. 335 represents his speed-curves for different diameter rolls in relation to the

[1] Sterling, *Tr. A. I. M. E.*, 1911, XLII, 284.
[2] *Trans. Inst. Min. and Met.*, 1902, X, 234.

size of ore fed for maximum capacity, the curves being terminated at the right by the nip-curve. The data for the curves were calculated from empirical formulæ based upon observed facts and upon the assumed reduction ratio of 4 : 1.

To show the application of the curves: The speed-curve for a 42-in. roll is intersected by the curve of nip on the 2-in. cube line at the point of 28 r.p.m.;

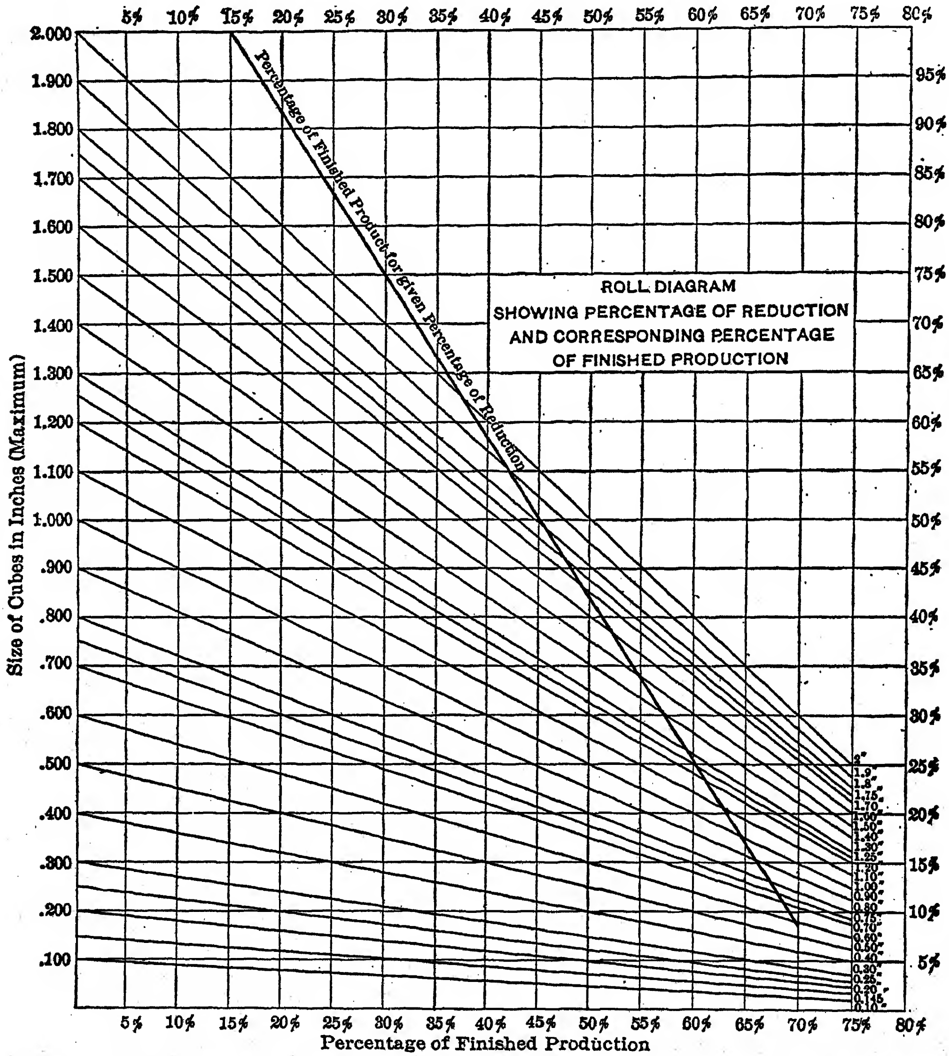


FIG. 336.—Percentage of reduction and corresponding percentage of finished product.

hence in order to crush 2-in. cubes (to 1/2-in. cubes) a 42-in. roll must make 28 r.p.m. In the same way with $D=26$ in., the maximum feed-size is 1.25 in. and the necessary speed 55 r.p.m., while for 0.50-in. cubes it is 73 r.p.m., for 0.25-in. cubes 88, etc.

The capacity of rolls may be said to be the number of cu. ft. per hr. (say of putty) that can be rolled out to a ribbon; its length would be the peripheral travel of the roll per hr.; its width that of the roll, its thickness the distance between the rolls. The capacity, $C=\frac{P\times W\times S\times 60}{1728}$, in which P = peripheral speed in inches per min., W = width of roll face in inches, S = space between rolls in inches. The ribbon of crushed ore, of course, is not homogeneous, as there are unfilled spaces in it which vary with the character of the ore. There is, however, a close relation between the percentage of reduction (1 cu. in. reduced to 3/4 in. = 25 per cent., to 1/2 in. = 50 per cent., to 1/4 in. = 75 per cent.) and the amount of finished product for any given ore. This is shown in Argall's

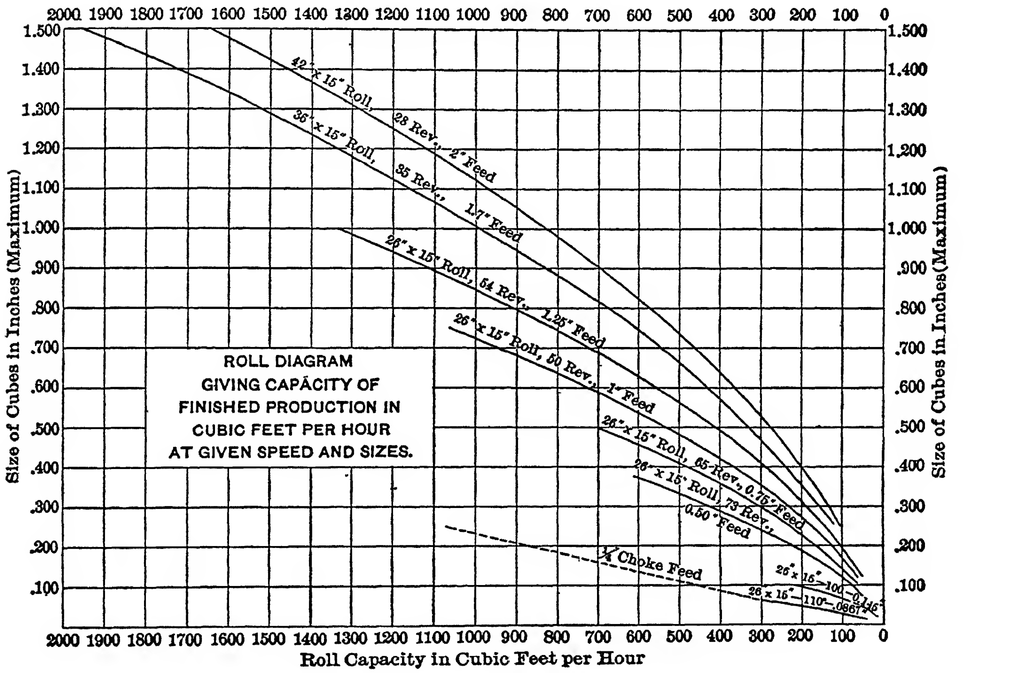


FIG. 337.—Capacity of rolls in cubic feet per hour.

second curve, Fig. 336, which is based upon results obtained in crushing quartzose ore.

The theoretical capacity of a 26×15-in. roll making 60 r.p.m., crushing ore of 1 in. to 1/4 in., is according to the above formula 589 cu. ft. per hr., assuming the mean diameter of the roll to be 24 in. Following in the diagram the line of 1000 in. cube from the left diagonally to its intersection with the line of 0.250 in. cube, this intersection is found on the ordinate marked 75 per cent. on the abscissa; taking this figure 75 to the right-hand scale and following its horizontal intersection with the line of finished product, this is seen to lie at 30 per cent., and shows that with 1-in. material there is obtained in a 75-per cent.

reduction 30 per cent. of the theoretical capacity as finished product, or 589 cu. ft.$\times$0.3=176 cu. ft.

The third diagram, Fig. 337, computed from Figs. 335 and 336, gives the capacity in cu. ft. per hr. of various sized rolls running at speeds most suitable for the size of the feed they are assumed to receive. Rolls, of course, require regular, that is, mechanical, feeding if they are to do a large amount of work.

An example of common belt-driven rolls is given in Fig. 332 representing style C of the Allis-Chalmers Co. The perspective view shows the timber base, the cast-iron main frame, the rolls with cast-iron core and steel shell, the movable frame of one roll held against the fixed frame of the other by springs, the pulleys of equal diameter, and the feed-hopper; the mechanical feeder and housing are omitted.

TABLE 174.—SOME COMMON BELT-DRIVEN ROLLS

Roll, diameter × face, inches	Driving-pulley, diameter × face, inches	Rev. per minute	Total weight, pounds	Horse-power
9×9	20×6½	175	1,300	3
12×12	36×8½	150	3,000	3
24×14	48×10½	100	9,200	6
30×16	48×10½	100	10,000	7
36×18	60×12½	75	19,300	7

2. RIGID ROLLS.—Fig. 333 represents, with the feed-hopper omitted, a longitudinal section through the shaft of a pair of rigid rolls made by the Denver Engineering Co. The cast-iron frame has the form of a box-girder; the bearings of the soft-steel shaft are made dust-proof by adjustable thrust-collars and hemp packing; the rolls are made up of three parts, the fixed center forced by means of an hydraulic press on to the shaft, the removable center drawn to the fixed by bolts, and the shell—the center and inner side of the shell having cone surfaces. The distance between rolls is made fixed. In order to provide for accidents, the end of the driving-shaft has keyed to it a driving flange; through it and into the driving pulley passes a safety shearing-pin which otherwise would

TABLE 175.—DENVER ENGINEERING WORKS RIGID ROLLS

Roll, diameter × face, inches	Driving-pulley, diameter × face, inches	Rev. per minute	Horse-power	Belt width, inches
20×12	40×7	160	12	6
27×14	54×9	120	16	8
36×16	72×11	80	25	10

run loose upon the shaft. In case a piece of iron comes between the rolls, the shearing-pin breaks and releases the pulley.

3. PLACING OF SMOOTH ROLLS.—In order to crush to uniform size with smooth rolls, it is usually necessary to have two pairs in series. In some instances a screen is placed between the two to remove the finished product, in others the first pair delivers straight into the hopper of the second.

4. CORRUGATED AND TOOTHED ROLLS.—These rolls are slow-moving and gear-driven. They serve for crushing soft material such as coal and shale. Corrugated rolls were first used by Coxe[1] for breaking anthracite. They have not come into general use with coal, as the continuous edge of a corrugation breaking along its entire length makes much fine coal. In clay-work this is no disadvantage.

Fig. 334 shows a pair of 21 1/2 by 21-in. toothed rolls[2] with housing removed. These make less fines than corrugated rolls, as the tooth striking a piece of coal splits it into several pieces, the cracks radiating from the point of attack. The teeth are usually inserted individually so that they can be exchanged easily. When teeth are cast on segments that are bolted to a roll, the whole segment has to be replaced in case of accident. The teeth, set so as to form a spiral, were formerly of cast-iron; at present they are of ferromanganese. Coarse rolls, so-called cracking rolls, range from 24-in. diam. by 48-in. face to 42-in. diam. by 52-in. face. One roll is usually driven faster than the other (125 and 95 r.p.m.), the slower one acting as feeder to the faster. The height of the teeth ranges from 3 to 1 in. Coarse rolls reduce lump coal, 12 to 16 in., down to 2 1/2 to 3 in. With fine rolls, the height of a tooth is rarely greater than 3/8 in.; they crush the delivery of the coarse rolls down to finer sizes. Table 176, of Allis-Chalmers Co., gives some of the leading facts in regard to toothed rolls.

TABLE 176.—ALLIS-CHALMERS TOOTHED ROLLS

Roll	Diameter, inches	15	24	27	36	40	42
	Face, inches.....	20	18	36	36	42	50
Capacity, tons per hour.		10–20	30–50	50–90	100–200	175–225	250
Horse-power required....		5–10	15–20	20–25	30–35	40–45	50

261. Fine-crushing in General.—Fine-crushing machinery finds its main metallurgical application in the amalgamation and lixiviation of gold and silver ores; here and there it is employed for comminuting other ores or metallurgical products. The leading apparatus are the stamp-, roller- and ball-mills, the arrastra and the pan, and the tube-mill. Next in importance are the machines

[1] *Tr. A. I. M. E.*, 1890–91, XIX, 414.

[2] Sterling, *Tr. A. I. M. E.*, 1911, XLII, 284.

for crushing coal and clay, such as disintegrators, hammer-crushers, etc. Only the leading mechanical features are treated here, with some working results to indicate the capacities.

262. Stamps in General.—The crushing effect of a stamp is that of a blow struck upon rock resting in a mortar. For effective work it is necessary to supply the mortar at regular intervals with rock of a suitable size, and to free it at a uniform rate from the fine-crushed material. The rock is introduced at the back by mechanical feeders; the fine material is discharged through a screen which closes the front, either as a wet pulp or a dry pulp depending upon whether water is fed into the mortar (wet-stamping) or not (dry-stamping). A better basis for classification of stamps is the manner of applying the power to strike the blow. Thus in fine-crushing there are three classes of stamps, gravity-stamps, steam-stamps, and pneumatic- or spring-stamps. In metallurgical work, the first are almost universal, the second are found here and there, and the third not at all.

263. Gravity Stamps.—Fig. 338, with legend denoting parts, represents a California[1] wet-stamp mill. A stamp, made up of stem *O*, tappet *P*, boss *N* and shoe *M*, is lifted by cam *U*, revolved by cam-shaft *R*, and allowed to fall by its own weight. Coming down with accelerated motion between guide-timbers *H* and *I*, it strikes the blow, when it has reached its greatest velocity, upon the ore lying on die *L* which has been introduced by feeder *Z*; the die protects mortar *J* which rests upon mortar-block *A*. The crushed ore is discharged through screen *K* on to the apron plate in front of it. A mortar holds almost always five stamps; the mortar with everything belonging to it is called a stamp-battery; two batteries are usually coupled together as in Fig. 338. The following discussion with its many details refers principally to the California stamp-mill as used in milling gold and silver ores.

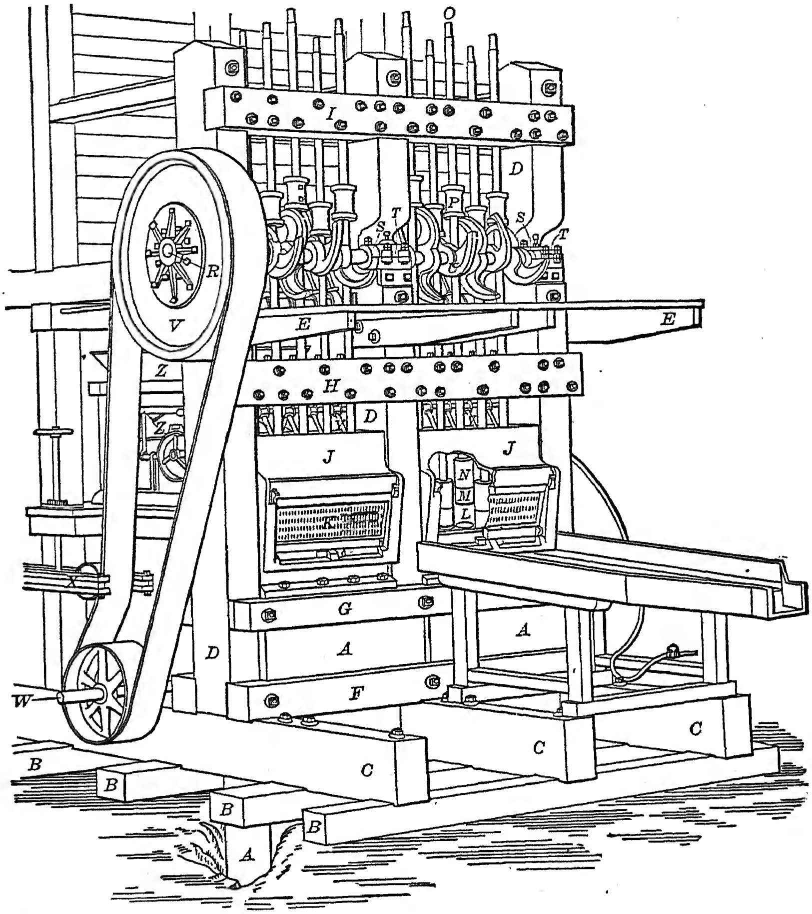
1. Foundation and Mortar Block.—A good foundation and mortar block are indispensable, as, if not built in the right way, the battery will shake itself to pieces. For the foundation a trench is dug 6 to 14 ft. deep, varying with the nature of the ground and the proposed height of the discharge of the mortar, and of sufficient length and width to leave an open space 24 in. wide around the block. The trench has been walled in in some cases.[2] The rough bottom of the trench is leveled by tamping down some sand or clay, and by placing on top two layers of 2-in. planking spiked together crosswise. A concrete layer, 24 to 30 in. thick, in some instances has taken the place of the tamping and planks.

The mortar block, which usually supports a single mortar, used to be almost always of wood. Planks 2 to 3 in. thick (thicker ones that are sound are difficult to obtain), 11 to 14 ft. long and varying in width are spiked together, fastened near the top by binders (*F.G*, Fig. 338) made of 8×12-in timbers bolted to one another by transverse rods. The block is of the same length as the mortars, but

[1] Origin: Stanford, *Min. and Sc. Press*, 1893, LXVII, 262; Martin, *Min. Eng. World*, 1912, XXXVII, 332.

[2] Abadie, *Tr. A. I. M. E.*, 1894, XXIV, 208.

3 in. wider in order to extend 1 1/2 in. beyond its flange. The life of a mortar-block is 8 to 10 years. The space around the block is tamped with tailings, earth or clay to the mud-sills (*B*, Fig. 338). The ends of the planks forming the block are sawed off, and the top is planed level and smooth. It receives a rubber cloth, 1/4 to 3/8 in. thick, to give an even bearing and to reduce the jar of



A. Mortor block.
B. Mud sills.
C. Cross sills.
D. Posts.
E. Platform.
F. *G*. Buckstaves.
H. Lower guide timbers.
I. Upper guide timbers.
J. Mortar.
K. Screen.
L. Die.
M. Shoe.
N. Boss.
O. Stem.
P. Tappet.
R. Cam shaft.
S. Collars.
T. Cam shaft boxes.
U. Cams.
V. Pulley.
W. Main shaft.
X. Tightner.
Y. Water pipes.
Z. Feeder.

FIG. 338.—Gravity stamp mill with ten stamps.

the mortar. The mortar, when put in place, is held down by eight bolts, 3 to 4 ft. 6 in. long and 1 3/8 to 1 1/2 in. in diam., passing through holes bored through the block. Ordinary bolts are held down by nuts in recesses cut into the block; eye-bolts may be let into the sides and are held by horizontal 2-in. rods passing through the eyes. With heavy stamps, masonry and concrete

foundations[1] have replaced wooden mortar blocks in most modern mills; they have not only well stood the wear, but have increased the crushing power of a stamp.

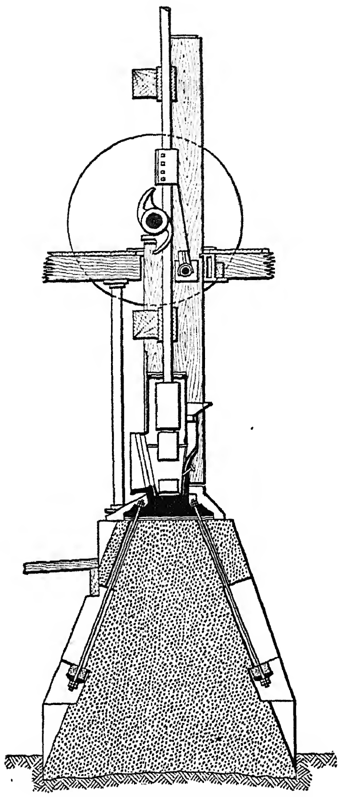


FIG. 339.—Concrete foundation for gravity-stamp battery with 1670-lb. stamps.

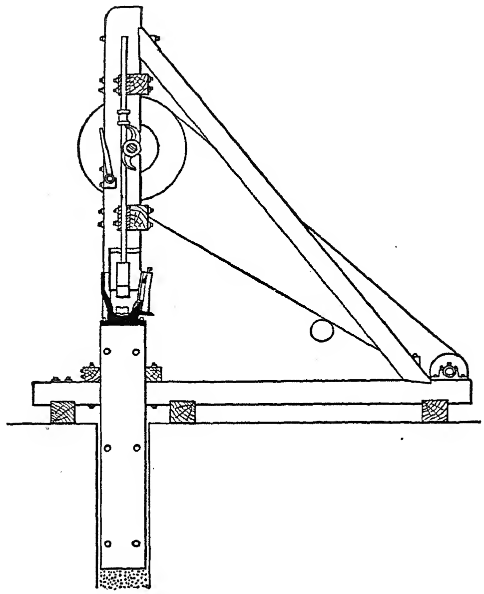


FIG. 340.—Front—A battery frame with cams in front.

Fig. 339 gives a cross-section through the new gravity stamp battery of the Simmer Deep Jupiter mill on the Rand.[2] The stamps weigh 1,670 lb.; with such

[1] Boss, *Min. Sc. Press*, 1903, LXXXVI, 72.
De Kalb, *Eng. Min. J.*, 1906, LXXXII 245.
Foote, *op. cit.*, p. 886.
Foote, *Min. Sc. Press*, 1906, XCII, 240.
Barbour, *Eng. Min. J.*, 1908, LXXXVI, 470.
Rice, *op. cit.*, p. 649.
Trustcott-Fuller, *Tr. Inst. Min. Met.*, 1908–09, XVIII, 25; *Eng. Min. J.*, 1909, LXXXVII, 175 (Goldfield).
Schmitt, *Eng. Min. J.*, 1910, LXXXIX, 763.
Caldecott, *Tr. Inst. Min. Met.*, 1909–10, XIX, 57; *Met. Chem. Eng.*, 1910, VIII, 458; *Eng. Min. J.*, 1909, LXXXVIII, 595.
Sewell, *Eng. Min. J.*, 1911, XCII, 629; *Min. Sc. Press*, 1911, CIII, 265.

[2] City Deep battery foundation, *Eng. Min.*, 1913, XCV, 951.

a weight, a concrete foundation becomes an absolute necessity, as battery posts of timber would be short-lived.

2. Frames and Guides.—The usual wooden frame consists of upright posts (*D*, Fig. 338) 12×24 in., placed on either end of the mortar on cross-sills *C* resting crosswise on mud-sills *B*. The posts are tied by the upper and lower guide-timbers *H* and *I*, and at the foot by binder *F* which is bolted to them and let into cross-sills *C*.[1] In the upper part of the posts are recesses to receive boxes *T* of the cam-shaft. The frames are stiffened and held in position by wooden braces and tie-rods. Their locations are governed mainly by position of main-shaft *H* (Fig. 338). There are two general types,[2] the A-frame and the knee-

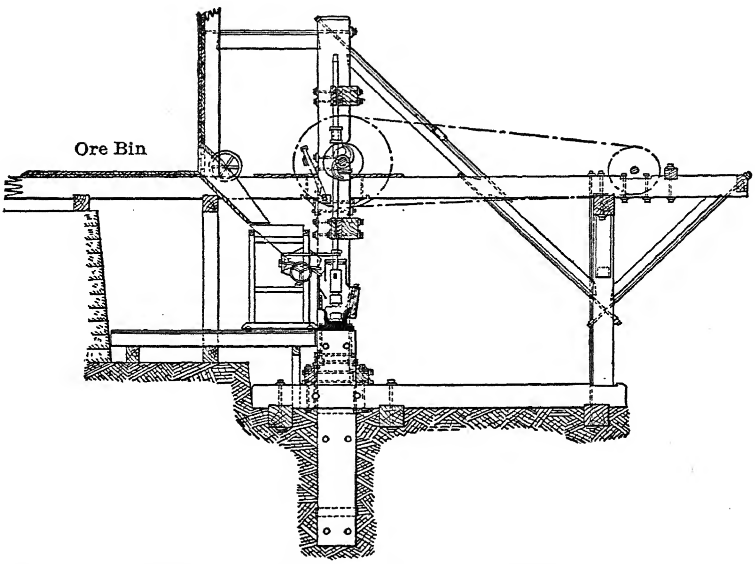


Fig. 341.—Front-knee battery frame connected with bin, cams in front.

frame; the former is common with a battery having its main-shaft on the cross-sill, the latter with the main-shaft on the cam floor. A few of the many forms are given in Fig. 340 (front-A battery frame with cams in front), Fig. 341 (front-knee battery frame connected with bin, cams in front), Fig. 338 (back-knee or single-post battery frame) and Fig. 342 (back-knee frame with brace).[3]

Iron frames are seldom used in the United States with gravity stamps where timber is abundant; they are more common in parts of Australia,[4] South America and South Africa.[5]

[1] Del Mar, Anchoring Battery Posts, *Min. World*, 1909, xxxi, 1015.

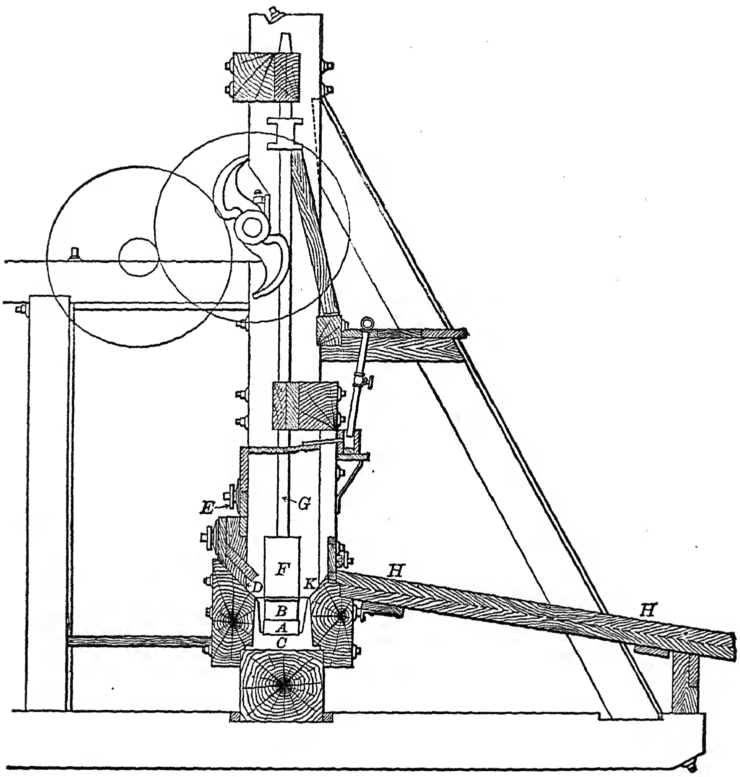
[2] Pheby, *Min. Sc. Press*, 1895, lxx, 376.
Preston, *Bull.* No. 6, State Mineralogist, Cal., 1895, p. 8.

[3] Rickard, *Eng. Min. J.*, 1895, lxxi, 265.

[4] Rickard, T. A., "Stamp-milling of Gold Ores," Scientific Publishing Co., New York; 1897, pp. 144, 159.

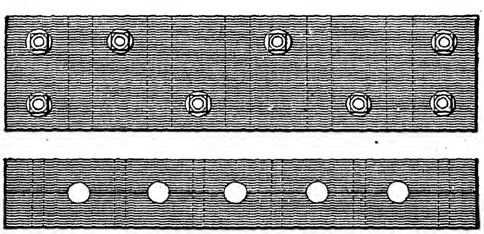
[5] *Min. Sc. Press*, 1895, lxxi, 265.

Two sets of guides hold the stamps in a vertical position while rising and falling; they are bolted to the lower and upper guide-timbers, *H* and *I*, Fig. 338, which also serve to tie the frames. The guides are placed about 3 ft. below and



A. Die.
B. Shoe.
C. Mortar.
D. Position of rear inside plate.
F. Boss.
G. Stem.
H. Amalgamating table.
K. Position of front inside plate.

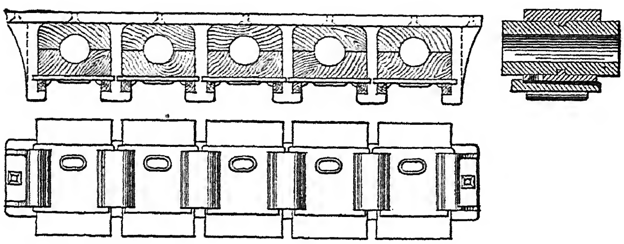
FIG. 342.—Back-knee (Gilpin county) battery frame with brace.



FIGS. 343 and 344.—Plain wooden guides.

above the cam-shaft *T*; the lower is thus situated between the mortar and the cam-shaft, the upper one above the tappets *P*. Plain wooden guides, Figs. 343 and 344, consist of hard-wood planks, usually 4 in. thick and of the same

width (12 to 14 in.) as the guide-timber, with semicircular grooves. At the beginning the planks, bolted to one another, are kept separate by shimes which are planed down as the wood of the guides wears off. Wooden guides in cast-iron frames, Figs. 345 and 347, have the advantages that the grain of the wood can be parallel to the stamp-stem which gives less friction than if it is horizontal as in the plain guide, that a bushing can be adjusted or replaced separately, and that a stem can be removed without disturbing the others. There exist many other forms which are to facilitate independent adjustment and to save time,



FIGS. 345 to 347.—Wooden guides in cast-iron frames.

such as the Acme, Broughall, Fargo, Wellman-Seaver-Morgan, Krupp, etc., but the plain wooden guide as well as the one with cast-iron frame is in common use.

3. MORTARS.—These are cast-iron boxes in which the ore is stamped. The ore is always fed near the top on one side; in single-issue mortars the pulp is discharged near the bottom on the side opposite the feed; in double-issue mortars on both sides. The former are common in wet-crushing and in the amalgamation of gold ores; the latter in dry-crushing of gold and silver ores which

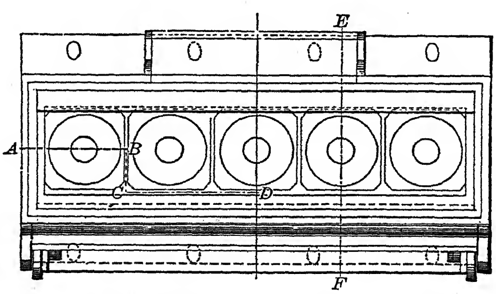


FIG. 348.—Homestake single-issue mortar.

have to be roasted before amalgamation. A single-issue mortar will allow liquid pulp to pass through the screen as fast as it is furnished, but not dry pulp which demands a larger surface. Double-issue wet-crushing mortars are used in ore-dressing when quick crushing without sliming is the leading aim. For the same purpose the so-called "individual mortars" (one for each stamp) with quadruple discharge have come into use.

The typical single-issue mortars used in the amalgamation of gold ores are

represented by the Homestake,[1] Figs. 348 to 350, the Newton,[2] Figs. 351 to 352, and the Union Iron Works, Figs. 353 to 360. The Gilpin County,

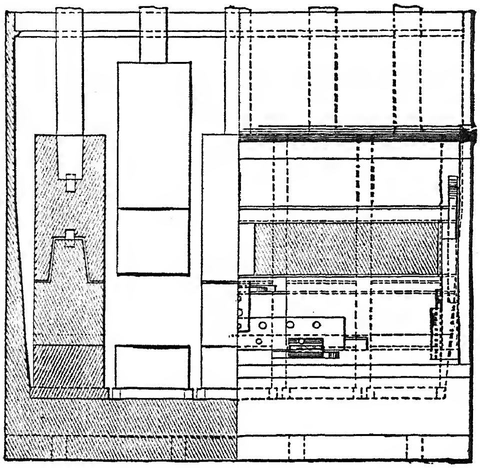


FIG. 349.—Homestake single-issue mortar.

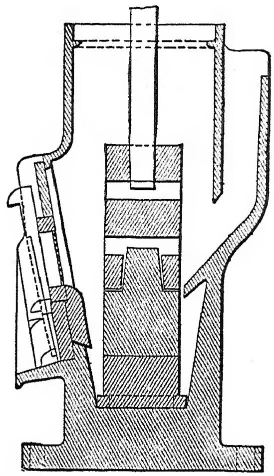


FIG. 350.—Homestake single-issue mortar.

Colorado, mortar, Fig. 342, is a local development. A double-issue, dry-crushing mortar of the Union Iron Works is shown in Figs. 361 to 364. Mor-

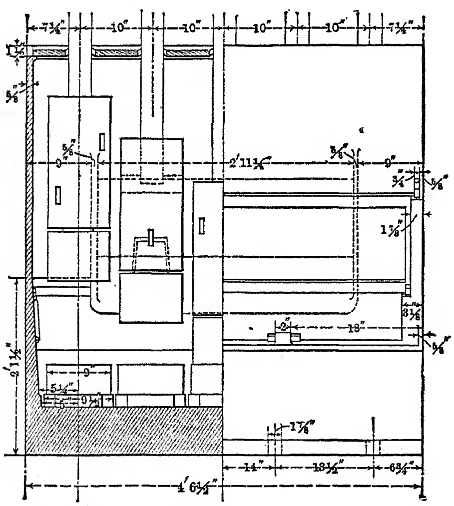


FIG. 351.—Newton single-issue mortar.

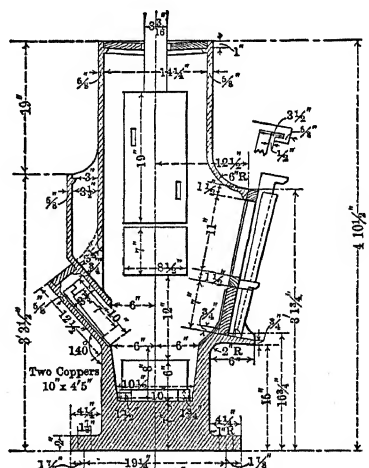
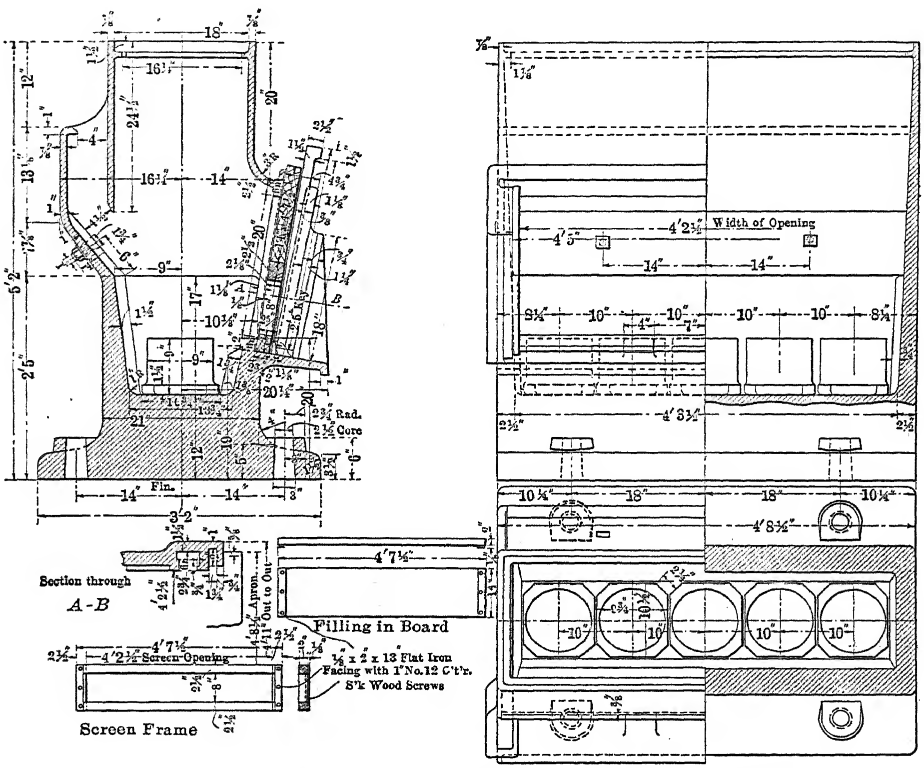


FIG. 352.—Newton single-issue mortar.

tars are usually cast in one piece; casting them in sections to be bolted together (sectional mortars), with pieces not weighing over 300 lb. to be suited for carry-

[1] Hofman, *Tr. A. I. M. E.*, 1888–89, XVII, 517.
[2] Snyder, *Eng. and Min. J.*, 1894, LVIII, 511.

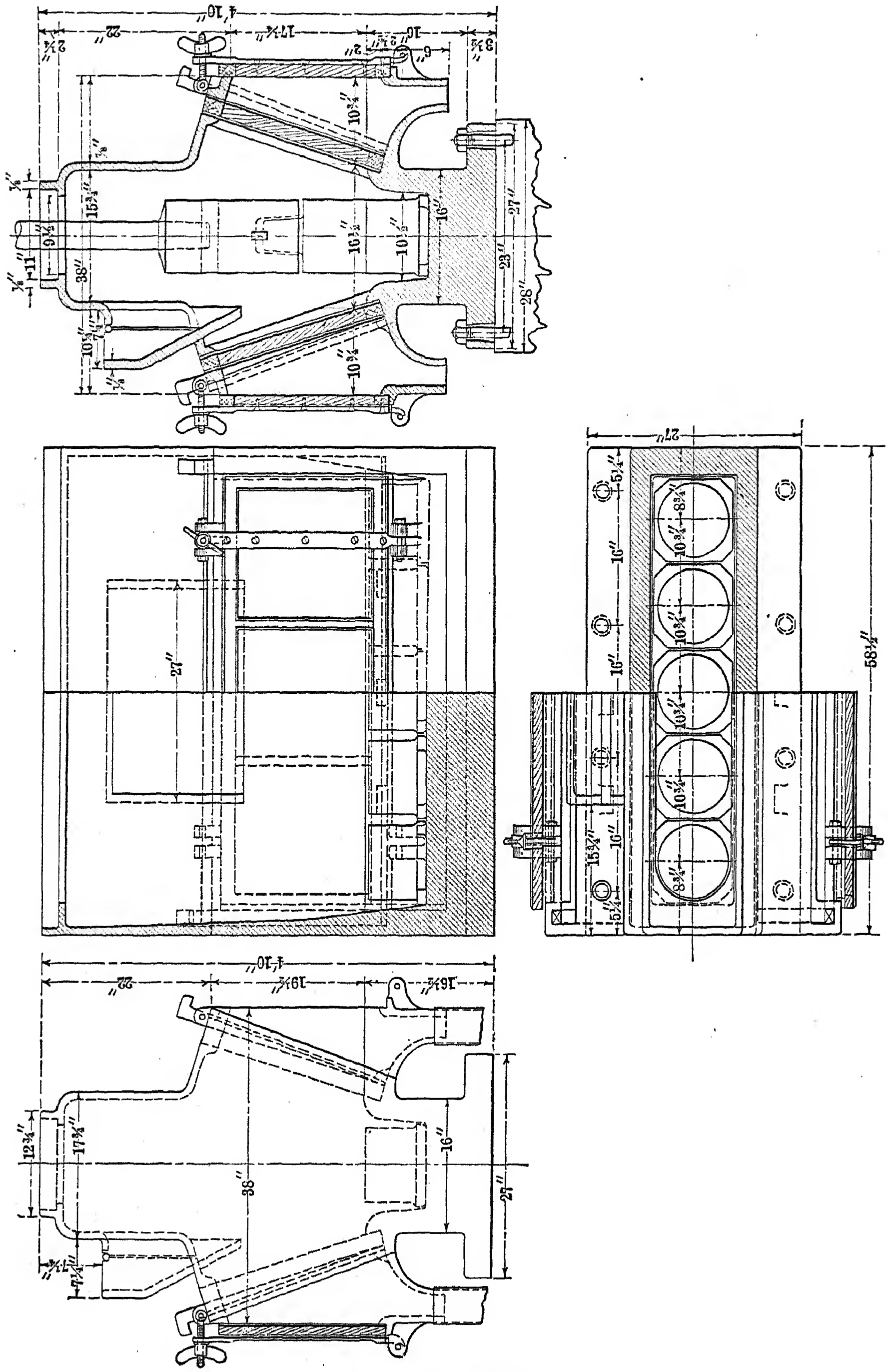
ing by mules, is exceptional. Mortars are thickest on the bottom which has to bear the chief strain; the sides and ends are heavy near the bottom and light at the top. Sometimes, Figs. 353 to 360, the sides are made less heavy than usual near the bottom and lined with a maximum of five chilled castings which are to take up the wear, viz., two for sides, two for ends, one for back-lining of feed-opening; the linings last about one year. This practice is advantageous with mills remote from foundries. The weight of a mortar, 3,000 to 7,000, av. 5,000 lb., varies with the weight of the stamps; the ratio is about 6 : 1. The width at



FIGS. 353 to 360.—Union Iron Works single-issue lined mortar.

bottom and at issue, and the height of discharge, that is, the vertical distance from top of die to lower edge of screen, govern to a large extent the work of the mortar. The wider the mortar, the slower the stamping; the greater the depth, the slower the stamping, the finer the pulp, the better the amalgamation (in wet-crushing), and the longer the life of the screens.

The dimensions vary with the character of the ore; the width at the top of the die ranges from 12 to 16 in.; the height of discharge from 6 to 10 in.; the feed-opening at the back is 3 1/2 to 4 1/2 in. wide; it extends over the length of the mortar or covers only the three middle stamps. The discharge at the front, nearly as long as the mortar, is inclined outward about 10° from the per-



FIGS. 361 to 364.—Union Iron Works double-issue mortar.

pendicular to assist in the discharge of the pulp. The mortar has a cover, resting on lugs made of two planks, Figs. 351 to 352, with the necessary semicircular openings for the passage of the stamps, and with the holes for the feed-water pipes (*Y*, Fig. 338) in wet-crushing. In dry-crushing the discharge openings are enclosed so as to retard the dust. Figs. 361 to 364 show the front on either side to be closed by three hinged doors; the dry pulp is delivered downward into covered troughs having screw conveyors.

4. SCREENS.—Discharge-screens regulate the maximum size to which the ore is to be crushed. A screen, Figs. 353 to 360, is mounted on a wooden frame about 8 in. wide and faced with iron on one side; sometimes it is divided

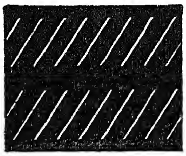


FIG. 365.—Clear-punched slot-screen.

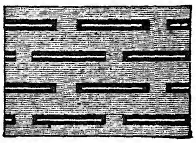


FIG. 366.—Burred slot-screen.

by vertical bars into two to four panels to prevent bulging of the screen, or in punched screens the whole surface is not covered with perforations, but parts of the plate are left blank (Foote screen). The frame is keyed to the planed face of the discharge by wedges driven in the end-grooves (Figs. 348 to 350, 351 to 352, 353 to 360). The upper part of the discharge not covered by the screen is closed by a suspended canvas curtain or a splash-board (Figs. 348, 350, 353 to 360). Screens are of punched metal (plate-screens) or wire-cloth (cloth-screens). The holes in plate screens are round or slotted, and may be clear-punched (Fig.

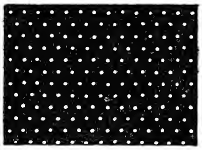


FIG. 367.—Clear-punched screen, round holes arranged in 60° rows.

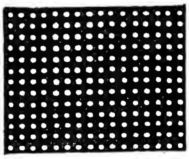


FIG. 368.—Clear-punched screen, round holes arranged in 90° rows.

365) or burred (Fig. 366); with the latter the rough edges facing the stamp can be closed when worn by striking with a mallet. The holes in wire screens are square. Round holes are either at 60 or 90° to one another (Figs. 367 and 368); slots are usually punched diagonally (angle-slot), sometimes horizontally, rarely vertically. Angle-slots are either in line or staggered. Round holes are used in ore-dressing for coarse (> 1 mm.) and medium-fine wet-stamping; slots in fine (< 0.01 mm.) wet-stamping as *e.g.*, in wet-stamping silver and gold mills using narrow mortars. In dry-stamping wire screens are used exclusively, as the mortars are wide, and as a wire screen offers a larger area of open spaces and a more even product than a plate screen. Plate screens are made princi-

pally of Russia iron and steel plate, occasionally of burnt or unburnt tin-plate which is thin and admits of rapid discharge; wire screens of steel or brass wire, preferably the latter. A slot is usually 1/2 in. long; its width is measured by the diameter of a sewing needle than can pass through it. Table 177 gives the numbers of the needles and the corresponding diameters of holes.

TABLE 177.—SIZES OF NEEDLES FOR SCREENS[1]

Needle No.	Thickness of plate, in.	Diam. of hole, in.	Diam. of hole, mm.	Needle No.	Thickness of plate, in.	Diam. of hole, in.	Diam. of holes, mm.
1	0.0243	0.058	1.47	7	0.0191	0.024	0.61
2	0.0243	0.049	1.25	8	0.0179	0.022	0.56
3	0.0243	0.042	1.07	9	0.01594	0.020	0.51
4	0.0243	0.035	0.89	10	0.01419	0.018	0.46
5	0.0219	0.029	0.74	11	0.01264	0.0165	0.42
6	0.0201	0.027	0.69	12	0.01264	0.015	0.38

Needle Nos. 7 to 12 (=30-to 60-mesh) are common sizes. The wire-cloth sizes of Tyler & Co. are given in Table 170. Screens last from 14 to 30 days; they wear out first near the bottom and are then turned upside down.

5. AMALGAMATION PLATES.—Inside and outside amalgamated plates belong to gold amalgamation.

6. DIES.—These are cylindrical iron blocks, one to a stamp, standing in the bottom of the mortar (*F*, Fig. 338) to save this from wear. The cylinder (boss) of a die is held in place either by lugs, Fig. 369, fitting into sockets in the mortar (common in dry-stamping), or by a square base (foot-plate) with beveled corners, common in wet-stamping. The foot-plates or the five dies cover the bottom of the mortar, the beveled corners leave spaces for inserting the point of a pick

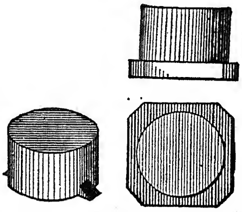


FIG. 369.—Die with foot-plate.

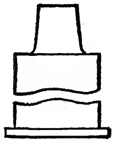
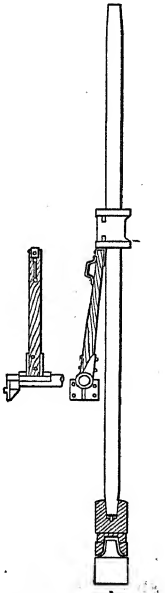


FIG. 370.—Worn shoe and die.



FIGS. 371 and 372.—Stamp with latch-finger.

to remove a die. The boss of the die is usually 9 in. diam. and 5 in. high, the foot-plate 10×10×1 1/4 in. Dies are made of gray to mottled iron

[1] Catalogue of Fraser and Chalmers.

chilled above the foot-plate, also of Mn- and Cr-steel, which are coming more and more into prominence; they weigh 120 to 160 lb. for stamps weighing 850 to 950 lb. Dies generally wear down more or less unevenly, Fig. 370, the boss becoming slightly convex (coning) and the shoe correspondingly concave (cupping); they are exchanged when worn down to within 1 or 1 1/2 in. of the foot-plate, the reduced weight being 20 to 50 lb. The life of a die, excluding quality of material, depends upon the hardness of the rock, the working of the stamp, the height of discharge of the mortar, and the fineness of the screen. The life of a cast-iron die is 30 to 40 days; that of a steel die 2 1/2 to 3 times as long; the consumption of iron per ton of rock stamped is probably <0.5 lb. with the die, and >0.5 lb. with the shoe.

7. STAMPS.—A stamp, Figs. 338, 371 and 372, consists of the stem which carries the tappet for lifting, and the boss (head) for connecting the shoe, which strikes the blow, and for giving additional weight. The weight of a stamp ranges in the United States from 800 to 1,200 lb., the different parts are proportioned about as follows: stem 40 per cent., tappet 15 per cent., boss $25\mp$ per cent., shoe $15\pm$ per cent.; the greatest weight beside that of the stem is in the boss.

(*a*) THE STEM is a solid round bar of wrought iron or mild steel, turned in a lathe or cold-rolled, which has a taper, 4 1/2 to 6 in. long and 1/2 to 3/4 in. per ft., at both ends to permit reversing in case of breakage. It is 12 to 16 ft. long, about 3 in. diam., weighs 300 to 400 lb., and lasts for years.

(*b*) THE TAPPET is shown in Fig. 373, which is a section of the common form of gib-tappet, a cylinder of cast-iron, sometimes of open-hearth steel, with central bore, to fit the stem loosely; it has planed faces. It carries in a recess a wrought-iron or forged-steel gib *B*, flat at the back and concave at the front, which is forced against the stem *A* by means of keys *K*, and thus attaches the tappet to the stem. A tappet is about 22 in. long, 9 in. diam. at ends and 6 in. at middle, has a wearing face 2 1/2 to 3 in., weighs $130\pm$ lb., and lasts three to five years.

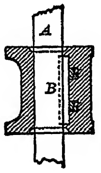


FIG. 373. Tappet.

In setting a tappet, a wooden block 11 in. higher than drop of stamp, is put between shoe and die, a prop is placed against the stem, allow tappet to rest on it and key fast. In raising a tappet, hang up stamp as in Fig. 371, put wooden block in place, loosen keys when stem glides on to block, and drive them fast again.

(*c*) THE STAMP HEAD OR BOSS is shown in Figs. 338 and 371. This is a tough cast-iron cylinder, often bound with a wrought-iron ring at ends, about 18 in. long and 9 in. in diam.; it weighs about 200 lb. and lasts about five years. On the top it has a bored socket to receive the stem, on the bottom a rough recess for the shank of the shoe; through it pass two horizontal keyways, at right angles to one another, for driving out stem and shoe by means of wedges. In fastening the head to the stem, the stem is lowered through guide-holes, driven with hammer, and lifted and dropped several times on timber placed beneath to take up shock.

(*d*) THE SHOE is illustrated in Figs. 361 to 364. It consists of a cylinder (butt) 9 in. in diam. and 7 to 8 in. high, surmounted by a truncated cone (shank), 4 1/2 to 5 in. long, and weighs 140 to 160 lb. It is made of cast-iron chilled to within 1 1/2 in. of the shank, also of C-, Mn-, or Cr-steel. It wears more evenly and slowly than the die, becoming slightly concave, Fig. 370; the die lasts longer as it is protected by about 1 1/2 in. of ore. When a cast-iron shoe is worn (30 to 60 days) down to 1 to 2 in. from the shank and weighs 20 to 50 lb., it is discarded. Steel lasts two to three times as long as chilled iron. The height of the shoe is often so chosen that shoe and die will be worn out at the same time. In fastening the shoe to the head, small wooden staves, Fig. 374, 1/4 in. thick and 1/2 in. wide, of dry pine are placed around the shank and held together by a strip of canvas nailed to them thus forming a collar. The shoe is put in place, and the stamp with head dropped until the shoe is wedged fast; in wet-stamping, the water of the mortar swelling the wood tightens the grip. Collars of wooden staves are kept on hand to save time when exchanging shoes.

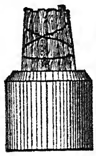
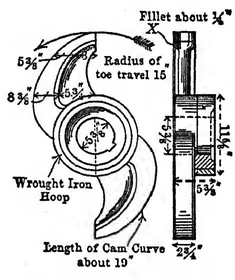


FIG. 374.—Staves on shank of shoe.

(*e*) THE LATCH-FINGER OR PROP, Figs. 371 and 372, serves to hang up the stamp so that the cam can revolve without striking the tappet. It is of wood, 3 ft. long and 3 in. square, is lined with 1/8-in. iron on the top and at the side facing the stamp, and rests in a cast-iron socket (latch-socket) which swings on a jack-shaft with bearings bolted to the battery-posts. The height of the jack-shaft is governed by the length of the prop and the height of drop of the stamp. When the stamp is hung up, the distance between shoe and die is about 1 in. greater than the height of drop. In hanging up a stamp, a flat stick, 1 1/2 in. thick, is placed on the face of the rising cam; this then raises the tappet sufficiently to allow pushing the prop in place. In disengaging the stamp, the operation is similar.

(*f*) THE CAM,[1] CAM-SHAFT AND CAM-SHAFT PULLEY are shown in Fig. 338. The cam *U* serves to lift and at the same time to rotate the stamp; it is keyed to the cam-shaft *R*, and this, resting in cam-shaft boxes and held in place by collars *S*, is driven by the cam-shaft pulley *V*.



FIGS. 375 and 376.—Double-armed left-hand cam.

Cams, Figs. 375 and 376, now are always double armed, that is, two arms cast to one hub; the curvature of the cam (cam-curve) is the involute of a circle (slightly fattened at the end) the radius of which is equal to the distance between the centers of the cam-shaft and the stamp-stem. Figs. 375 and 376 represent a left-hand cam, one that is to the left of the stamp when the upper arm of the cam is moving from the observer, the hub would also be on the left; the reverse would be a right-hand cam. Cams are usually of tough cast-iron with chilled lifting surfaces; the hub used to be strengthened by a wrought-iron band shrunk on it; C- and Cr-steel are used in some instances;

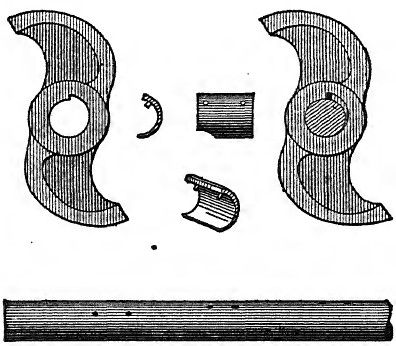
[1] Laml, *Eng. Min. J.*, 1909, LXXXVIII, 66.
Hutchinson, *Min. Sc. Press*, 1911, CIII, 767.

sectional cams have been suggested. The width of face of a cam is 2 to 2 1/2 in., the distance from center of hub to tip of cam 15 to 17 in.; the hub is 3 to 3 1/2 in. thick, the strengthening rib, 3 1/2 in. wide at face, grows to 9 1/2 in. at hub; the natural life is three years. The rotation of the cam causes the stamp, while being lifted, to make about 1/8 revolution; the effect is a more even wear of shoe and die, and lubrication is also made easy. A cam is keyed to the cam-shaft, usually with two steel keys which are driven toward the stem.

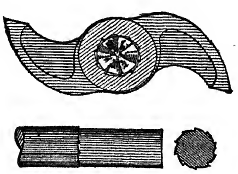
The cam-shaft, Fig. 377, has two long key-seats for the cams of a pair of batteries, while the key-seats in the cams are cut so as to advance 36° on the

Fig. 377.—Camshaft for ten stamps.

consecutive cams for a 5-, and 18° for a 10-stamp battery. The old Blanton cam, Figs. 378 to 382, is an improvement on the ordinary form in that it is quickly loosened and exchanged. A taper bushing is attached to the shaft by means of two pins fitting into corresponding holes, and becomes automatically wedged fast by the slipping of the cam when this is lifting the stamp. A modification is found in the new Blanton cam, Figs. 383 and 384, which has 10 taper faces which fit accurately 10 faces cut into the shaft. Thus no keys are required, and



Figs. 378 to 382.—Old Blanton cam.



Figs. 383 and 384.—New Blanton cam.

the order of drop is not definitely fixed as in the old Blanton, but can be changed to suit condition of work. Other forms are those of Boss, Canda, Krupp, etc.

The cam-shaft is of mild steel, 4 1/2 to 6 in., usually 5 1/2 in. diam., depending upon the weight of the stamp, and about 14 ft. long. The life ranges from 5 to 10 years. It has two collars, *S*, Fig. 338, held by set-screws to keep it in position in the cam-shaft boxes *T* let into the posts *D*. The boxes are frequently open, but preferably covered with a cap.

The cam-shaft pulley, *V*, Figs. 338, 385 and 386, is usually built up of wood on a hub of cast-iron with flanges. The diameter is 6 to 7 1/2 ft., the face 14 to

16 in. A rubber belt transmits the power from the counter-shaft w, and is kept taut by a tightener X. Transmission of power by gearing is characteristic for the slow-dropping Colorado stamp, Fig. 342.

(*g*) DROP.—The order, height and number have to be considered. The order of drop[1] is of importance, as upon this depends largely the even distribution of the rock upon the dies.

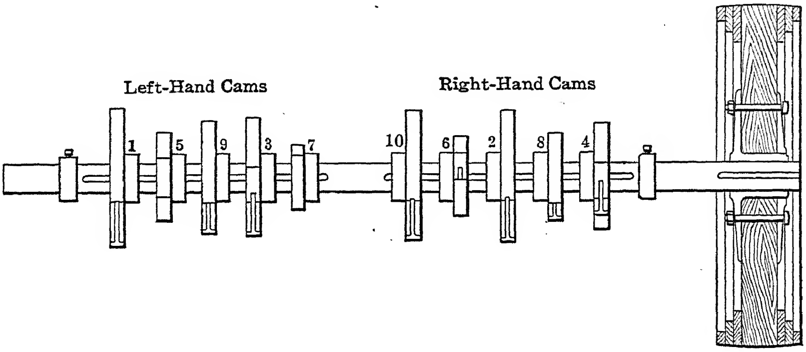


FIG. 385.—Camshaft with cams and wood pulley.

The governing rules are that two adjacent stamps shall not fall in succession, and that when one stamp falls its neighbor shall rise. Adjacent stamps dropping consecutively cause the ore to accumulate at one end of the mortar and form a cushion, while at the other end there is a lack of ore, and the stamp begins to pound; one stamp dropping forces ore under the neighbor which is rising. The following two orders of drop are considered the best: one is 1, 4, 2, 5, 3 or numbered backward 1, 3, 5, 2, 4; the other 1, 5, 2, 4, 3 or numbered backward 1, 3, 4, 2, 5; the former complies with both rules given above, and is more general than the latter which complies with neither; any piling up of ore under a stamp is corrected by giving this a higher drop. As two 5-stamp batteries are generally driven from a single cam-shaft, the order of drop in one battery will start with an end-stamp, while in the other with a middle-stamp, that the work may be evenly distributed; for the same reason the straight order of drop is used in one battery (stamps VI–X) and the reversed order in the other (stamps I–V). Figs. 385 and 386 show this arrangement; they denote also that one battery has left-hand and the other right-hand cams; the object of this is to balance the lateral thrust of the two sets of cams.

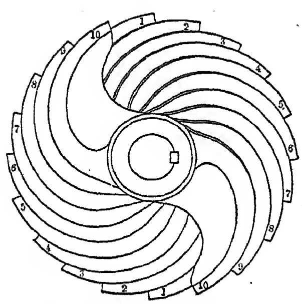


FIG. 386.—Camshaft with cams.

The number of drops is governed by the height of the drop. A large num-

[1] Richardson, *Eng. Min. J.*, 1903, LXXVI, 228.
Bell, *op. cit.*, 1910, LXXXIX, 597.
Munroe, *op. cit.*, 1910, XC, 949.

TABLE 178.—ORDER OF DROP IN A TEN-STAMP MILL

	Reversed order					Straight order				
Order of drop........	1	3	5	2	4	1	4	2	5	3
No. of stamp...........	I	II	III	IV	V	VI	VII	VIII	IX	X
Sequence of drops......	1	5	9	3	7	10	6	2	8	4

ber of drops requires a short drop and *vice versa;* the former with 80 to 110 drops per min. and a height of 4 to 9 in. represents California practice; the latter with 26 to 32 drops per min. and a height of 16 to 20 in. that of Colorado. The California method which aims to put through a large quantity of free-milling ore

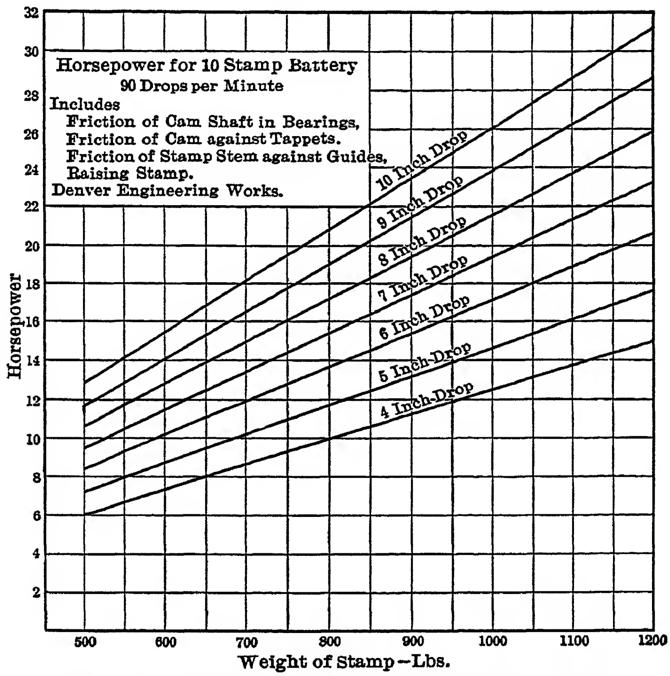


FIG. 387.—Horse-power required for a ten-stamp mill.

is the one generally followed; the Colorado method, which aims to extract from refractory ore as much gold as possible in the mortar by means of quicksilver, is almost local.

(*h*) POWER.[1]—The power necessary for running a stamp battery is consumed by the overcoming of gravity and of friction. The former can be calculated.

[1] Del Mar, *Eng. Min. J.*, 1910, LXXXIX, 7, 258 (Westwood).
Warwick, *Min. World*, 1910, XXXIII, 135.
General Engineering Co., Salt Lake City, *Mines and Minerals*, 1909–10, XXX, 87.

Richards[1] gives from different sources the ratio of total h.p. and that required to overcoming gravity as ranging between 1.099 and 1.202. Fraser and Chalmers assume 1, 1 1/4, 1 1/2, 1 3/8, 1 7/8 h.p. for stamps weighing 650, 850, 900 and 950 lb. Fig. 387, of the Denver Engineering Co., shows approximately the total h.p. required for a 10-stamp battery with stamps weighing 500 to 1,200 lb. which have a drop of 6 to 10 in. and 90 times per min.

(*i*) Duty and Product.—The duty or crushing capacity of a stamp in 24 hr. depends upon the efficiency of the stamp, the character of the ore, and the discharging capacity of the mortar. In the wet-crushing gold mills it ranges from about 1 ton with the Colorado to 4 tons with the California stamp. In recent years the last figure has been raised to 6 tons, since heavy stamps have become more common than they used to be, and the treatment of the large percentage of slime resulting therefrom has become more efficient. In dry-crushing silver mills the duty is from 0.5 to 1.5 tons.

Table 179 by Shaw[2] in his review of modern cyanide practice in the U. S. and Mexico gives the leading details of stamps of some of the principal gold mills. In Table 180 are recorded the data of Caldecott[3] showing the increase of stamp duty with increase of weight of stamp.

Table 179.—Details of Stamps in Modern Cyanide Mills

Name of mill	Number of stamps	Weight of stamps, pounds	Height of drop, inches	Drops per minute	Duty per 24 hr., tons	Screen mesh	Life of die, days	Life of shoe, days	Life of screen, days
Colorado	60	1,050	6–8	100	3.8	26	50	112	3
Combination	20	1,200	6	108	4.5	....	74	96	10
Desert	100	1,050	6	104	4.79	12–14	59	76	30
Don Estresllas No. 2	120	1,250	6.5	102	4.2	16, 26	65	65	2–5
El Oro	100	1,000	7.5	104	3.75	35	.....	.....	....
	100	1,150	6	102	4.00				
Goldfield Consolidated	100	1,050	....	108	......	16	.....	.....	....
Guanajuato Consolidated	80	1,050	7.5	104	3.6	50	.....	.....	30–35
Guanajuato Development (Pinguico).	40	1,050	6.5	104	6.25	2, 4, 8	......	....	....
Guanajuato Reduction	160	1,050	7.5	100	3.1	26	......	....	....
Homestake	1,000	900	10.5	88	4.0	No. 8 slot	......	....	.....
Loreto	40	1,050	......	106	3.0	16	......	.....	......
Montana-Tonopah	40	1,050	7	100	3.5	20	......	....	....
North Star	80	1,050	8	96	3.1	20	......	....	25
San Francisco	30	1,050	6.5	104	......	20	......	....	....
Standard	20	1,000	4–6	96–106	2.3	30	57	122	55
Veta Colorado	100	1,050	7	....	.....	8–10	......	.....	....

[1] *Op. cit.*, I, 204.

[2] *Tr. A. I. M. E.*, *Bull.* No. 31, July, 1909, pp. 591–619; *Min. Ind.*, 1909, XVIII, 741.

[3] *Tr. Inst. Min., Met.*, 1909–10, XIX, 61.

TABLE 180.—INCREASE OF DUTY WITH WEIGHT OF STAMPS

Apertures per square inch in mortar screen	Running weight of stamps, pounds	Set height of drop, inches	Height of discharge, inches	Tons of water per ton ore	Tons ore crushed per stamp per twenty-four working hours	Over 0.01 in. size in screen pulp, per cent.
981 (0.021 in.)	1,196	8	3	5.8	5.88	22.63
981 (0.021 in.)	1,279	8	3	5.8	6.58	22.23
981 (0.021 in.)	1,531	8	3	5.7	6.74	20.86
1,512 (0.016 in.)	1,216	8	11	5.40	4.26	5.16
1,512 (0.016 in.)	1,288	8	11	5.30	4.29	4.19
1,512 (0.016 in.)	1,293	8	11	5.43	4.55	9.49
1,512 (0.016 in.)	1,337	8	11	6.27	4.96	6.66
1,512 (0.016 in.)	1,562	8	11	5.05	5.17	....
1,512 (0.016 in.)	1,605	8	11	6.30	6.02	11.66

(*j*) FEEDING.—Mortars were formerly fed by hand, a man supplying three 5-stamp batteries with ore during a 12-hr. shift. Hand-feeding is still practiced in small mills in which, *e.g.*, the highest level is that of the mortar-floor, and the ore arriving in teams is dumped and broken with sledges. Mechanical (automatic, self-) feeders have increased the crushing power of a battery from 15 to 20 per cent., reduced the wear of shoes, dies and screens, and saved labor. A self-feeder consists essentially of a hopper with movable bottom; it receives the ore crushed to about 1 in. size, and discharges it into the feed-opening of the mortar. Motion is imparted to the bottom by the tappet of a stamp, usually the central one, striking the buffer of a vertical (bumper) rod, connected by some means with the bottom, when the layer of ore on the die becomes too thin. The differences in feeders lie in the form and the motion of the bottom. A feeder should deliver the ore regularly, a given number of lb. per min., when it has once been adjusted. The Hendy, Tulloch and Hammond feeders may serve as types.

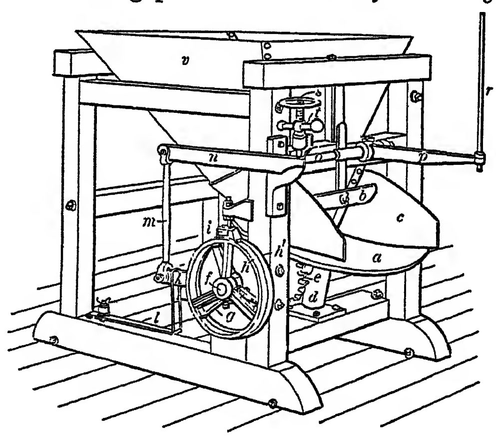


FIG. 388.—Hendy Challenge ore-feeder supported by frame.

a, Carrier plate; *b*, adjustable gate; *c*, scrapers; *d*, inclined shaft on step; *e*, pinion; *f*, shaft; *g*, friction pawls; *h*, outer rim; *h'*, inner rim; *i*, friction brake; *k*, lever; *l*, spring; *m*, connecting rod; *o*, shaft; *p*, lever; *r*, bumper-rod; *s*, hand-screw; *t*, lock-nut; *v*, hopper.

THE HENDY CHALLENGE ORE-FEEDER (Fig. 388).—Sheet-iron hopper *v*, ending in fixed scraper *c*, delivers from chute, under adjustable gate *b*, ore upon inclined (12 1/2°) table (carrier-plate) *a* having a bevel gear on under side (not shown), and standing on shaft *d*. The motion is imparted to *a* from central bumper-rod *r*, through lever *p*, shaft *o*, lever *n*, connecting rod *m*, lever *k*,

friction pawls g and shaft f to pinion e engaging with bevel gears under a. Spring l returns lever k to its place after each stroke, while at the same time friction brake i prevents shaft f from turning. Lever k is attached to inner rim h', loose upon shaft f, and drives by friction the outer ends of pawls g, the inner ends of which are attached to shaft f. The amount of movement is limited by hand-screw r which is clamped by lock-nut t. The ore on plate a is scraped into the mortar feed when it has reached a certain point on the plate. The feeder is supported by a wooden frame.

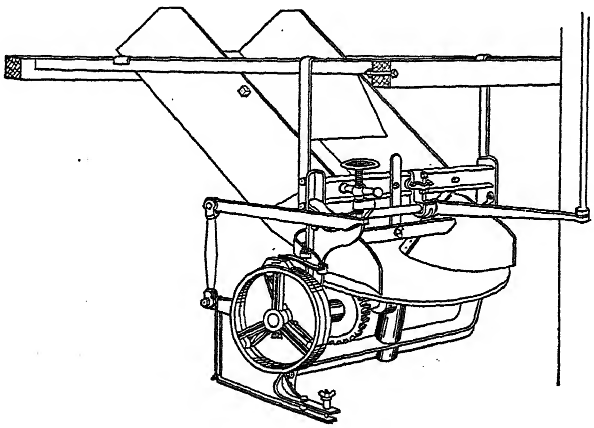


FIG. 389.—Hendy challenge ore-feeder, suspended.

In Fig. 389 is shown a later form, the "Challenge Improved Suspended Self-feeder," which makes the feeding-side of the mortar more accessible. The Challenge feeder is the most expensive form, but is suited for all sorts of ore, even when clayey or sticky.

THE TULLOCH ORE-FEEDER is shown in Fig. 390. The hopper resting on a carriage is closed by an inclined oscillating chute suspended by three rods. When

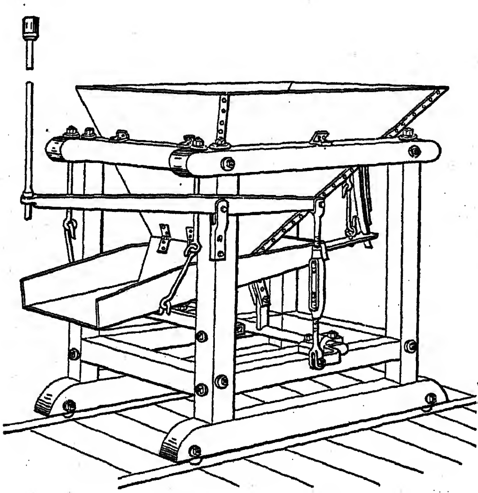


FIG. 390.—Tulloch ore-feeder.

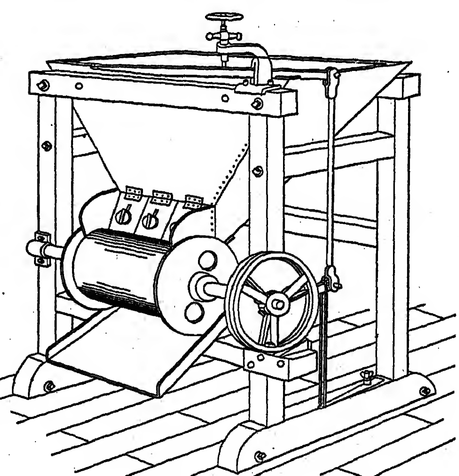


FIG. 391.—Hammond ore-feeder.

the tappet strikes the bumper-rod, the chute is thrown backward through the action of the tilting-lever, link, crank-arm, rock-shaft and arm, and causes the ore at the end of the chute to drop off. This is followed by a forward motion, induced by the spring, which causes lugs of the tray to strike a jar-rod, and

TABLE 181.—SCREEN ANALYSES OF STAMPED AND ROLLED PULP

WET STAMPING

Gold-silver ore; gangue: quartz, slate, granite	Screen analysis	Gold-silver ore, gangue: quartz	Screen analysis	Silver ore, gangue: quartz	Screen analysis
Montana Mining Co., Marysville, Montana, Richards, "Ore Dressing," I, 303.	Mesh Per cent. on 30— 0 02 on 40— 0.12 on 60— 9.10 on 80—11.99 on 100—10.16 th. 100..... 68.58	Smuggler Union Mg. Co., Telluride, Colo., Richards, "Ore Dressing," I, 304.	Mesh Per cent. on 20— 0 02 on 40—18.17 on 60—18 93 on 80— 9.40 on 100— 4.93 on 120— 4 96 on 150— 9.04 th. 150.....34.55	El Oro Mining and Railway Co. Mexico; Butters, *Tr. Inst. Min. and Met*, 1904–05, XIV, 33.	Mesh Per cent. on 40— 2.73 on 60— 9 40 on 80—11.87 on 100— 8.87 on 150—12.67 on 200— 5.50 th. 200 { 28 80 (sand) 20.10 (slime)

DRY STAMPING

Silver ore rich in blende and galena	Screen analyses		Gold ore, exceptionally hard	Screen analysis	Gold ore, eruptive rock and limestone	Screen analysis
Del Oro, Mexico., Hofmann, *Eng. Min. J.*, 1889, XLVII, 165.	Mesh Per cent. th. 20—100 th. 30— 93.8 th. 40— 87.3 th. 60— 78.8 th. 80— 71.2 th. 90— 67.2	Mesh Per cent. .... th, 40—100 th. 60— 98.95 th. 80— 93.80 th. 90— 90.50	New Zealand, McConnell, *Tr. Inst. Min. Met.*, 1888–89, VII, 27.	Mesh Per cent. .... th. 30— 100 on 40— 1.20 on 60— 9.00 on 90—12 50 th. 90—77 24	Aspen, Colo., Argall, *Tr. Inst. Min. Met.*, 1901–02, X, 268.	Mesh Per cent. .... on 30— 0.33 on 60— 8 70 on 120—13.07 on 200—16 92 th. 200—60 89

DRY ROLLING

Gold ore, exceptionally hard	Screen analysis
Locality not given, Argall, *Tr. Inst. Min. Met.*, 1901–02, X, 268	Mesh Per cent. th. 30—100 on 40— 7 80 on 60—25 00 on 80—24 20 on 200—35 00 th. 200— 8.00

draws ore from the hopper, the amount being regulated by the gate and an adjustable scraper in the hopper (not shown). The feeder is light and cheap, but is suited only for granular ore; it fails with wet, sticky, clayey material.

THE HAMMOND ROLLER SELF-FEEDER[1] is shown in Fig. 391. The bottom of the hopper is closed by a hollow iron cylinder, the shaft of which is revolved by friction pawls similar to those of the Hendy feeder. It is cheaper than the Tulloch feeder, but its use is restricted to certain classes of ores. It has given satisfaction in Nova Scotia and Alaska gold mills.

264. Gravity Stamps and Rolls.—In a comparison of these two leading apparatus for fine-crushing, the machine as a whole and the physical character of the pulp will furnish the leading considerations. Stamps are simple in construction and operation, and are suited for a great variety of ores which they crush fine in a single operation; they are, however, expensive to erect, require much space, and are wasteful of power. The reverse is the case with rolls. The pulp produced with a stamp contains a large portion of material that is much finer than the size the screen calls for; this is especially the case with ore that is strongly mineralized. The pulp from rolls is more uniform in size and contains a smaller proportion of impalpable dust. This sliming or dusting of part of the ore is desirable in some cases, and has to be avoided in others; special grinding apparatus is used when slime or dust is called for. Screen-analyses[2] of some products from stamping and rolling are given in Table 181 and 182.

TABLE 182.—SCREEN ANALYSES OF PULP FROM DIFFERENT CRUSHING MACHINES[3]

Kind of ore	Kind of machine	Screen-size, mesh	Percentage which passes the next coarser sieve and remains on sieves with the following mesh-sizes			
			30 Per cent.	40 Per cent.	60 Per cent.	90 Per cent.
Gold ore from South Africa; gangue: quartz conglomerate with some pyrite.	Wet stamps........	30	5.60	12.66	17.58	64.16
	Wet stamps........	26.46	11.15	28.53	9.21	51.11
	Dry stamps.........	20	20.30	9.80	21.80	49.10
	Rolls...............	20	26.63	33.99	13.06	26.30
	Rolls..............	22.36	9.30	41.85	15.38	33.47
	Ball mill............	22.36	20.07	24.38	13.88	41.67
	Niagara pulverizer.[4]	....	20.17	24.30	24.30	31.23

[1] Lamb, *Tr. A. I. M. E.*, 1910, XLI, 672.

[2] Hardman, *Tr. Min. Soc. Nova Scotia*, 1892–93, Vol. I, Part I, p. 36; *Eighth Ann. Rep.*, State Mineralogist, Cal., 1888, p. 700.

[3] Johnson, *Proc. Chem. Met. Soc. S. Africa*, 1897–98, II, 214.

[4] Similar to the Tustin, except that the roller is weighted by a spring, *Coll. Guard.*, 1895, LXX, 652; *Engl. Inst. Civ. Eng.*, 1892, CVIII, 127.

265. Steam Stamps.—This class of stamps which has become of considerable importance in ore-dressing has seldom been applied[1] in gold- and silver-milling, the leading branch of metallurgy in which stamps are used at all. Small steam stamps have been designed for testing gold ores on a working scale in connection with opening up a property. They are easily transported, require little foundation, and are quickly mounted and dismounted. The leading type is the Tremain Stamp.[2] Two stamps, each 300 lb. in weight, which carry the pistons of live-steam cylinders, ascend and descend alternately in a single mortar, making 200 drops per min., and crush the ore with a power corresponding to that of an 800- or 1,000-lb. gravity stamp. The capacity is about that of an ordinary 5-stamp mill.

266. Arrastra.—This primitive grinding apparatus which probably had its origin[3] with the Patio process, consists of a circular pavement of hard rock, over which boulders (drag-stones) are slowly pulled around to grind crushed ore in the presence of water. The circular pavement,[4] Fig. 392, 6 to 20 ft. in diam., is of coarse-grained quartzose rock (granite, porphyry, quartzite) which is harder than the ore and does not become smooth in use. The blocks, about 12 in. thick, are fitted closely, and the joints filled with tailings or, better still, cement. In either case, the arrastra is run for a few days with waste rock, using first 1, then 2, 3 and 4 drags to wear even the pavement and drags, and to harden the filling of the crevices. The pavement is laid upon a layer of sand, 2 to 3 in. thick, spread over a foundation of clay (puddle-bottom), beaten down firmly, or preferably concrete, about 1 ft. thick. The foundation projects 1 to 2 ft. beyond the grinding floor to guard against leakage. The pavement is surrounded by a wall, 2 to 4 ft. high, made of flagstones, laid in clay, or preferably in cement, or of wooden staves, tied with iron hoops or rods. In the center is a stone or wooden block with cast-iron step for the pivot of the rotating shaft journaled at the top. From the shaft extend 2 to 8, usually 4, horizontal arms to which are attached by means of chains, ropes or strips of raw-hide, 2 to 8, usually 4 dragstones weighing 80 to 2,000, ordinarily 300 lb.

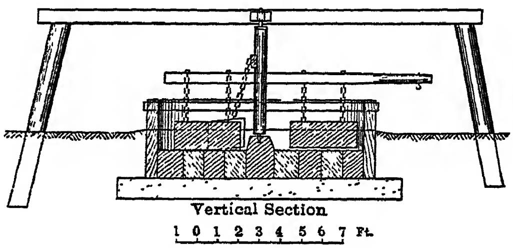


Fig. 392.—Arrastra.

These stones, generally of the same material as the pavement, are more or less flat with the lower face wider than the upper; two holes are drilled in front of

[1] Rickard, T. A., "Stamp-milling," Engineering and Mining Journal, New York, 1894, p. 247.
Louis, H., "Gold Milling," Macmillan, New York, 1899, p. 280.

[2] Sperry, *Tr. A. I. M. E.*, 1896, XXVI, 545.
Loram, *op. cit.*, 1905, XXXV, 707.
Parsons, *Min. Sc. Press*, 1908, XCVII, 386.

[3] Boss, *Tr. A. I. M. E.*, 1903, XXXIII, 244.

[4] Louis, H., "Gold Milling," Macmillan, 1899, p. 284.

the median line and plugged with dry wood to receive eye-rings for attaching the chain, etc. A stone is suspended in such a way that the outer end is a little in advance of the inner, and that the front is 3/4 to 1 in. higher than the back which just touches the pavement; the front is higher in order that at the start the stone may ride over the coarse ore. In small arrastras driven by animal power, the horse or mule is harnessed to a separate arm or to one of the drag-stone arms, made long enough to reach beyond the retaining wall, and walks around the circle about six times a minute. A large arrastra is often driven by water-power[1] when the central shaft makes 10–14 r.p.m.; several large arrastras may receive their power from a single steam engine.[2]

The ore to be ground is coarse-crushed to 1/2 or 3/4 in. A 10-ft. arrastra receives a charge of about 1,000 lb. in 2 or 3 portions. This, when spread uniformly over the floor, will form a layer 1 or 2 in. in thickness; it is dampened and grinding begun, the shaft making at first 2 to 4 r.p.m. As the grinding proceeds, the revolutions are increased; water is added at intervals to form with the plup a uniform thick mud. With an excess of water the ore is pushed forward and not ground; with too little it is packed down. The time of grinding ranges from 6 to 12 hr. When ground to the requisite size, the pulp is thinned with water and discharged through openings in the retaining wall. Drags last about 30 days, the wear amounting to 2 to 11 per cent. of the weight of the ore; when worn to one-third their original weight they are replaced one at a time in order that the grinding may proceed uniformly. One man attends two arrastras in a 12-hr. shift; animals work in 6-hr. periods. The arrastra is cheap to build and to run, but has a very small capacity.

267. Pans.—A pan is a circular vat in which wet pulp is ground fine between cast-iron plates. In amalgamation processes it has two additional functions, the chemical reduction of the metal-bearing mineral and the intimate mixing of it with quicksilver. At first, pans were used for grinding only; later they formed the standard apparatus for silver amalgamation; in recent years, since pan amalgamation has lost much of its former importance, pans have again come into more general use for sliming tailings from gold-amalgamating stamp-mills previous to treatment with potassium cyanide.

A large variety of pans has been constructed since the early fifties. The first was probably that of Knox.[3] It was followed by those of Wheeler, Varney, Hepburn, Horn, Greeley, Patton, McCone, etc.[4] A conical bottom was favored for grinding, a flat bottom for amalgamating; the latter has replaced the former, as experience has shown that the power required is not greater, that the wear is more even, and that repairs are easier.

The standard form of amalgamating pan is the combination pan which com-

[1] Illustrations in Richards, "Ore Dressing," I, p. 238.
Chism, *Tr. A. I. M. E.*, 1883, XI, 61.
Mathyas, *Thirteenth Rep.*, Cal., State Min. Bureau, 1896, p. 394.

[2] Catalogue Fraser and Chalmers, "Patio Process."

[3] Raymond's Reports, U. S. Treasury Dept., 1870, 685.

[4] U. S. Geol. Exploration of the Fortieth Parallel, Washington, 1870, vol. III, "Mining Industry," p. 217.

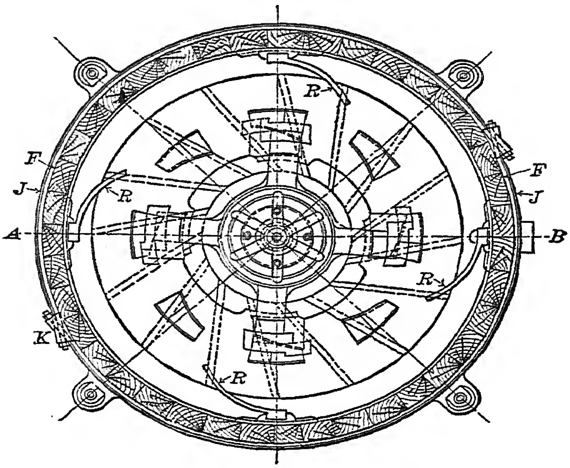


FIG. 393.—Combination pan of Fraser and Chalmers.

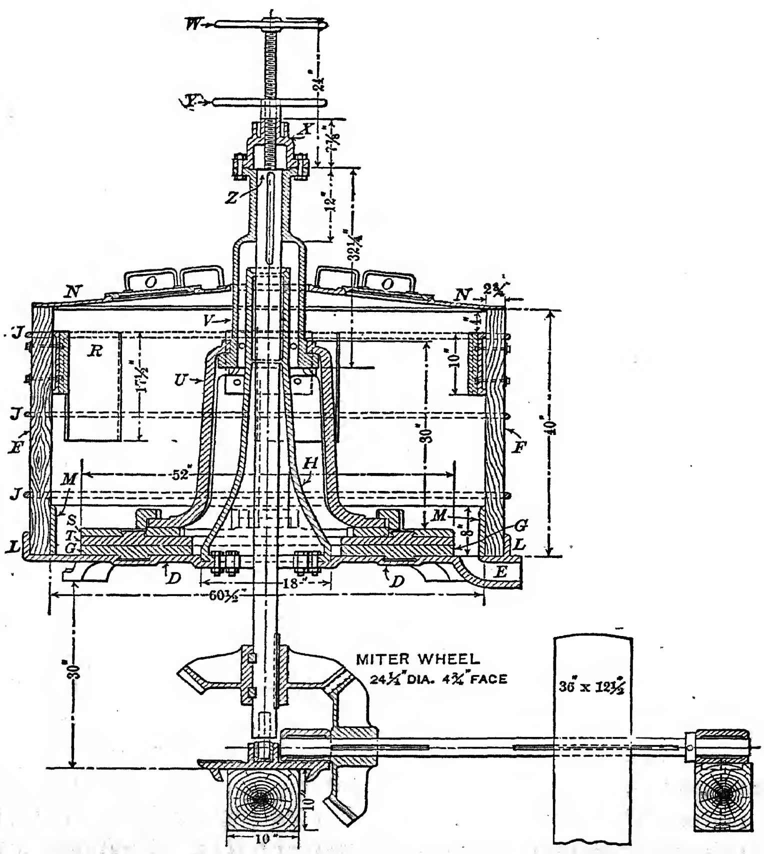
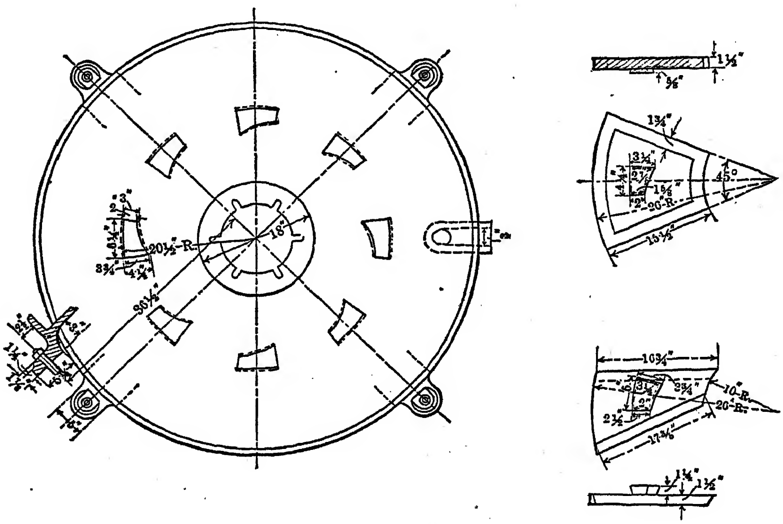


FIG. 394.—Combination pan of Fraser and Chalmers.

bines some of the valuable features of the Wheeler and Patton pans, and has characteristics of its own. Figs. 393 and 397 represent the type developed by Fraser and Chalmers.

The flat annular cast-iron pan-bottom *D*, usually 60 in. diam., has four feet which are bolted to two longitudinal timbers common to a row of pans; the timbers are supported by pairs of posts under each pan, which stand on cross sills and have cross bars carrying boxes or bearings. A steam-chest is sometimes bolted to the bottom of a pan to heat the pan-charge with waste steam. The pan discharge *E* is at the side near the bottom; its mouth, 3 in. diam., is closed with a wooden plug. The side *F*, formerly cast in one piece with the bottom, is now built up of wooden staves, 2 3/4 to 3 in. thick and 40 in. long, which are



FIGS. 395–397.—Combination pan of Fraser and Chalmers.

tied by 3/4-in. rods *J*; these pass with their threaded ends through binder blocks and are tightened by means of nuts. The staves are backed by the flange *L* and the joint made tight by calking; they are protected inside from wear by the ring *M*, usually a separate casting weighing about 400 lb.; the ring beside protecting the wood, furnishes finely-divided iron for the chemical reduction of silver compounds. The sides carry wood or iron covers *N* made in two sections, each with an opening for inspection and sampling closed by a lid. In charging a pan, one cover only is removed. The central part of the pan is closed by the cone *H* ending in a cylinder which forms a guide for the shaft. The cone may be cast in one piece with the pan, but is preferably joined to it by flanges and bolts. The bottom of the pan is protected against wear by dies *G* of chilled iron which form a flat ring 1 1/2 to 2 in. thick. This grinding surface may consist of several pieces, eight in the figure, or form a single ring; the independent pieces are fastened to the bottom of the pan by dovetail lugs fitting into corresponding grooves; a single ring is usually cemented in. Channels, 1 to 1 1/2 in. square, oblique to the radius are left open between the single dies or cast in the

die-ring in order to assist in the travel of the pulp and in spreading it between the grinding surfaces. A die ring is easier to exchange and protects the bottom better than do independent dies, but does not wear evenly. The chilled-iron shoes *T*, which form the upper grinding surface, are oblique sectors, 1 1/2 to 2 in. thick, fastened with dovetail wedges to the muller *S*. This, an annular cast-iron plate, has on the surface 4 lugs for the legs of the spider *U*, or plate and legs form a single casting. The legs end in a cone that is bolted to the driver *V*; the latter

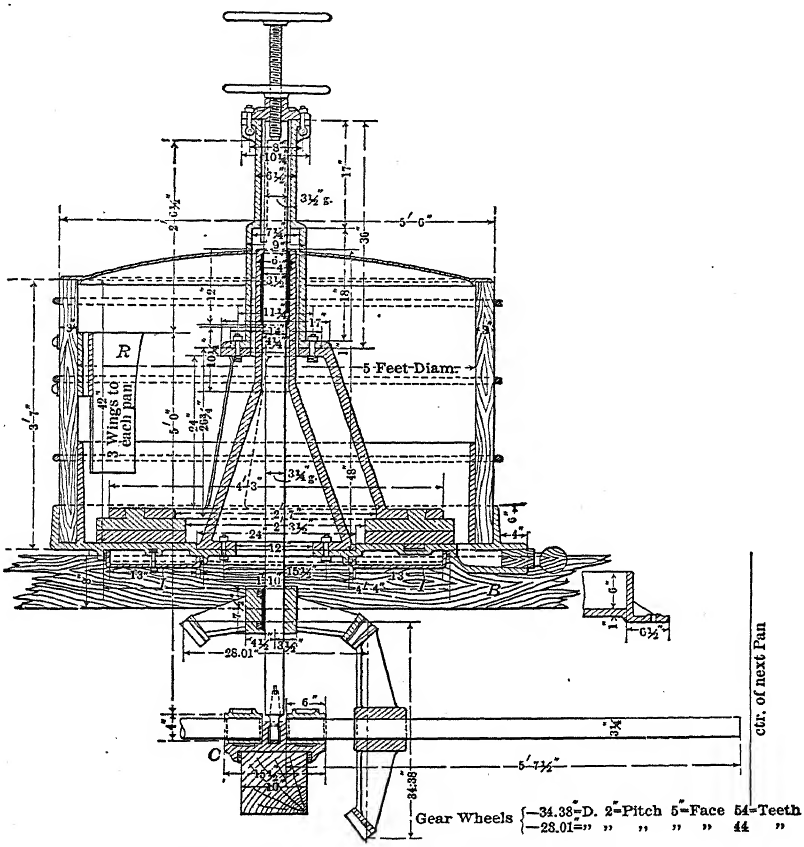


FIG. 398.—Combination pan of M. P. Boss.

has a long bearing with vertical groove into which fits the spline of the working shaft *Z* and thus permits rotation of the muller. To the driver is bolted the threaded cap *X*; its screw resting on top of the driving shaft *Z* allows raising and lowering the muller by means of the handwheel *W*, and maintaining it in position by the lockwheel *Y*. By driving through groove and spline, and by the mode of suspension, the muller can be raised and lowered while rotating. The driving shaft *Z* turns in a step below the pan and is held in position by the bearing of the driver *V*; it is rotated by a bevel gearing below which receives its power from a line-shaft by a slack belt and a tightener. To the sides of the

pan are bolted or dovetailed four wings R, plough-shaped castings, which extend some distance downward and deflect the pulp downward and toward the center of the pan. The proper circulation of the pulp, the pulp current, is of vital importance for effective work. The pulp is whirled around by the centrifugal force of the muller; it is made to pass from the center outward by the oblique channels of shoe and die which cut like plows, and to rise at the sides; it is then deflected at the top by the wings toward the center between the legs of the muller, and descends to start again on its outward flow. The consistency of the pulp, the construction of the pan and with it the r.p.m. of the muller are the leading factors.

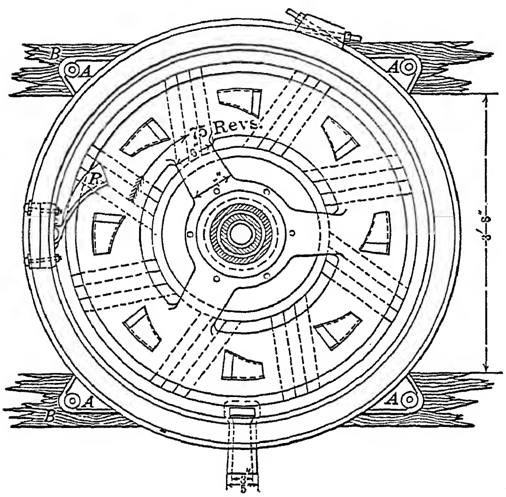


FIG. 399.—Combination pan of M. P. Boss.

A 5-ft. pan weighs about 8,000 lb., makes 75 r.p.m., and requires about 6 h.p; it receives a charge of about 2 tons 30- to 60-mesh ore, grinds this in 3 to 4 hr. to 100-mesh and amalgamates the silver for the same period of time. It consumes from 1/5 lb. iron per ton with ore that is free-milling, poor and not ground, to 8 lb. with ore that is free-milling and ground, and to 25 lb. with ore that is base, rich and roasted (Washoe and Reese River processes).

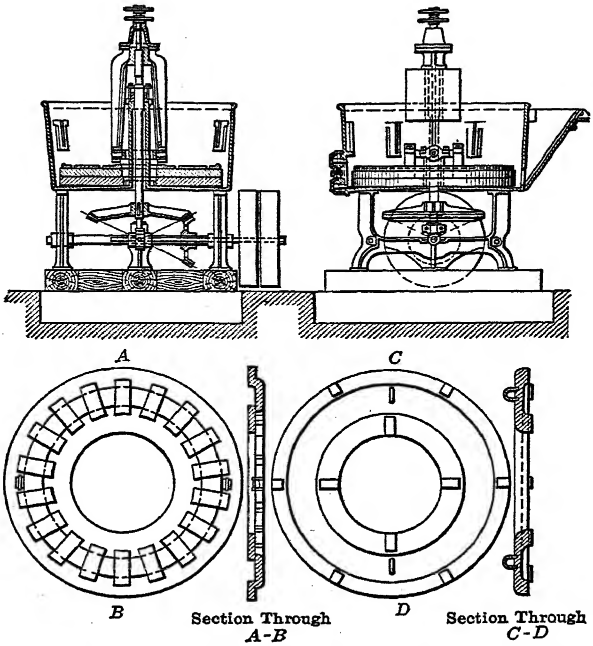
The pan of M. P. Boss, constructed for the continuous or Boss system of amalgamation, is shown in Figs. 398 and 399. The three major characteristics are that the steam-chest of the bottom extends up into the cone and increases the heating surface, that there are inlets and outlets for the flow of the pulp from pan to pan, a number being set up in a row close to one another, which requires a different connection with the main driving shaft, and that the pans serve for amalgamating only, the grinding being carried on separately in special continuous grinding pans working in pairs.

268. Continuous Grinding Pans.—This class of pan has been introduced into gold and silver mills to grind fine particles about 0.02 in. diam. In South Africa and Australia the pan of Denny Bros. is the one best known; in the United States that of the Colorado Iron Works. Both are modifications of the Wheeler pan; the earliest continuous grinding pan is probably that of M. P. Boss.

The Denny Bros. pan,[1] shown in Figs. 400–403, is 5 ft. diam., has a set of 18 cast-iron shoes, 3 in. thick, attached to a muller plate by means of eye-rings and wooden keys, also 18 corresponding dies. The machine makes 57

[1] Allen, R., "West Australian Metallurgical Practice," Kalgoorlie, 1906, p. 11.
Denny, G. H., and H. S., *Tr. S. Afr. Assn. Eng.*, 1905–06, XI, 211.

r.p.m. and requires 6.5 h.p. Shoes and dies last eight weeks with raw and six weeks with roasted ore. In order to correct the loss in weight due to wear of shoes, one or two compensating rings weighing about 600 lb. are slipped over the spider and rest upon the muller. The work accomplished is shown in Table 183.



FIGS. 400–403.—Denny continuous grinding pan with compensating weights.

TABLE 183.—WORK OF GRINDING PANS IN GOLD MILLING PLANTS OF WESTERN AUSTRALIA[1]

Location	Size of pan, feet	Capacity, tons 24 hr.	Horse-power	Revolutions per minute	Remarks
Ivanhoe Mill	5	20	6.5	57	With amalgamation.
Assoc. Northern Gold Mine.	5	8	4.8	47	With amalgamation.
Sons of Gwalia Gold Mine	5	33	5.3	45	With amalgamation.
Great Boulder Perseverance.	8	12.6	9	31	With amalgamation.
	8	8.4	6	27	With amalgamation.
Great Boulder Propietary	5	15	4.5	62	With amalgamation.
Assoc. Gold Mine	5	16.5	5.2	47	Without amalgamation.
Lake View Consol. Gold Mine.	5	30	7	45	Without amalgamation.
Great Fingall Gold Mine.	5	33	5	45	Without amalgamation.

[1] Allen, *op. cit.*, pp. 12, 32, 44, 70, 95, 114, 128, 162.
Richards, "Ore Dressing," III, 1284.
Jenson, *Eng. Min. J.*, 1913, XCV., 952.

A sizing of the ground pulp of the Ivanhoe mill with and without the use of compensating rings gave the data of Table 184.

TABLE 184.—GROUND PULP OF IVANHOE MILL[1]

Mesh	Grinding 18 tons per day		Grinding 22 tons per day	
	With compensating weights, per cent.	Without compensating weights, per cent.	With compensating weights, per cent.	Without compensating weights, per cent.
on 40	....	....	0.1	0.5
on 60	0.3	3.5	1.7	16.7
on 100	11.7	48.5	24.2	50.2
on 150	11.7	10.2	18.5	8.9
th. 150 on ...	76.3	37.8	55.5	23.7

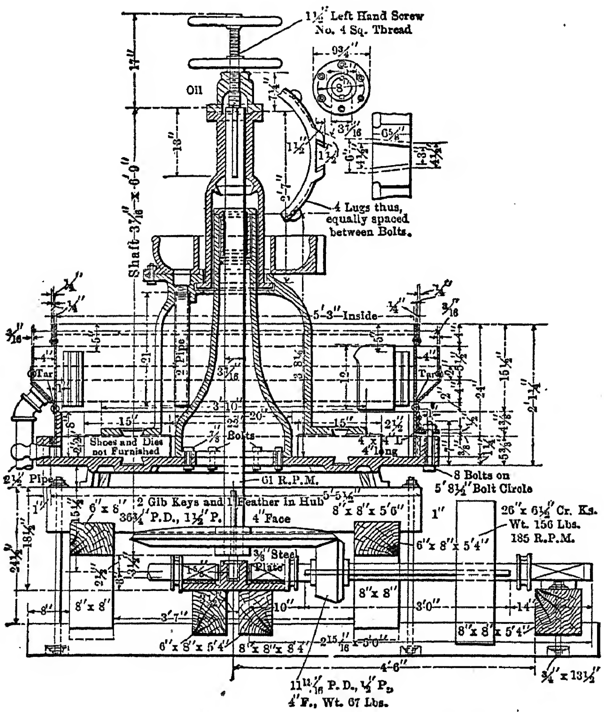


FIG. 404.—Colorado Iron Works continuous grinding pan.

The continuous grinding pan of the Colorado Iron Works, Denver, is shown in Figs. 404 and 405. It is 5 ft. diam., 2 ft. deep, weighs 7,500 lb. The arrangement of shoes, dies, wings, etc., is the same as in the combination pan. The pulp is fed into the annular trough attached to the spider, passes downward through four pipes to the inner edges of the muller, is ground between shoe and die, and overflows over the edge of the curb into the launder surrounding the pan. The work of this pan and that of two tube-mills at the Homestake mine is given in Table 185.

[1] Allen, *op. cit.*, p. 12.

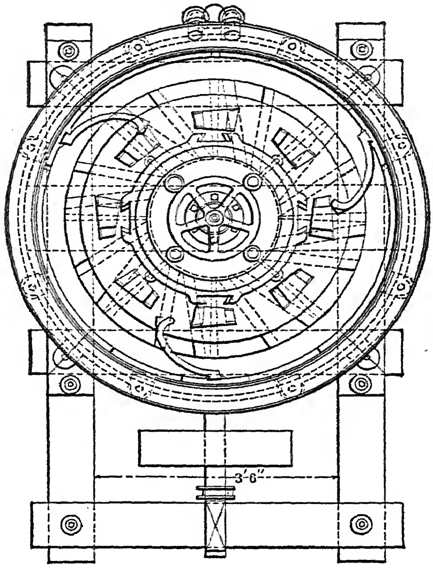


FIG. 405.—Colorado Iron Works continuous grinding pan.

TABLE 185.—WORK OF GRINDING PAN AND TUBE-MILL AT HOMESTAKE MINE[1]

	5-ft. grinding pans		5×14 ft. tube-mill		5×14 ft. tube-mill	
	12,308 tons ground by 7 pans		Regular adjustment, medium feed		Special adjustment, heavy feed	
ɔtal tons ground per day....	19.34 per pan		73		110	
ɔns ground per day to pass 200-mesh sieve.	10.83 per pan		43		52.8	
'ater in feed, per cent......	80–90		38.0		38.4	
	Heads	Tails	Heads	Tails	Heads	Tails
ssay, gold value per ton....	$2.66	$2.07	$2.49	$2.04	$2.49	$2.04
zing test, per cent. on 50-mesh.	47	6	39	5	18	7
Between 50- and 80-mesh.	34	14	38	12	49	15
Between 80- and 100-mesh.	9	14	12	13	17	14
Between 100- and 200-mesh.	6	26	7	28	11	26
Through 200-mesh......	4	40	4	42	5	38

[1] Clark-Sharwood, *Tr. Inst. Min. Met.*, 1912–13, XXII, 68; *Min. Eng. World*, 1912, CXVII, 1091.

TABLE 185.—WORK OF GRINDING PAN AND TUBE-MILL AT HOMESTAKE MINE.—*Continued.*

Tons ground per horse-power per day at one passage through grinder.	2 30	1.94	2 92
To pass 100-mesh sieve...	1.31	1.14	1 40
To pass 200-mesh sieve...	0 83	0 74	0.97
Material consumed, pounds per ton.	Iron, worn 3.41 = 80%, Scrapped 0.82; Total 4.23lb.	Pebbles 1 66	Pebbles 1.30

The results of comparative tests with pan and tube will show that the pan is more economical in power than the tube-mill, but that if from 30 to 40 per cent. more sands are fed to the tube-mill, this disadvantage is neutralized. No new pans will be introduced at the Homestake mine, and the old ones will be replaced in time by tube-mills.

269. Roller Mills.—These are machines in which rolls moving in a circular enclosure crush ore by pressure or by pressure and abrasion. There are two general types.

(A) Radial roller mills, in which vertical rolls, with their axes coinciding with the radius of the horizontal enclosure, exert the necessary force mainly by the weight of the rolls. The center of a roll crushes by pressure; the sides sliding upon the ore crush by pressure and abrasion. Either the rolls move around a central axis and the enclosure (pan) is stationary, as in the Chile, Bryan, Lane, etc., mills, or the axes of the rolls are fixed and the pan rotates in a horizontal plane causing the rolls to revolve as in the edge-runners, Fig. 232, commonly used for crushing refractory materials.

(B) Centrifugal roller mills, in which rolls suspended from a rapidly rotating disk or pulley swing outward and, revolving against a steel ring, crush the ore by the pressure of the centrifugal force. The Huntington and Griffin mills may serve as examples.

1. THE CHILE MILL.—This was originally used to prepare ore for the arrastra. Thus at Guanaxuato,[1] in 1882, a single roller of iron or stone, 5 1/2 ft. diam. and 1 1/4 ft. wide, bound by an iron tire, of the same width as the roller and 4 in. thick, rotated on a horizontal shaft, one end of which was attached to a vertical spindle in the center of the mill while the other had arrangements to hitch on mules. The wheel ran in a gutter 1 ft. 7 in. wide paved with iron. Between the gutter and the central spindle was a conical screen covered with brass wire-cloth with holes 0.5 to 0.6 in. square. The mill crushed 8 to 10 tons of ore in 24 hr. to be ground in the arrastra. A water-propelled mill of similar primitive form used at Cerro de Pasco for fine-grinding has been described by Pfordte.[2] It has a single stone roller, 8 ft. 2 1/2 in. in diam. and 1 ft. 6 in. face,

[1] Rul, *Eng. Min. J.*, 1882, XXXIII, 104: see also *Harper's Magazine*, Feb., 1907; illustration in *Mining World*, 1908, XXIX, 522.

[2] *Tr. A. I. M. E.*, 1894, XXIV, 107.

makes 12 r.p.m. on a bed 5 ft. 6 in. in diam. and crushes fine 1.5 tons ore in 24 hr. Fig. 406[1] represents a more modern Chile mill once used for fine-crushing at the Haile gold mine, S. C. It had two rollers, weighing about 1 ton; they were 4 ft. in diam. and 8 in. face, had white-iron tires 8 in. thick, and made 40 r.p.m. The capacity in 24 hr. was 90 tons of 1/4-in. tough quartzite to 40-mesh pulp, and the wear 12 lb. iron per ton of ore. A sizing-test gave on 40-mesh, 8.30 per cent.; 40- to 60-mesh, 5.10 per cent.; 50- to 60-mesh, 1.09 per cent.; 60- to 70-mesh, 3.32 per cent.; 70- to 80-mesh (?); 80- to 90-mesh, 1.66 per cent.; 90- to 100-mesh, 1.66 per cent.; through 100-mesh, 79.98 per cent.; the large amount of the last product illustrates the grinding action of the machine.

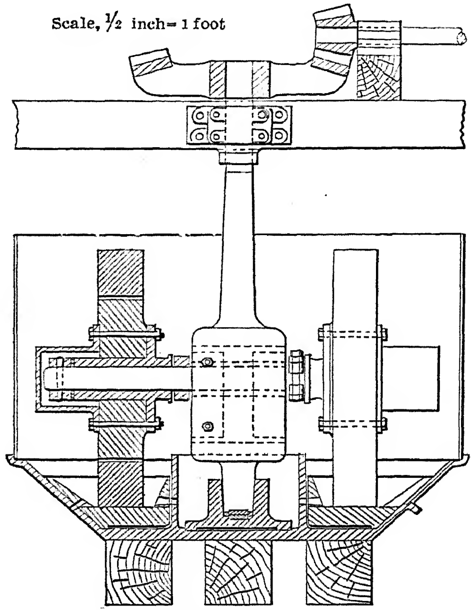
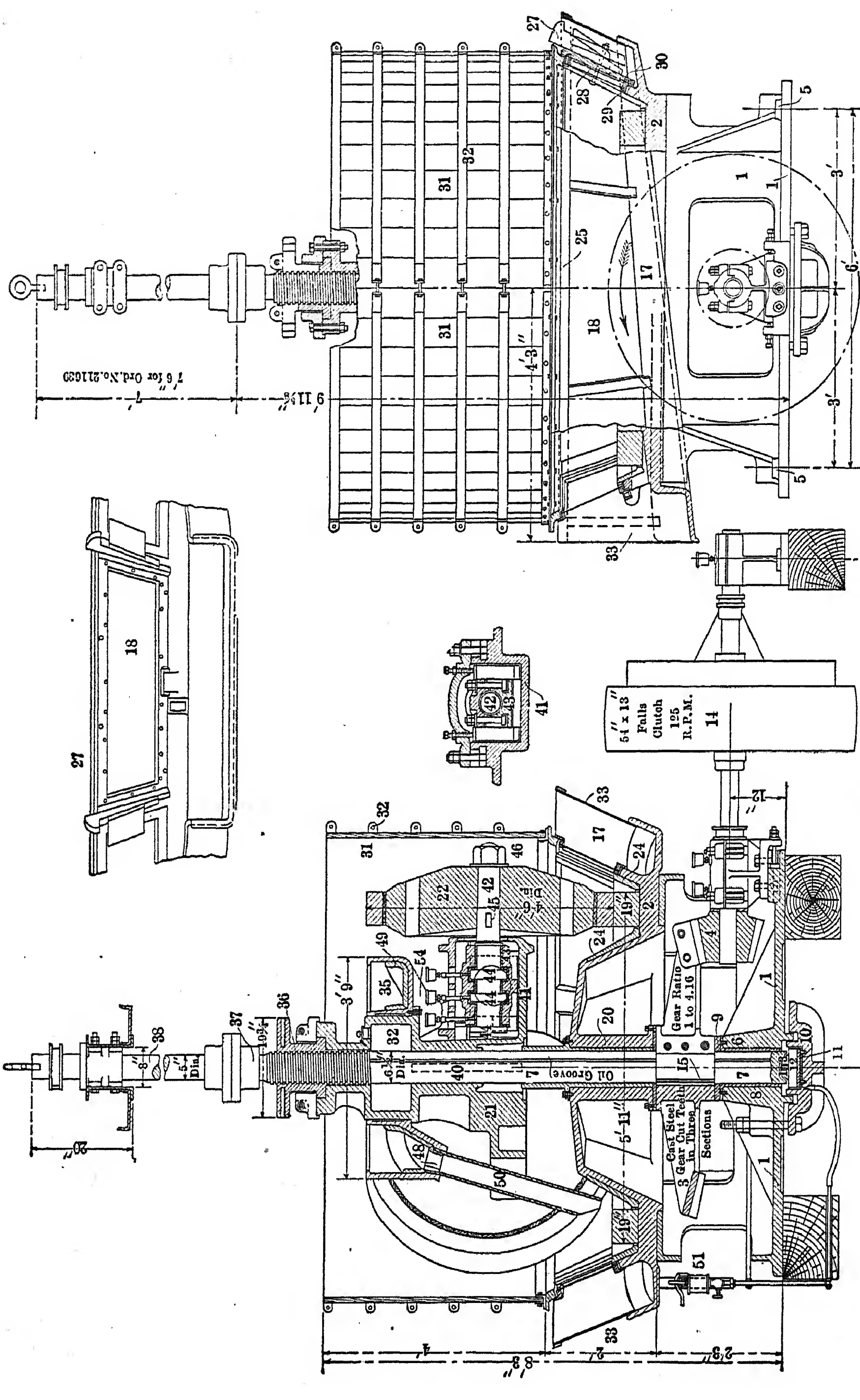


FIG. 406.—Vertical section of Chile mill.

The progress that has been made in recent years in the construction of Chile mills is well illustrated by Figs. 407 and 410 which represent a 6-ft. Cole mill with three rollers as made by the Allis- Chalmers Co. Mills of this type are common in ore-dressing plants; in metallurgical works they are used mainly for re-crushing, to say 40- or 60-mesh, the first tailings from stamped and concentrated silver and gold ores in order to either set free particles from included grains for recovery by further concentration, or prepare the material for the tube-mill. In the drawings, the cast-iron base 1, which carries mortar 2 and supports driving gears 3 and 4, is provided with four holes for foundation bolts 5; in the center it has circular bearing 6, about 13 in. long, for spindle 7, lined with bronze bushings 8, held in place by a flange and countersunk screw; the bushing carries bronze friction-washer 9. The bottom of the central opening is closed by an adjustable cast-iron step-box 10, carrying steel disc 11, held in place by dowel pins and bronze disc 12, with oil-grooves which support steel toe 13 of a ball-and-socket joint, the toe rotating with the spindle. The driving gear consists of cast-steel bevel gear 3 clamped to spindle 7 and bevel pinion 4, the shaft of which, 3 1/16 in. diam. and 7 ft. 6 in. long, carries driving pulley 14 (54 in. diam. and 12 in. wide) which is of the friction-clutch type. Between the mortar and bevel gear 3 is a bronze bushing 15. The pulley makes 125 r.p.m. and with a gear ratio of 1 : 4.16 causes the spindle to rotate at the rate of 30 r.p.m. The cast-iron mortar 2 with inclined launder 17 is 8 ft. in diam. and 2 ft. high. It has five openings, 18 for screens. It is made 4 in. thick beneath dies 19 and is strengthened by ribs in the center. Here the bearing 20 for the spindle

[1] Richards, "Ore Dressing," Vol. 1, p. 270.



FIGS. 407–410.—Cole 6-ft. Chile mill of Allis-Chalmers Co.

is 2 ft. 4 in. long and, reaching up into spider 21 carrying rollers 22, prevents grit or water from entering. The inside bottom surface of mortar 2, on which ring-die 19 rests, is planed. The die of rolled steel is 4 ft. 9 in. inside diam., 7 in. wide and 5 1/2 in. thick. On either side of the die the mortar has white-iron liners 24. A screen-frame consists of an outer part of cast-iron with flange to rest against the outside of the mill and to be held in place by wrought-iron keys 27, a central part 28 of wood, and an inner one 29 of wrought iron, the whole held together with carriage-bolts 30. The mortar has a housing 31 of wood tied by wrought-iron bands 32, the launder a splash-apron 33 of sheet metal. The spindle or shaft, 7, 6 1/2 in. in diam. and about 10 ft. long, of hammered steel carries spider 21, for driving the three rollers 22, and feed trough 35. At the point where the driving gear is placed, the diameter of the spindle is reduced to 6 in. to prevent the shaft from moving vertically when the gearing is in place. The upper part is provided with a 29-degree Acme thread of 3/4 in. pitch to match a bronze nut 36 for the adjustment of the driving spider.

The top is provided with a flanged face coupling 37 for attaching an extension shaft 38, 5 in. in diam. and 7 ft. long, which is held between channel irons furnishing an upper bearing. The driving spider 21, of cast-iron and ribbed to give additional strength, is movably attached to the shaft by means of feather-key 40 (groove and spline) to permit vertical adjustment. It has three arms 41 for the three roller shafts 42. An arm has two bearings for trunnions 43, 8 1/4 in. in diam. and 3/4 in. below the shaft, by means of which a roller-bearing shaft is supported to allow for the rise and fall of the roller in its passage through the pan. A roller-shaft, tapering at the end, is 5 in. in diam. in the roller and 3 ft. 2 in. long; it further has three collars 44, 8 in. in diam., that it may present the necessary wearing surface to sustain centrifugal force without becoming heated. A feather key 45 is fitted to the shaft and roller to assist in driving; roller 22 of cast-iron is held in place by a heavy nut 46 which, when drawn tight, wedges securely the shaft upon the roller. The feed-trough 35, having three discharge-openings 48, is an annular box fastened to the spider. It is of cast iron, the bottom and lower sides are lined with wrought iron 49 to take up the wear; each discharge-opening has a wrought-iron pipe 50, 4 in. in diam., which delivers the pulp upon the ring-die in front of its roller. In the center of each a hole is bored through to fit the tapered shaft; the body has a number of holes cast in to facilitate handling, and the circumference is tapered for the reception of wooden wedges used to hold the steel tire in place. This has an outside diameter of 4 ft. 6 in., a 7-in. face, and is 3 3/4 in. thick.

The lower bearings of the vertical shaft of the step-box are oiled by means of pipes supplied from the hand oil-pump 51. The center bearing is supplied with oil from a cavity 52 in the spider by means of oil-grooves 53; the roller-bearings each have three grease-cups 54, and the counter-shaft bearings two.

The mill is rated at 50 tons per 24 hr. when handling material averaging 3/4 in. size and reducing to 30- or 40-mesh.

TABLE 186.—WORK OF CHILE MILL

Location	Bed-plate diameter, inches	Die width by thickness, inches	Rollers					Power	Ore		Water, tons per ton ore	Product, mesh	Capacity, tons 24 hr.	Authority
			No.	Diam. by face, inches	Weight, pounds	Tire thickness, inches	Rev. per minute		Character	Size, mesh				
Guanaxuato	......	19×?	1	66×15	......	4	.....	Horse	Siliceous silver	..	....	0.5–0.6	8–10	(a)
Cerro de Pasco..... ...	66	..	1	86.5×18	......	...	12	Water	Siliceous silver	....	......	fine	1.5	(b)
Haile, S. C...	48	12×4	2	48×8	2,000	8	40	... .	Quartz	0.25 in.	.	40	90	(c)
Union, Mexico..... ..	70	16×?	2	96×16	8,000	..	10	10 h.p.	Siliceous silver	1.5 in.	10	80 % <20	15–20	(d)
Santa Elena, Mexico. .	....	...	..	... .	......	.... .	15	15 h.p.	Siliceous silver	<2 in.	8–10	40	25	(e)
Amboy, Cal	84	5×?	4	42×5	1,200	3	10	.. ..	....	.. . .	.. ....	..		(f)
Beresovsky, Russia.. ..	90	?×3	2	66×10	5,400	9–10	12	7–12 h.p.	Siliceous gold	1 in.	10	1 mm.	16	(g)

(a) *Eng. Min. J.*, 1882, XXXIII, 104; (b) *Tr. A. I. M. E.*, 1894, XXIV, 107; (c) Richards, "Ore Dressing," I, 269; (d) *Eng. Min. J.*, 1908, LXXXVI, 989; (e) *Op. cit.*, 1910, XC, 967; (f) *Op. cit.*, 1911, XCII, 305; (g) *Tr. Inst. Min. Met.*, 1910–11, XX, 125.

2. THE BRYAN ROLLER QUARTZ MILL.[1]—Fig. 411 represents one form of this mill with screens and part of casing removed. It is suitable for wet-crushing, with quicksilver, gold ore that is to be amalgamated on an apron-plate. Ores with a soft gangue (decomposed schist or slate with little quartz) and with gold present in a loose free state, appear to give better results than quartzose ores with finely divided particles of gold.

The apparatus consists of a cast-iron pan, 4 to 5 ft. in diam., with three discharge-screens, a stationary center-post to the top of which are fastened branching supports for carrying the two boxes of the horizontal driving shaft. Its pinion drives the horizontal bevel gear; this is attached below to an annular plate to which are journaled three steel-tire rollers; they travel on a sectional die-ring; the rollers distributing the pulp. In a 5-ft. pan a roller is 44 in. in diam., 7 in. wide and weighs 3,650 lb.; the steel tire weighs 1,100 lb.; the annular plate makes 40 r.p.m.; the mill requires 10 h.p., 400 to 1,000 gal. water, and treats 25 to 35 tons ore in 24 hr. A 4-ft. mill with 60 r.p.m. requires 5 h.p., 300 to 750 gal. water, and treats 12 to 20 tons ore.

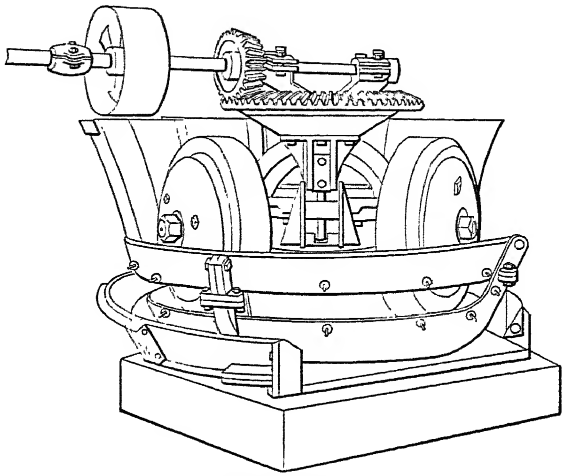


FIG. 411.—Bryan roller quartz mill.

3. THE LANE MILL.[2]—This is a low-speed edge-roller mill, made by the Standard Iron Works of San Francisco, for fine-crushing and amalgamating gold ore crushed by rolls or stamps. It has six rollers, 42 in. in diam. and 5 in. face, with steel-faced wooden centers and steel tires 2 1/2 in. thick, attached to a spider with 6 arms which is supported in the center by ball-bearings and weighted by an overhead tank charged with 6 tons of scrap iron. The rollers travel 8 r.p.m. in a pan of riveted steel upon a track 10 ft. in diam., 2 1/2 in. thick and 5 1/4 in. wide made in sections of rolled steel. The arms of the spider are attached to a driving gear 12 ft. in diam.; 8 to 12 h.p. are required. Six pipes deliver the sands in front of the rollers.

At the Bonita mill, Baram, Wash., six runners weighing 900 lb. each making 6 r.p.m. and requiring 12 h.p. crushed in 24 hr. 40 tons 1/4-in. ore through an

[1] Risdon Iron Works, San Francisco.
Preston, *Bull.* No. 6, State Mineralogist, Cal., 1895, p. 62; *Min. Sc. Press*, 1895, LXX, 193.
Tays, *Tr. A. I. M. E.*, 1899, XXIX, 776, 1054.

[2] Lane, *Min. Rep.*, 1907, LV, 515.
Stewart, *op. cit.*, p. 543.
Editor, *Eng. Min. J.*, 1908, LXXXV, 1053.
Slow-speed Mills: Empson, *Tr. Mex. Inst. Min. Met.*, 1910–11, II, 173.
Denny, *op. cit.*, 1912–13, III, 25.

8-mesh screen with a 6-in. discharge. A sizing test of the pulp after having been treated on a Wilfley table is given in Table 187.

TABLE 187.—SIZING-TEST OF PULP FROM LANE MILL AFTER CONCENTRATION

Passing through, -mesh	30	40	60	80	100
Tailing, per cent	96	88	66	56	36
Concentrate, per cent	99	96	92	88	80

4. HUNTINGTON CENTRIFUGAL ROLLER MILL.[1]—The mill serves a similar purpose as the Bryan mill (see above). Fig. 412 represents a perspective view of one of the mills made by Fraser and Chalmers. The cast-iron pan *C* with

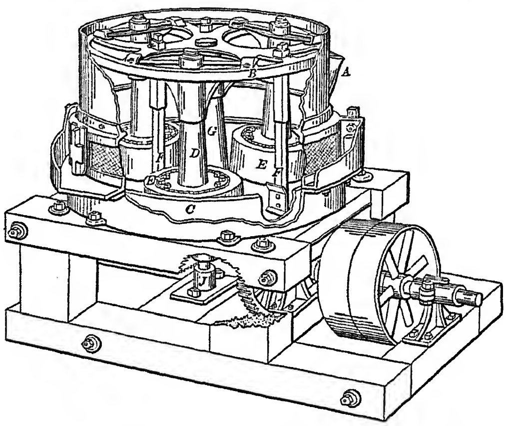


FIG. 412.—Huntington centrifugal roller mill.

central cone *G* carries the housing which is made in two sections; this contains the feed-hopper *A* and the discharge-screens which are 9 in. wide and occupy half of the circumference. The pan is mounted on a wooden frame. The driving shaft, pivoted in *J* and passing through cone *G*, receives its motion from below through a bevel gear, shaft and pulley. To its upper end is keyed a frame *B* which carries three pairs of boxes for the trunnions of the sleeves from which are suspended the shafts of the three steel-tire rolls *E* which revolve freely around their shafts. These are inclined so that the roller is 1/4 in. above the bottom at the outer edge and 1 3/4 in. at the inner. When rotated, the rollers are thrown outward radially from the center-shaft against the circular steel die of the pan, and by the pressure of the centrifugal force crush in water the ore

[1] Preston, *Bull.* No. 6, State Min. Bureau, 1895, p. 60.

Parson, *Trans. Inst. Min. Met.*, 1905–06, XV, 587.

Report State Min. Cal., 1887, VI[2], 51 (Spanish Mine), 1889, VIII, 144 (Quaker Mine), 442 (Spanish Mine), 670 (Golden Treasure Mine).

Eng. Min. J., 1888, XLV, 324 (Spanish Mine).

(not >3/4 in. in diam.) to the required size; they gyrate around the central shaft and revolve around their suspension rods. The bottom of the pan is covered with an annular cast-iron disk which takes up the wear.

Scrapers *F* keyed to the frame turn over the pulp and drive it toward the die. The pulp, when sufficiently fine, passes through the screens, is collected in an annular space and delivered to an apron plate. Quicksilver is added to the mill in the usual way; amalgam collecting on the bottom is removed at the regular clean-up.

TABLE 188.—DETAILS OF THE HUNTINGTON CENTRIFUGAL ROLLER MILL[1]

Diam., feet	Rollers, No.	Rev. per minute	Horse-power	Ore fed; size, inches	Product; size, mesh	Capacity 24 hr., tons	Water, gallons per hour	Iron, con-sump-tion
3½	3	90	4	¾	30	12	750	.....
5	3	70	6	¾	30	20	1,000–1,200	......
6	3	55	8	¾	30	30	1,400–1,700	... .
3½	4	90	5–7	¾	30	8–12	750	......
5	4	70	8–10	¾	30	20–25	1,000–1,200	....
6	4	55	10–14	¾	30	40–50	1,400–1,700	....

The mill is used as a dry pulverizer in cement plants for crushing rock and clinkers through a 20-mesh screen preparatory to fine-grinding in the tube-mill.

5. THE RAYMOND FOUR-ROLLER MILL, Fig. 413, works on the same general principles as the Huntington mill. The details are easily seen from the figure and the legend. At the smelting works of the Canadian Copper Co., Copper Cliff, Ont., two of these machines crush in 10 hr. 50 tons of coal containing 10 per cent. ash. The coal is cracked, dried in a Ruggles-Cole dryer and fed to the Raymond machines that deliver a product of which nearly the whole will pass a 200-mesh screen; the dust serves as fuel for reverberatory smelting furnaces.[2]

The Fuller-Lehigh pulverizer mill[3] used at Lebanon, Pa., for the production of fuel dust to be used in heating for nodulizing magnetite concentrates works on a similar principle, hardened balls revolving in cast metal races at high speed. It is said to crush from 3.5 to 4 tons of coal per hr. to a fineness of 95 per cent. through a 100-mesh screen with a consumption of from 32 to 35 h.p.

6. THE GRIFFIN ROLLER MILL (Fig. 414).—A single suspended roller 31, gyrating in a cast-iron pan provided with liners 70, crushes dry or wet ore fed at one side and discharges the fine material through a screen into a pocket beneath the pan, whence the pulp is removed by a screw-conveyor. Power is received through pulley 17 which has two journals 27 and 26 attached above and

[1] For other data, see Schwerin, *Eng. Min. J.*, 1904, LXXVII, 403.
Raymond, *Inst. Chem. Eng.*, 1913, XI, 108.

[2] See Hofman, "Metallurgy of Copper," 1914, p. 267.

[3] Ennis, *Eng. Mag.*, 1907–08, XXXIV, 474.
Meade, *Tr. Am. Inst. Chem. Eng.*, 1909, I, 109.

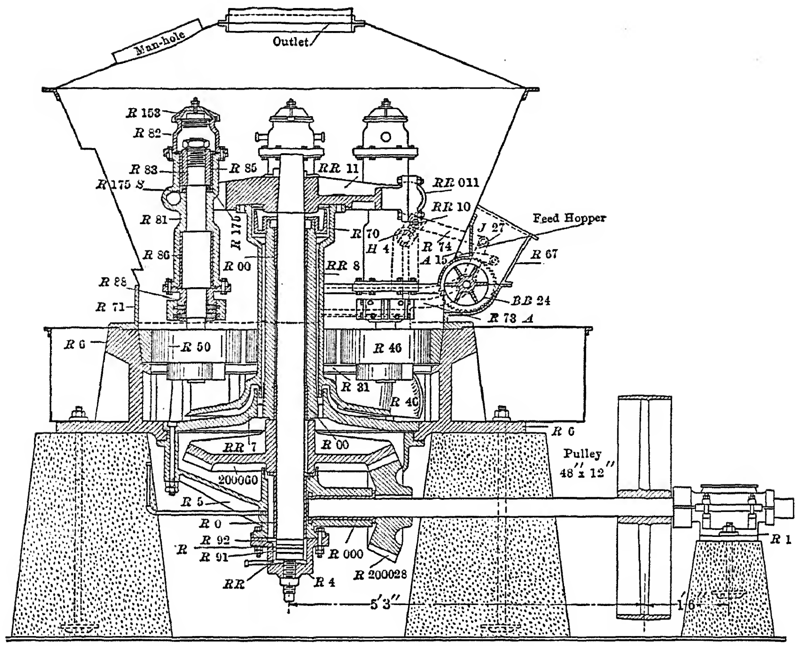


FIG. 413.—Raymond four-roller mill.

R-1	Pillow block.
R-4	Step box for vertical shaft.
R-5	Lower journal for vertical shaft. Gear housing.
R-6	Mill base.
RR-7	Mill bottom and support for vertical shaft.
RR-8	Plow support.
R-10	Bronze bushing in crank and pitman.
RR-11	Spider supporting roller journals.
RR-011	Cap for spider trunnion bearings.
R-174	Half bushing in cap RR-011.
R-31	Plow-tip holder.
R-46	Plow tip.
R-48	Wedges for holding down bull ring.
R-50	Chilled-iron roll.
R-67	Front of feed hopper.
R-69	Feed hopper sides, right and left.
R-70	Oil reservoir for lubricating vertical shaft.
R-71	Separator support.
R-73-A	Bracket holding feed drive, right and left.
R-73-B	Cap for feed roll-shaft bearing.
R-74	Pitman.
R-184	Feed-roll guard, roll 8 inches diameter.
F-31	Door on separator.
R-64	Door frame.
R-152	Feed roll, 8 inches diameter.
BB-24	Ratchet wheel.
J-27	Ratchet lever.
H-4	Crank.
A-15	Pawl.
R-O	Bushing for vertical-shaft bearing in R-5.
R-00	Bushing for vertical-shaft bearing in RR-7.
R-000	Bushing for horizontal-shaft bearing in R-5.
R-92	Steel button on end of shaft.
R	Bronze button in step bearing.
R-91	Steel button in step bearing.
RR	Bronze button in step bearing.
R-81	Housing for roller journal.
R-82	Grease cap.
R-83	Top bearing, when threaded.
R-183	Top bearing, when put on with taper and key.
R-85	Bushing for upper journal.
R-86	Bushing for lower journal.
R-88	Stuffing box.
R-130	Lock nut for doors.
R-131	Pitman.
BB-24	Ratchet wheel.
A-15	Pawl for ratchet.
J-27	Ratchet lever.
H-4	Crank for feed hopper.
R-170	Stuffing-box cover.
R-171	Stuffing-box cover.
R-153	Grease-plug cover.
R-90	Trunnion-shaft washers.
R-175	Bronze carrying ring.
R-175-S	Steel carrying ring.
200060	Bevel gear on vertical shaft.
200028	Bevel pinion on horizontal shaft.
	Bullring, 50 inches inside diameter.

below. The upper one with its journal-pin revolves in an extension of one of the standards, the lower one in a tapered adjustable bearing 21 which is carried by the three standards.

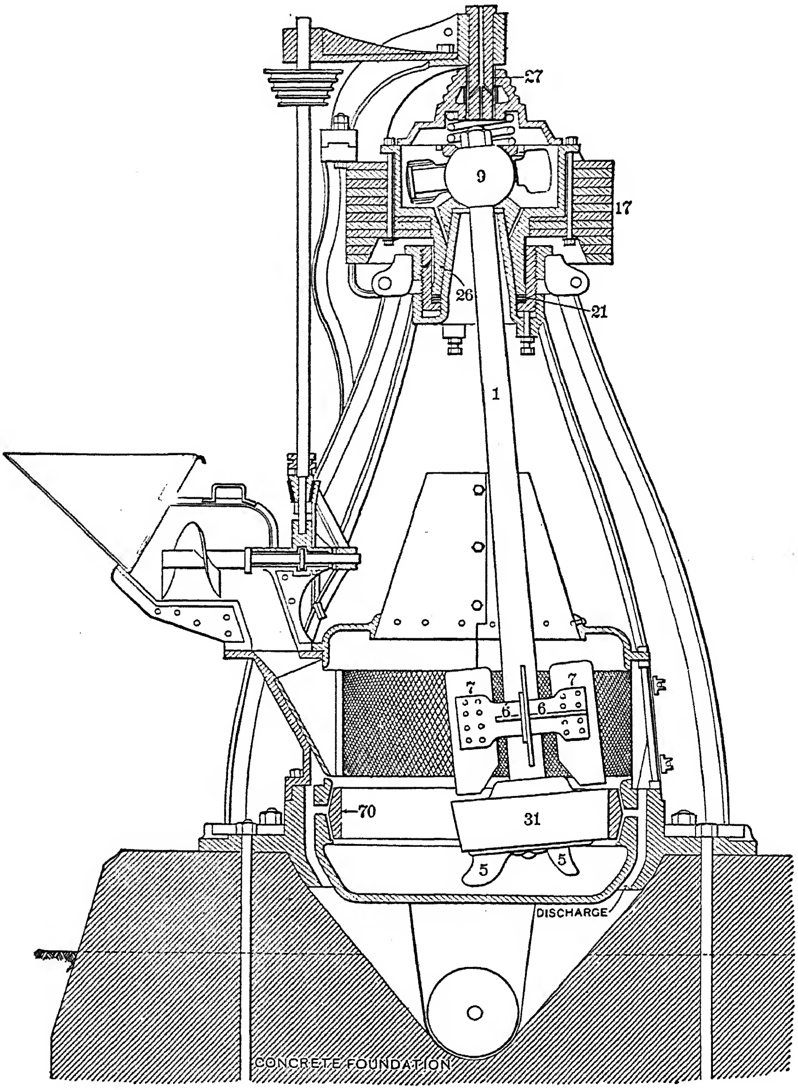


FIG. 414.—Griffin roller mill.

Shaft 1 which carries roller 31 and passes through the lower journal, is suspended by means of universal joint 9 in the center of the pulley; the joint consists of a sphere with trunnions which rock up and down in recesses of the pulley. Under roller 31 are plows 5 to keep the ore stirred up, and above it fans 6 and 7, used in dry-crushing to draw in air through the top of the sheet-iron cone and force it through the screens which are mounted on the pan and surrounded

by a housing. The ore, introduced through a hopper at the left, is fed by means of a screw, driven from a step-pulley above, to fill the pan up to plows 5 which, stirring it up, throw it against die 70 so that it will be acted upon by roller 31.

When in full operation, the ore is whirled around in the pan and crushed; the fine particles are driven through the screen and the coarse ones fall down to be again acted upon. Roll 31 is revolved in the same direction as shaft 1 is driven, but when in contact with the die it gyrates backward thus acting in two different ways upon the ore.

Table 189.—Details of Griffin Roller Mill

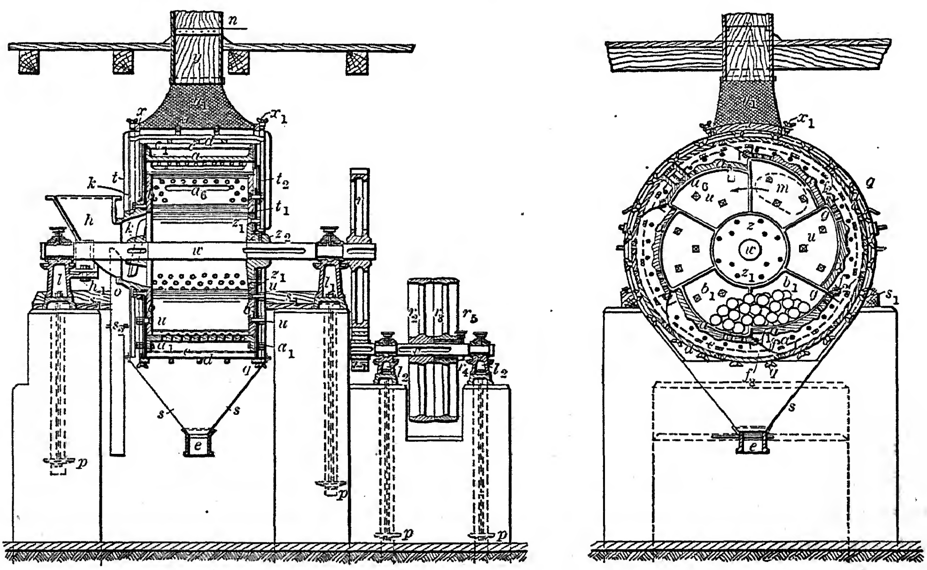
	30-in. mill	36-in. mill
Total height above foundation	8 ft. 2½ in.	8 ft. 7½ in.
Total width	5 ft. 3 in.	6 ft. 3 in.
Height, top of foundation to center of pulley	6 ft. 4½ in.	6 ft. 6½ in.
Total weight	10,500 lb.	14,500 lb.
Pulley, revolutions per minute	190–200	135–150
Horse-power	15–25	25–30
Diameter of pulley	30 in.	40 in.
Diameter of roll	18–20 in.	22–24 in.
Diameter of ring or die	30 in.	36 in.
Depth of contact surface on roll	6 in.	6 in.
Weight of ring or die	260 lb.	408 lb.
Weight of tire	100 lb.	175 lb.
Pressure of roll against die	6,000 lb.	8,000 lb.

Table 190.—Work of Griffin Roller Mill

Location	Material	Mill			Ore			Reference
		Diam. of die, in.	Rev. per min.	Horse-power	Capacity 24 hr., tons	Size of feed, in.	Size of product, mesh	
Great Boulder Perseverance, W. Australia.	Schistose gold ore.	30	198	18	37	½	14*a*	Allen, West-Australian Metallurgical Practice, 1906, pp. 65 and 89.
Great Boulder Proprietary, W. Australia.	Schistose gold ore.	30	200	20	30	½	15*b*	
?	Portland cement clinker.	30	200	25–28	24–36	½–¾	90–95 % through 100	Bradley Pulverizer Co.
?	Limestone...	30	200	25	48–60	½–¾	90 % through 100	
?	Bituminous coal.	30	200	20	48	1	95 % through 100	

Table 191.—Screen Analysis of Griffin Roller Mill Product

	Perseverance ore			Proprietary ore	Ball mill product of Proprietary ore
Screening	14-mesh of 22-gauge wire	10-mesh of 21-gauge wire	10-mesh of 20-gauge wire	15-mesh of 21-gauge wire	30-mesh of 28-gauge wire
	Per cent.	Per cent.	Per cent.		
Through on 20	0.30	3.50	2.50	....	....
Through 20 on 30	1.00	5.15	4.00	1.07	0.22
Through 30 on 40	2.20	4.75	4.50	1.01	5.21
Through 40 on 60	5.50	8.75	7.15	3.36	15.74
Through 60 on 80	5.35	5.25	5.35	4.60	6.64
Through 80 on 100	4.15	4.00	4.85	5.99	8.07
Through 100 on 120	2.00	2.65	2.25	1.27	1.67
Through 120 on 150	7.35	5.00	6.00	10.60	6.32
Through 150 on 200	4.05	2.15	1.65	4.38	5.80
Through 200........	68.10	58.80	61.75	67.72	50.33



Figs. 415 and 416.—Gruson or Krupp ball mill.

270. Ball Mills.[1]—These machines are characterized by a short revolving horizontal cylinder in which the ore is pulverized by the blows, the grinding

[1] History: Sell, *Dingler Pol. J.*, 1893, CCCVI, 38, 59, 83.
Ball, *Tr. Inst. Min. Met.*, 1911–12, XXI, 2.

and the pressure of chilled cast-iron or steel balls or cylinders upon one another and upon a die-ring through which the pulp is screened out as fast as it is formed. They can be run dry or wet, are compact, give a finished product low in slime or dust, and are suitable where small units are required. The Gruson, Gates and Tustin mills may serve as examples.

1. The Gruson or Krupp Mill (Figs. 415 and 416).—This consists of a stationary sheet-iron housing *s*, with feed *h* at one side, off-take *y* for dust at the top and discharge *e* for the pulp at the bottom. Inside revolve with the horizontal shaft *w*, the die-ring *a* carrying the crushing balls, the enclosing coarse perforated sheet-steel cylinder *c* which takes up the wear, the concentric wire-gauze screen *d* which sifts out the pulp, and the discs *t* which form the sides. The die-ring is composed of five overlapping chilled-iron plates *a* which are perforated at the front, and thickened and bent spirally at the back to form steps for the balls to drop, and to furnish open spaces for the return of the oversize from the screens. The coarse screen *c* consists of five sections with spaces between. The fine screen *d* usually is cylindrical; in some cases, as in Fig. 417, the straight fine sieve has been arranged in step form to assist the screening of ores that do not pass readily through the meshes of the surface, curved in the ordinary way. The ore, crushed to egg size, is charged into the hopper *h* and fed from this by the two-blade propeller *k* into the cylinder where it is pulverized by steel balls which are of mixed size in order to increase the contact surfaces.

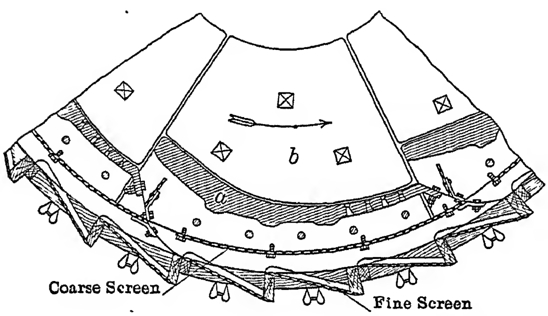


Fig. 417.—Improved screen of Gruson or Krupp ball mill.

The ore passes through the openings in the dies, falls on to the punched screen *c*, which retains medium-sized particles, and the gauze-screen which allows the fine pulp to pass through; the oversize from both screens is returned by the deflectors *f* to the grinding center; dust is drawn off through *y* into a settling or filtering chamber. Access to the center is obtained through the manhole *m* in the side. In order to permit the removal of metallics there is provided a slot a_6, which passes through the screens and die-ring and is smaller than the diameter of the balls. Removing the cover and allowing the drum to make several revolutions causes the metallics to drop out. The mill is made in different sizes. Mill No. 4 is 6 ft. 3 in. in diam., 3 ft. 3 in. wide, takes 1,540 lb. of balls; No. 5, 7 ft. 5 in. in diam., 3 ft. 10 in. wide, takes 2,240 lb. balls; No. 8, 8 ft. 10 in. in diam., 4 ft. wide, takes 4,400 lb. balls.

2. The Gates Mill, made by the Allis-Chalmers Co., is shown in cross-section in Fig. 418. The principle is the same as the Gruson mill.

TABLE 192.—WORK OF DRY-CRUSHING GRUSON BALL MILL

Locality	Ore	Mill No.	Rev. per minute	Tons, 24 hr.	Product; size, mesh	Iron wear, pounds per ton of ore	Horse-power	Reference
Atacama, Chile	Tough quartzose, gold ore.	4	25	6.5	80	5.7	11	Loram, *Tr. A. I. M. E.*, 1899, XXIX, 497.
Mt. Morgan, Queensland	Hard sulphide ore	5	....	23	20	0.75	13	White, *Tr. Austr. Inst. Min. Eng.*, 1900, VI, 37.
Mt. Morgan, Queensland	Hard sulphide ore	4	....	19	35	0.75	10	White, *Tr. Austr. Inst. Min. Eng.*, 1900, VI, 37.
Oruro, Bolivia	Slate	4	....	40–50	25–30	....	11	Basadre, *Eng. Min. J.*, 1895, LX, 440.
Associated Gold Mine	Schist	5	25	40	30	0.25	16.8	p. 111 Allen, R., "West Australian Metallurgical Practice," Kalgoorlie,[1] 1906.
Kalgoorlie	Schist	5	24	40	32	....	25	p. 52
South Kalgoorlie	Schist	8	24	110	30	0.5	57	p. 101
South Kalgoorlie	Schist	5	25	40	30	0.5	35	p. 101
Great Boulder	Schist	8	23	90	30	0.75	....	p. 90
Associated Northern	Schist	5	25	40	30	....	16.6	p. 29

[1] See also Bernewitz, *Min. Mag.*, 1911, V, 139; *Min. Sc. Press*, 1911, CIII, 68.

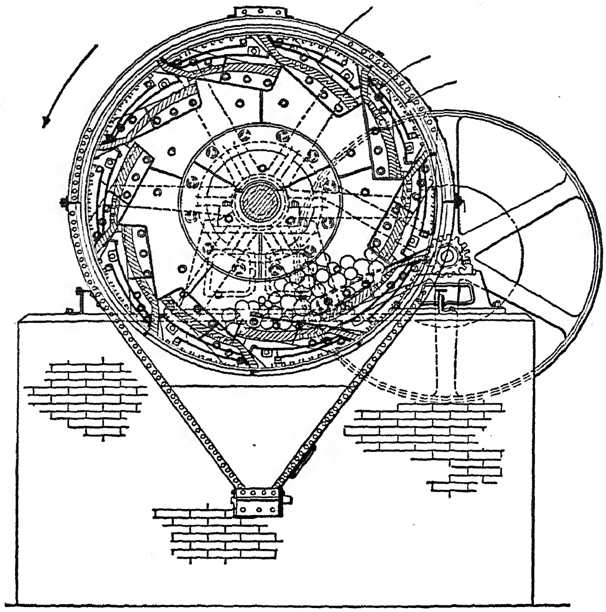


FIG. 418.—Gates ball mill.

TABLE 193.—SCREEN-ANALYSIS OF ORE TREATED IN TABLE 192

Screen	Associated Northern	Kalgoorlie	South Kalgoorlie	Great Boulder
	Per cent.	Per cent.	Per cent.	Per cent.
On 30	....	....	....	0.22
On 40	11.6	7.96	8.80	5.21
On 60	13.6	10.92	12.79	15.74
On 80	10.2	....	7.01	6.64
On 100	7.3	21.67	7.20	8.07
On 120	2.0	6.65	3.45	1.67
On 150	1.0	5.00	2.45	6.32
Through 150	53.4	47.8	58.30	56.13

3. THE TUSTIN ROTATING PULVERIZING MILL (Figs. 419 and 420).—The revolving cast-iron casing, 7 and 8, with liners near the ends to take up the wear, carries die-ring 11. This consists of 8 sections of grate-like castings of chilled iron or steel, 3 3/4 in. thick, with slots 12×1/4 in. that widen outwardly. At the end of a section, a slot 13 is 1 in. wide in order to furnish the space necessary for the oversize to return to the barrel. The casing is mounted on two hollow trunnions; one of them carries gear-wheel 5 driven by pinion 4 and pulley 1; through it passes indicator shaft 20 with indicator yoke 16 at the inner and

index-pointer 22 at the outer end; the other holds feed tube 33 receiving crushed ore from hopper 31 and water from pipe 39.

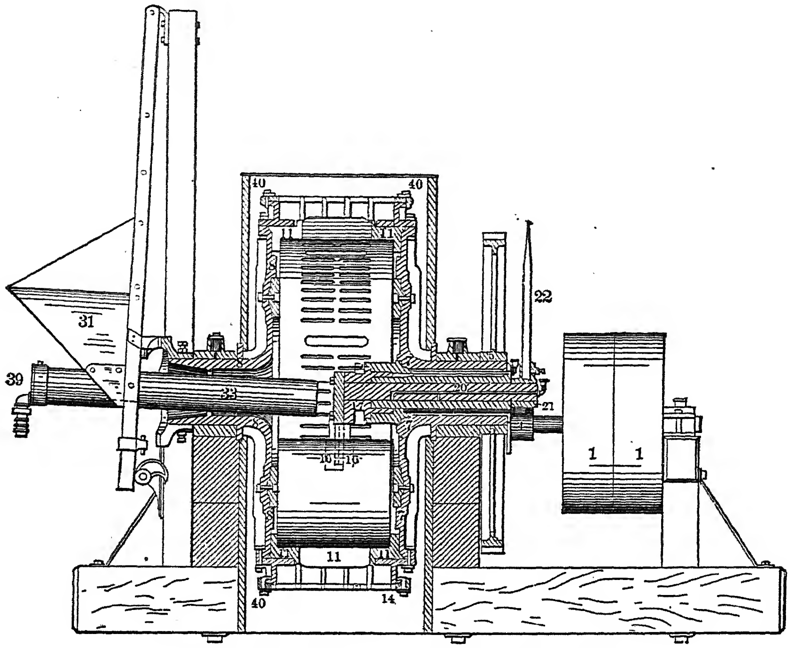


FIG. 419.—Tustin rotating pulverizing mill.

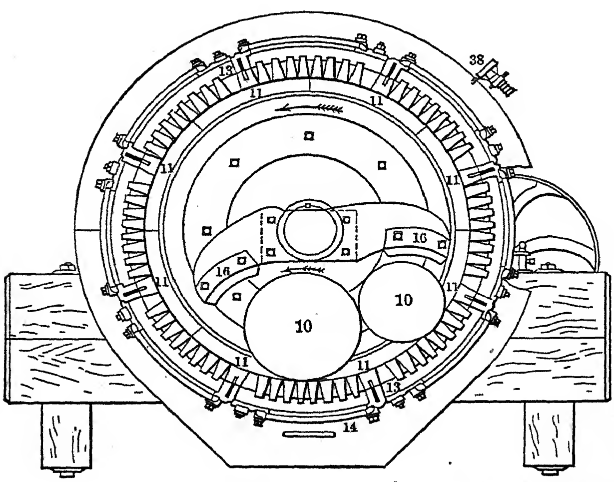


FIG. 420.—Tustin rotating pulverizing mill.

The feeding is accomplished by pushing the hopper outward by means of a ratchet wheel on trunnion 8, and pulling it suddenly back by a spring attached to hopper 31; the throw of tube 33 is regulated by the position of a cam at the

bottom of the spring-rod. The die-ring is enclosed by screening surface 14 which delivers into stationary plate-iron housing 40, discharging at the bottom, and is cleared by water-jet 38. The barrel has two rollers 10 which crush the ore by their weight. Indicator yoke is acted upon by the rollers; it is stationary with a proper feed, but oscillates to and fro when the machine is underfed, being struck by the rolls, and indicates this by the swing of pointer 22. By attaching a cam to indicator shaft 20, its motion can be used to operate a self-feeder. Covered hand-holes opposite slots 13 permit removing bits of iron that may have accidentally entered the machine. By uncovering several hand-holes, any surplus ore may be removed from the machine when this has been overfed or choked.

TABLE 194.—DETAILS OF TUSTIN ROTATING PULVERIZING MILL

Size	Die-ring		Large roller			Small roller			Tons ore, 24 hr.	H. P.	Rev. per min.
	Inside diam., inch	Length, inch	Diam., inch	Length, inch	Weight, pounds	Diam., inch	Length, inch	Weight, pounds			
Large	54	18	19½	18	1,200	14	18	750	12–24	4½	20
Small	40	14	13	14	475	11	14	350	4–10	1½	32

Work: Willard Min. Co., State Min. Cal., 1886, VI², p. 41.
Wagoner: *Tech. Soc. Pac. Coast*, 1886, III, 45.

271. Tube-mills.[1]—A tube-mill, Fig. 424, is a lined horizontal revolving steel cylinder 3 ft. 6 in. to 5 ft. in diam. and 14 to 22 ft. long, charged with flint pebbles for fine-grinding, dry or wet, ore that is fed continuously at one end and discharged at the other.

If an empty revolving cylinder is partly charged with brittle material, closed and then revolved at a certain speed, the charge will be pulverized after some time by the attrition of the particles upon themselves and upon the walls of the cylinder. This procedure is used in testing the resistance to abrasion of brick and coke, and in polishing malleable castings. If balls harder than the material to be pulverized are charged into a closed barrel, as in the "Alsing Pulverizing Cylinder with Intermittent Feed and Discharge," the comminution is hastened. The grinding action, quick at first, becomes slower as the amount of pulp formed

[1] Wilson, *Mines and Minerals*, 1908, XXIX, 8.
Fox, *op. cit.*, 1907–08, XXVIII, 537.
Del Mar, *Min. World*, 1910, XXXII, 371.
Shepard, *Met. Chem. Eng.*, 1912, X, 216, 219.
Urbiter, *Eng. Min. J.*, 1911, XCII, 257.
Diehl, *Met. Chem. Eng.*, 1911, IX, 604.
Truscott, *Min. Sc. Press*, 1912, CIV, 533.
Brown, *op. cit.*, p. 207.
"Text-book of Rand Metallurgical Practice," Lippincott, Philadelphia, 1912, Vol. I, pp. 105–140.

increases. If the pulp is screened out after a time, the grinding action is even quicker than it was at first. This shows that in order to do quick work it is necessary that the apparatus have a continuous feed and discharge; also that the ratio between grinding- and charge-surface be diminishing as the comminution progresses (Davidsen principle).

All tube-mills have a central feed. Mills with a peripheral discharge (Davidsen type) embody the two requirements of good work, as the charge thins out while the volume of pebbles remains constant. As in wet-grinding the peripheral discharge is connected with mechanical difficulties, most machines of this class have a central discharge (Krupp type), the leading exception being the Abbé mill. The simplicity of this method of delivering the pulp has probably been the reason why the central-discharge machines are used quite extensively also for dry-grinding, in the United States almost exclusively (Krupp, Alsing, Denver Engineering Co., Gates, Hardinge mills). Although the central discharge appears to violate the second requirement for good work, it must be remembered that in a revolving barrel the coarser parts of a charge have a tendency to work their way downward and to raise the lighter ones to the surface where they pass off through the only available opening. This appears to be an important reason for the good results obtained with central-discharge machines.

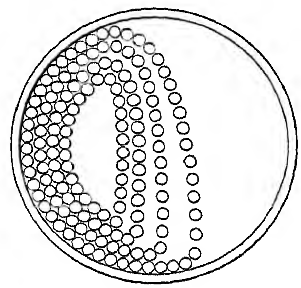


FIG. 421.—Movement of balls in tube mill.

A tube-mill, which is a fine-grinding apparatus, can do satisfactory work only with material that has been crushed medium fine; the limit in coarseness is 8-mesh, while smaller sizes such as 16- to 20-mesh are more common. In gold and silver mills the pulp from a preliminary crushing machine, *e.g.*, a stamp-mill, passes through a classifier; the overflow is fine and the underflow (sand) goes to the tube-mill to be slimed for cyaniding; the product of the mill again passes a classifier which separates any coarse material from the fine, of which 90 per cent. will pass a 150- or a 200-mesh sieve. The coarse sand is returned to the mill.

The theory of the working of a tube-mill has been studied mainly by Davidsen, Fischer,[1] Warwick[2] and White.[3] If a tube about half-filled with pebbles is revolved very slowly, the surface of the pebbles will become inclined and at a certain angle the whole mass will glide back into the original position. This is readily observed in slowly revolving cylindrical roasting furnaces of the Brückner or White type. If the speed is increased, the pebbles will rise until the angle of repose is slightly exceeded, whereupon they begin to roll down the slope. With a greater speed, Fig. 421, centrifugal force begins to exert itself, the pebbles spread out, become loose, rise on the ascending side of the tube and being projected in parabolic paths, pulverize the ore, while in the center there is left

[1] *Zt. Ver. deutsch. Ing.*, 1904, XLVIII, 437; *Eng. Min. J.*, 1904, LXXVIII, 791.

[2] *West. Chem. Met.*, 1905, I, 40.

[3] *J. Chem. Met. Min. Soc. So. Afr.*, 1904–05, V, 290; *Eng. Min. J.*, 1905, LXXX, 539.

an empty space; the ore broken by impact of the balls is ground fine by the rolling motion of the pebbles. The breaking of ore by impact must be less effective in a wet than in a dry tube-mill, as in a wet mill the balls strike the water before they hit the ore. If the speed of the mill is still further increased, the effect of the centrifugal force overcomes that of gravity, the pebbles spread out over the surface of the cylinder, leave the center hollow, and pulverization stops.

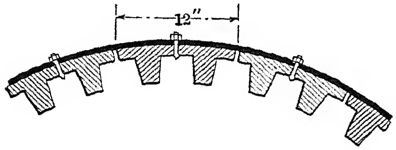


FIG. 422.—El Oro lining.

A tube-mill is either supported by trunnions or runs on tires. The diameter and length vary with the required capacity; the range has been given on page 594; there is a growing belief that in gold and silver milling with 5-ft. in diam. tubes, a length of 14 ft. is sufficient for all practical purposes. The number of revolutions a tube is to make has been formulated by Davidsen[1] as $N=\frac{200}{\sqrt{D}}$ in which D=diam. in inches. Thus a tube 5 ft. in diam. would make 200 : 7.746=26 r.p.m.; the common figure is a little higher, viz., 28. The proper speed will be that at which there is the least slipping of pebbles.

The lining for crushing gold and silver ores has been made of hard and soft wood, of silex (natural flint brick 2 1/2×4×8 in.), 4 in. thick, or quartzite laid in cement,[2] of chilled cast-iron[3] (1 1/2 in. thick), cast steel and manganese steel (1 in. thick) plates attached to the shell with counter-sunk bolts,[4] and more recently with wedged flint pebbles (El Oro lining).[5] In the El Oro lining, Figs. 422 and 423, cast-iron segmental plates with slightly dovetailed projections are bolted to the steel shell so as to form longitudinal ribs. A tube loaded with pebbles is revolved for a few minutes when pebbles become wedged into the channels and form an effective grinding surface. Any dislodged pebbles become quickly replaced. Blocks of wood placed on end are

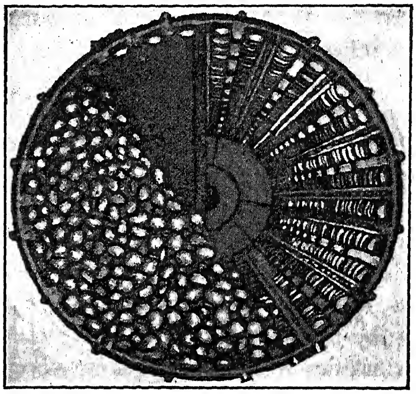


FIG. 423.—El Oro lining.

[1] *Tr. Inst. Min. Met.*, 1904–05, XIV, 154.

[2] *Min. Sc. Press*, 1906, XCIII, 108 (Barry), 534 (McKicken), 594 (Drucker); *Eng. Min. J.*, 1911, XCI, 1017 (Quartano).

[3] Komata Liner, *Eng. Min. J.*, 1912, XCIV, 1135.

[4] Drucker, *Min. Sc. Press*, 1906, XCIII, 594.

[5] *Mines and Minerals*, 1906–07, XXVII, 520; 1908, XXIX, 8 (Wilson); *Min. Mag.*, 1911, V. 35 (Mackay); *Eng. Min. J.*, 1909, LXXXVII, 684 (Rice).

Stanley, *J. Chem. Met. Min. Soc. So. Afr.*, 1907–08, VIII, 376; 1908, IX, 120.

Butters, *Min. Sc. Press*, 1906, XCII, 344.

Caetani, *Tr. A. I. M. E.*, 1906, XXXVII, 3.

serviceable in dry-crushing; they fail in wet-crushing.[1] Chilled castings have been known to last from 3 to 9 months; manganese steel has at times shown a life of 15 months and again has proved itself inferior to chilled castings; silex has been in use as long as 3 years, but the average life is much shorter.[2] Tests by Butters[3] at El Oro with a 20-×4-ft. tube gave white iron, 1 1/8 in. thick, a life of 3 months; silex, 2 1/2 in. thick, 3 months, and El Oro pebbles 10 months; with a 24-×5-ft. tube, white iron 3 months, silex 4 months, pebbles 10 months. The El Oro lining appears to have satisfactorily solved this question, both as regards expense and increased production. Other linings are the Osborne,[4] Brown,[5] Gibson and Schillie.[6]

The pebbles used for grinding, called Greenland or French flints, are flint nodules, 1 to 4 in. in diam., and are obtained from the chalks of Denmark.[7] They contain[8] SiO_2 87 to 97.1 per cent., $(Fe.Al)_2O_3$ 0.6 to 13.3 per cent., CaO 0.22 to 2.9 per cent. In sliming gold ore in Australia it has been found that 1 lb. pebble will grind 1 ton sand.

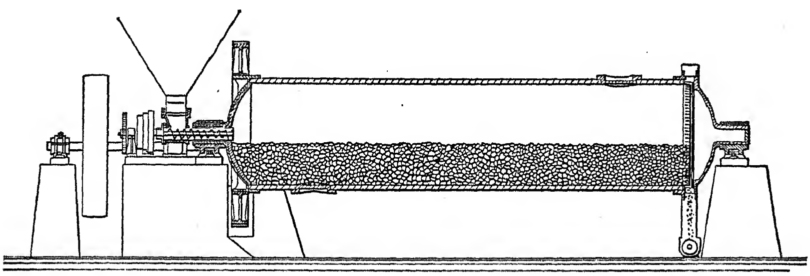


FIG. 424.—Davidsen tube mill.

The weight of pebbles necessary is calculated by Davidsen as $W=44\,N$ lb., with N=internal volume of tube in cubic feet.

The power required is large for the amount of work performed. Davidsen estimates it for a tube charged with pebbles making the r.p.m. as figured on page 596, as $P=0.15\ N$, with N=internal volume of tube in cubic feet. Experiments[9] have shown that on the whole the power increases with the weight of the pebbles.

The capacity and product are governed by the character of the ore, the degree of preliminary crushing, the rate of feeding, the weight of pebbles, the

[1] Brown, *Min. Sc. Press*, 1906, XCIII, 261.
[2] Dowling, *J. Chem. Met. Soc. So. Afr.*, 1906, VI, 305.
[3] *Min. Sc. Press*, 1906, XCII, 344.
[4] *Eng. Min. J.*, 1910, XC, 250.
[5] *New Zealand Mines Rec.*, 1909, XII, 396.
[6] *Eng. Min. J.*, 1909, LXXXVIII, 1283.
[7] Thackara, *U. S. Consular Report*, No. 1231, Jan. 6, 1902.
Hill, *Eng. Min. J.*, 1903, LXXVI, 692.
[8] Eckel, E C., "Cements, Limes and Plasters," Wiley, New York, 1905, p. 465.
[9] Fox, *Mines and Minerals*, 1907–08, XXVIII, 537.

r.p.m., and in wet-crushing, the amount of water, *i.e.*, the thickness of the pulp, which reaches 50, but rarely 60 per cent. sand. The crushing power increases up to a certain point with the weight of pebbles and of water, but at a different ratio, and then again decreases; the thicker the pulp the less the wear on lining and on pebbles.[1] The character of the product is given in screen-analyses at the end of the section.

The Davidsen Mill,[2] Fig. 424, came into the market in Europe in 1895. The tube is supported by hollow trunnions. At the feed-end it carries a large spur wheel which meshes with a pinion driven by a pulley-shaft; the same shaft carries a step-pulley for rotating the screw-conveyor, serving to feed the charge which, when ground, is discharged through openings at the periphery into a conveying trough. The mill is used mainly in dry-grinding of cement clinker. Three sizes are made, of which the leading facts are assembled in Table 195.

TABLE 195.—DETAILS OF THE DAVIDSEN TUBE-MILL

Number of mill	12	16	18
Length, feet	16	20	20
Diameter, feet	4	5.5	6.5
Horse-power	27	85	125
Revolutions per minute	27	24	22
Weight per charge, pounds	9,000	17,000	25,000
Capacity, 14-mesh, raw cement material, tons	50	115	185
Capacity, 18-mesh, cement clinker, tons	32	72	115
Capacity, 16-mesh, coal, tons	48	110	170
Product	95 per cent. through 100-mesh sieve.		

The lining is of silex or cast-iron. Silex lasts with raw cement material 5 1/2 years, with cement clinker 3 1/2 years, with siliceous ore 3 to 4 months. About 1/2 lb. silex is consumed per barrel of cement produced. The life of 1 in. of iron is equal to about 2 1/2 in. of silex. Some imperfect data of the work of the mill on gold ores in South Africa have been published.[3]

The Abbé mill, Figs. 425 and 427, occupies a place between the Davidsen and Krupp types. The tube runs on a pair of tires supported by two pairs of friction rollers; the driving gear is placed between the tires which is mechanically better than if the spur-wheel is placed at the feed-end. The distinguishing characteristics are the manner of feeding and discharging. This is shown in Figs. 426 and 427, which represent the ends of the tube with outside plates removed. At the feed-end, the stationary cover, Fig. 425, has a central opening with hopper to receive the ore from a chute. It is followed a short distance inward, Fig. 426, by a second plate revolving with the cylinder and having a crescent-shaped opening, in which the feed-spiral is placed. The two plates form the ore-chamber. With each revolution a certain amount of ore enters the spiral when the

[1] Fox, *op. cit.*

[2] Drawings in Eckels, *op. cit.*, p. 461.

[3] James, *Min. Sc. Press*, 1909, XCVIII, 52.

opening arrives at the bottom, and is gradually delivered into the central feed-opening. After two or three revolutions the required ore is fed regularly.

At the discharge-end, Fig. 427, are two concentric rows of openings; the ground pulp is discharged near the periphery into the spiral which brings it to the center whence it is removed through a pipe. The figure also shows a hand-hole

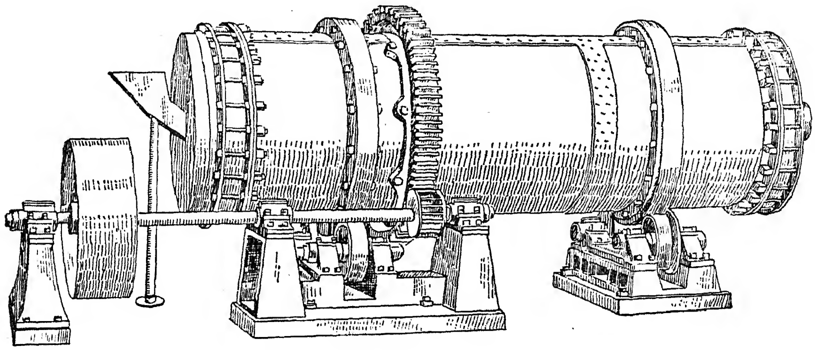


FIG. 425.—Abbé tube mill.

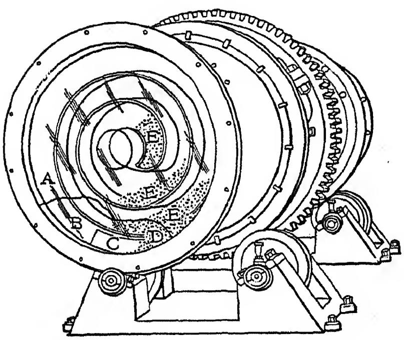


FIG. 426.—Abbé tube mill.

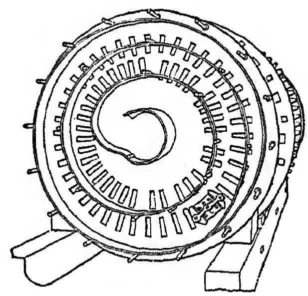


FIG. 427.—Abbé tube mill.

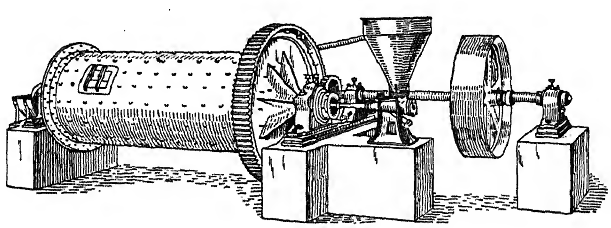


FIG. 428.—Krupp tube mill.

through which pebbles can be discharged. These are introduced at the feed-end just as is the ore.

The Krupp Mill, Fig. 428, went into operation about 1895. The tube is supported by trunnions. The feed-end always has a collar bearing, and the discharge-end sometimes a roller bearing. The common form of drive is similar

TABLE 196.—TUBE-

Locality	Mill					
	Make	Size, feet	Lining			Horse-power
			Kind	Life, months	Consumption, pounds per ton ore	
Lake View Cons. Gold Mine, Western Austral.	Krupp .	16½× 4	Manganese steel 1 in. thick.	.	0 25	30
Lake View Cons. Gold Mine, Western Austral.	Krupp . . .	12 × 4		......	0.25	25
Golden Horseshoe.......	Krupp	16½× 4	Chilled iron.	6	....	30
Telluride, Colo......... .	Abbé...	5 ×22	Silex	10–12	0.5–0.6	47
Guanajuato, Mex..	Abbé	4½×20	Silex..	9	....	30
Colorado City, Colo	Abbé.... ...	5 ×23	Silex...... .	. ..	. ...	46
Grass Valley, Cal.........	Abbé	4½×20	Silex....... ..	. .. .	....	20
Grass Valley, Cal.	Abbé	4½×20	Silex... .. .	...	. ...	16
Liberty Bell, Cal . .	....	.. . ...	....	... .	....	....
Montana-Tonopah, Nev. .	Allis-Chalmers.	5 ×22	Silex...	8	2.4	40–45
Peregrina, Guanajuato, Mex.	Allis-Chalmers. .	5 ×22	Silex (El Oro)	6–7	6	69
Peregrina, Guanajuato, Mex.	Allis-Chalmers ..	5 ×26	Silex..........	6–7	6	75
U. S. Reduction and Refg. Co., Colorado Springs, Colo.	Stearns Rogers Mfg. Co.	5 ×22	Silex..........	21–22	0.43	....

(a) Screen-analyses are given in Table 197.

MILL DATA

Revolutions per minute	Pebbles: Charge, tons	Pebbles: Consumption, pounds per ton ore	Raw material: Kind	Raw material: Size, mesh	Capacity, tons 24 hr.	Product; size, mesh	Reference	
29	4.5	1 1	Siliceous gold ore	20	35	(*a*)	p. 131	Allen, R. "West Australian Metallurgical Practice," Kalgoorlie, 1906.
29	3.5	1.1	Siliceous gold ore	20	28	(*a*)	p. 131	
30	5 0	1 23	Siliceous gold ore	50	30–33	(*a*)	p. 145	
25	14	2 0	Siliceous gold ore	14,22 wire	97	100	E. H. Nutter, Sup.	
30	8	1 5	Siliceous silver ore	26	75	120	E. H. Nutter, Sup.	
25	11	....	Siliceous gold ore	12	160	100	E. H. Nutter, Sup.	
20	8	....	Siliceous gold ore	30	30	150	E. H. Nutter, Sup.	
20	6	0.02	Concentrate	0.04″	48	200	A. D. Foote, Sup.	
35	10	3.29	Siliceous gold ore.	20	50	200(*a*)	E. A. Collins, Sup.	
26	3–4″ below center	1.2	Quartz and clay	10% through 200-mesh	182	47% through 200-mesh	E. E. McIntire, Sup.	
28		1.2	Quartz and clay		204			
27.2	....	0.43	Tails chlorination plant.	12	240	80(*a*)	H. W. Fox, Sup.	

to that of the Davidsen mill. The ore is introduced by a central "squirt" feed (a hopper with an elbow-shaped discharge pipe having a conical nozzle and an overflow pipe) and discharged through a hollow trunnion at the opposite end. The tubes are made 3 ft. 7 in., 4 ft. 1 in., 4 ft. 7 in., 5 ft. 1 in., and 5 ft. 11 in. in diam., and 9 ft. 10 in., 13 ft. 2 in., 16 ft. 5 in., 19 ft. 8 in., 23 ft. 0 in., and 20 ft. 3 in. long, making in all 26 combinations.

The Gates Mill, made by Allis-Chalmers Co., is similar to the Krupp Mill.

The Denver Engineering Works Mill is supported by trunnions. The characteristic feature is the scoop-feed which takes up, with every revolution, from a tank or launder a definite quantity of ore and delivers it through the hollow trunnion into the grinding chamber, whence it is centrally discharged through the corresponding trunnion at the opposite end. A mill at the works of the

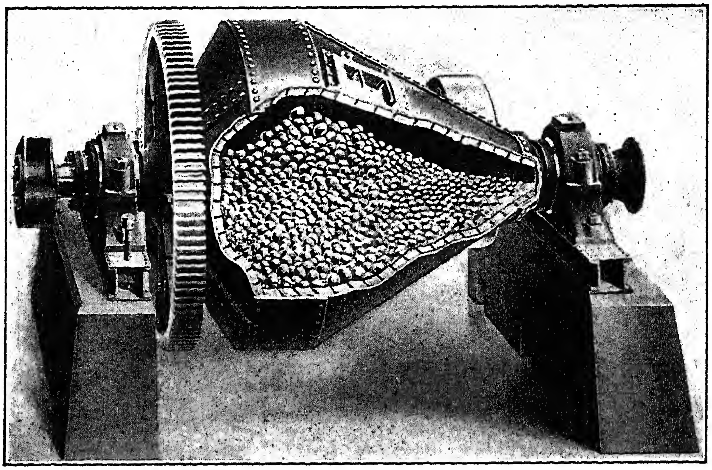


FIG. 429.—Hardinge conical pebble mill.

U. S. Smelting, Refining and Mining Co., Midvale, Utah, is 5 ft. in diam. by 14 ft. long, lined with 4 1/2-in. silex, which lasts 12 months. The mill at 28 r.p.m. requires 35 h.p., crushes and grinds 5.6 tons per hr. table middlings (through 16-mesh) to a product of which the screen-analysis shows that 9 per cent. is on 40-mesh, 15 on 60, 19 on 80, 17 on 100, 20 on 200, 10 through 200, and 10 per cent. slime.

The Hardinge Conical Pebble Mill,[1] Fig. 429, represents a new departure in tube-milling. It is a conical mill in which pebbles as well as ore to be crushed gradually become sorted so that the coarse parts of the ore are crushed by the larger pebbles which, on account of the greater peripheral speed of the deepest part of the drum, strike a heavier blow than the smaller ones near the discharge-end. A 6-ft. mill has a capacity of about 50 tons in 24 hr., using about 15 h.p.; about 90 per cent. of the pulp passes through a 100-mesh screen.

[1] Hardinge, *Tr. A. I. M. E.*, 1908, XXXIX, 337, 1913, XLV, 194; *Min. World*, 1912, XXXVII, 487; *J. Can. Min. Inst.*, 1912, XV, 129; *Eng. Min. J.*, 1910, XC, 1057.

TABLE 197.—SCREEN-ANALYSES OF TUBE-MILL PRODUCTS
(See Table 196)

Screen mesh		Lake View Consolidated		Golden Horseshoe		Liberty Bell		Montana Tonopah Mill	
Through	On	Feed	Product	Feed	Product	Feed	Product	Under-flow	Over-flow
...	20	...	...	...	...	3.8	...	...	...
...	30	...	...	...	...	...	...	6.2	...
20	30	6.00	0 50	...	...	...	...	...	...
20	40	...	...	...	...	51 2	0 2	...	...
30	40	8.00	1 00	...	...	...	...	...	...
30	60	...	...	...	...	...	...	42.6	...
40	60	4.03	1.87	...	...	18 8	0 6	...	...
...	50	...	...	10.0	3.7	...	...	...	...
...	60	...	...	...	...	...	...	...	1 0
60	80	24.05	16.71	...	...	11.2	7.3	...	1.5
50	100	...	...	52.0	46.0	...	...	...	...
60	100	...	...	...	...	...	...	21 2	...
80	100	8.93	10 64	...	...	3.1	2 7	...	4.0
100	...	...	...	...	...	11.9	...	...	...
100	150	19 85	21.37	9.2	10 8	...	89 2	...	...
100	200	...	...	...	...	...	...	16 2	27.5
150	200	15 18	22.95	5.5	3.5	...	...	...	...
200	...	13 96	24 96	23 3	36.0	...	...	11.6	66 0

A table by S. F. Shaw[1] gives details of the tube-mills of some modern cyanide plants in the U. S. and Mexico.

Table 198 by H. W. Fox gives the results of experiments carried on at the works of the U. S. Reduction and Refining Co., Colorado Springs, Colo., to obtain data for the relation existing between per cent. of ore and solution fed, volume of pebbles used, horse-power consumed, and character of product obtained. The tube-mill used was one made by the Stearns Roger Mfg. Co. of Denver, given in Table 196.

272. Grinding Pans and Tube-mills.—The question whether grinding pans or tube-mills are to be preferred for sliming ores has not yet been settled. In Australia mill men prefer the pan, in the United States the tube. Tests by Nicholson and Butt[2] and by King[3] favor the pan; the Homestake Mill, Table 185, has abandoned them. Other data are those of Wainwright-McBride,[4] Denny[5] and Cobbe-Middeton.[6] See §268.

[1] *Tr. A. I. M. E., Bull.* 31, July, 1909, p. 531; *Min. Ind.*, 1909, XVIII, 742.
[2] *London Min. J.*, 1905, LXXVIII, 244.
[3] *New Zealand Mines Record*, 1905, IX, 227.
[4] *Tr. Austral. Inst. Min. Eng.*, 1909, XIII, 38.
[5] *Proc. So. Afr. Assn. Eng.*, 1905–06, XI, 280.
[6] *Min. Mag.*, 1909, I, 213.

TABLE 198.—RELATION BETWEEN PER CENT. ORE AND SOLUTION, FINENESS OF GRINDING, AND HORSE-POWER

SCREEN-ANALYSIS OF SAND FED TO TUBE-MILLS, 12 FT. LONG, 5 FT. DIAM.

On 20	On 30	On 40	On 60	On 80	On 100	On 120	On 150	Through 150
6.0	20.0	24.0	23.0	11.0	8.0	4.0	2.0	2.0

VARIABLE PEBBLE VOLUME, FIXED ORE AND SOLUTION

Pounds pebbles	On 60	On 100	On 150	Through 150	Per cent. ore	Per cent. solution	Tons ore per 24 hr.	Indicated horse-power
3,000	42.5	27.5	8.0	22.0	63.72	36.28	172	18.8
6,000	46.5	23.5	8.0	22.0	70.17	29.83	172	20.37
9,000	42.0	26.0	8.0	24.0	74.29	25.71	172	22.5
12,000	32.0	32.0	12.0	24.0	60.00	40.00	172	32.16
15,000	29.0	30.0	14.0	27.0	65.38	34.62	172	39.13
16,800	18.0	36.0	12.0	34.0	66.67	33.33	172	42.88
18,000	3.5	29.0	16.0	51.5	66.67	33.33	172	47.16
19,000	4.0	28.0	13.0	55.0	66.67	33.33	172	51.45
20,000	9.0	32.0	15.0	44.0	71.88	28.12	172	56.28
21,000	6.0	30.0	13.5	50.5	71.88	28.12	172	60.1
22,000	6.0	29.0	15.0	50.0	71.88	28.12	172	65.39
23,000	6.0	30.0	14.0	50.0	70.37	29.63	172	77.18
24,000	3.0	27.0	16.0	54.0	70.96	29.04	172	68.61
24,500	4.0	26.0	13.0	57.0	68.18	31.82	172	69.68
25,000	3.0	26.0	14.0	57.0	66.67	33.33	172	75.04
26,000	5.0	28.0	15.0	52.0	70.00	30.00	172	68.6
27,000	8.0	33.0	14.0	45.0	68.00	32.00	172	64.85

VARIABLE ORE AND SOLUTION, FIXED PEBBLE VOLUME

Pounds pebbles	Feed	Tons ore per 24 hr.	On 60	On 100	On 150	Through 150	Per cent. ore	Per cent. solution	Ind. horse-power
20,000	3″	172	7.0	32.0	13.0	48.0	64.71	35.29	56.4
20,000	3″	172	13.0	35.0	11.0	41.0	66.67	33.33	54.28
20,000	3½″	190	12.5	36.0	10.0	41.5	71.05	28.95	51.6
20,000	3½″	190	14.0	34.0	12.0	40.0	67.86	32.14	54.8
20,000	4″	216	16.0	34.0	14.0	36.0	68.18	31.82	53.2
20,000	4″	216	14.0	36.0	16.0	34.0	69.70	30.30	49.4
20,000	4½″	231	26.0	38.0	11.0	30.0	66.67	33.33	47.5
20,000	4½″	231	30.0	30.0	10.0	30.0	72.22	27.78	43.5

VARIABLE SOLUTION, FIXED PEBBLE VOLUME AND ORE FEED

Pounds pebbles	Ore feed	Tons ore per 24 hr.	Tons solution per 24 hr.	On 60	On 100	On 150	Through 150	Per cent. ore	Per cent. solution	Ind. horse-power
20,000	3"	172	68.0	12.5	36.0	10	41.5	71.43	28.57	45.0
20,000	3"	172	75.0	13.0	34.0	12	41.0	69.56	30.44	48.9
20,000	3"	172	90.0	8.0	30.0	13	49.0	65.67	34.33	55.8
20,000	3"	172	92.0	8.0	32.0	14	46.0	65.20	34.80	57.4
20,000	3"	172	98.0	9.0	33.0	12	46.0	63.78	36.22	58.0
20,000	3"	172	111.0	8.0	33.0	13	46.0	60.70	39.30	56.9
20,000	3"	172	113.0	7.0	31.0	13	50.0	60.44	39.56	55.0
20,000	3"	172	136.0	8.0	34.0	12	46.0	55.71	44.29	55.8
20,000	3"	172	196.0	7.0	32 0	14	47.0	47.10	52.90	59.0
20,000	3"	172	207.0	5.5	30.5	13	51.0	45.40	54.60	62.3
20,000	3"	172	268.0	8.0	32.0	12	48.0	38.90	61.10	62.3

273. **Impact Pulverizers.**—The leading types of machines of this class are disintegrators and hammer crushers. To the former belong the Carr (Stedman), to the latter the Jeffrey, Williams, and Aëro machines.

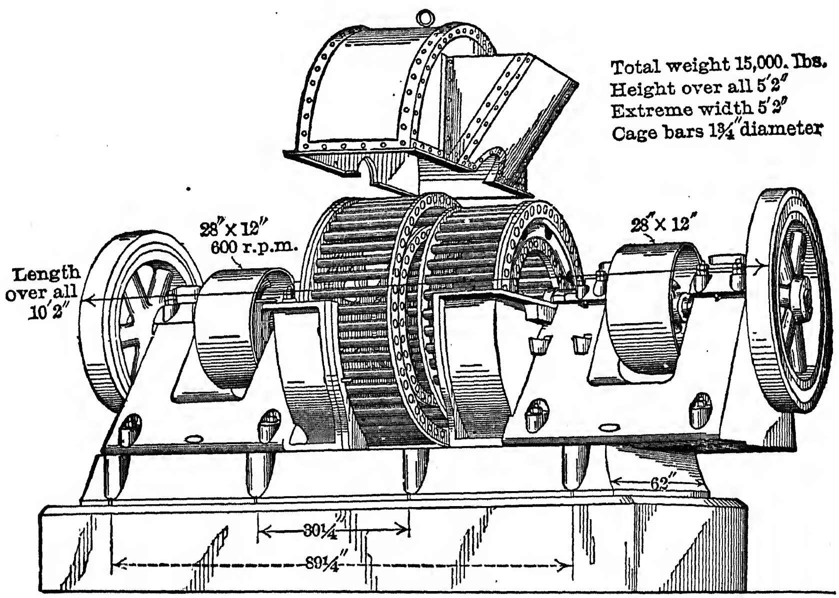


FIG. 430.—Carr or Stedman disintegrator.

THE CARR OR STEDMAN DISINTEGRATOR.[1]—In Fig. 430 is a horizontal revolving shaft carrying a circular cast-iron plate to which is attached one set of bars (10.25 in. long) forming a cage; the ends of these are connected with an annular plate carrying a second set of bars (15.125 in. long) also forming a cage. The horizontal shaft to the right, revolving in the opposite direction, carries a single

[1] *Eng. Min. J.*, 1894, LVIII, 129, see also page 198.

cast-iron disc with two sets of bars (13.125 and 17.125 in. long) forming cages. Brittle material, such as coal or clay, fed through the hopper into the center is struck by the bars of the inner cage, broken and thrown outward; it meets the bars of the second cage revolving in the opposite direction which disintegrate the partly broken material with double the energy; it finally passes through cages 3 and 4 when the dry pulp is collected by the housing, and is discharged at the bottom. A stationary bar, Fig. 431, reaching into the cage breaks up lumps and prevents the fine material from banking up. The steel bars in the cages are 1 to 1.75 in. in diam. The fineness of the product is governed by the speed of revolution. Details of the Stedman machine for crushing coal and clay are given in Table 199.

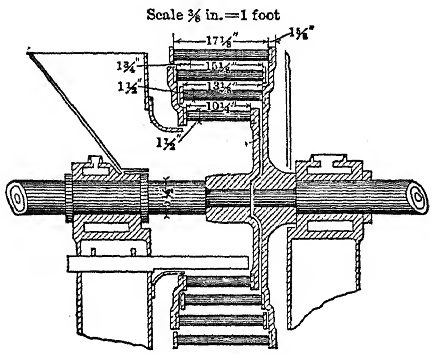


FIG. 431.—Section of Carr or Stedman disintegrator.

TABLE 199.—THE STEDMAN DISINTEGRATOR

Diameter, inches	Revolutions per minute	Horse-power	Capacity in 10 hr.	
			Coal, tons	Clay
36	600–700	12–15	....	Clay for 20,000 brick.
40	500–550	35–50	170–200	....
		15–20	....	Clay for 30,000 brick
44	....	40–60	200–250	....
50	....	70–100	350–400	....
60	....	100–125	500	....

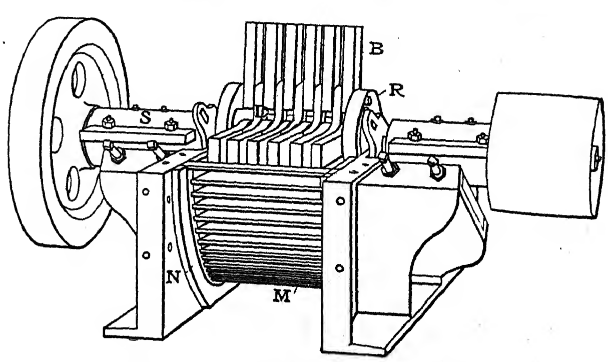


FIG. 432.—Jeffrey swing hammer.

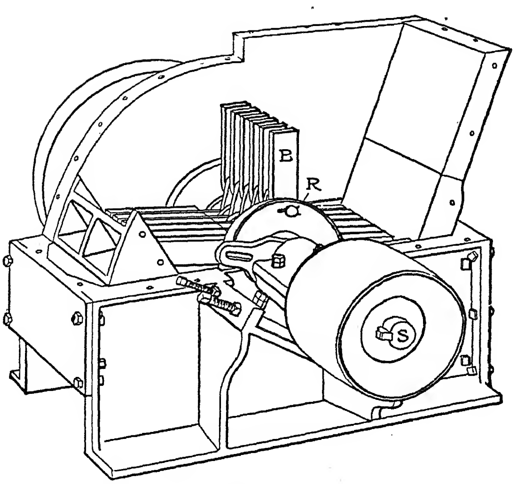


FIG. 433.—Jeffrey swing hammer.

The Jeffrey Swing Hammer, Figs. 432 and 433, is intended for crushing brittle material to medium sizes. In an iron casing with hood and feed-opening, solid sides and grated bottom, is revolved the main shaft carrying five 4-armed spiders with openings at the ends for steel bolts on which are hinged six swing arms having a free arc movement of 120°. The driving shaft which makes from 700 to 1,500 r.p.m. causes the arms to swing out and strike blows upon the coal which, when sufficiently fine, passes through the grated bottom. Table 200 gives the leading data.

TABLE 200.—JEFFREY SWING HAMMER

Size, inches		Material crushed		Grating; distance, inches	Finished material		Horse-power	Rev. per min.
Spider	Arm	Kind	Tons per hour		Per cent.	Through mesh		
18	9	Soft brick.....	$\frac{1}{4}$	$\frac{1}{4}$	40	24	6	....
30	15	Hard brick....	3	$\frac{1}{8}$	50	16	25	....
42	36	Vitreous brick.	12	$\frac{1}{8}$	50	16	80	....
36	30	Culm.........	50	1	100	$\frac{1}{4}$ in.	80	....
42	48	Bit. coal......	100	1	100	$\frac{1}{4}$ in.	100	....

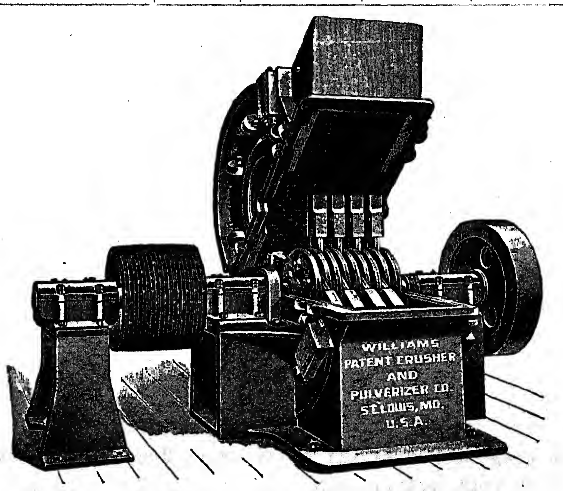


FIG. 434.—Williams hinged-hammer crusher.

The Williams Hinged-hammer Crusher, Figs. 434 and 435, represents a type of machine intended for fine-crushing; the work is similar to that of the Jeffrey machine. The machine is made in eight different sizes. Of these No. 3 may serve as example, as it is the one most frequently used. It occupies a floor-space 7 ft. 10 in. by 6 ft. 8 in., weighs 6 tons, makes 1,000 r.p.m. and requires 35 to 40 h.p. Its capacity varies with the material to be treated and the size to which this is to be reduced. The following examples give a general idea of the work that the machine is doing.

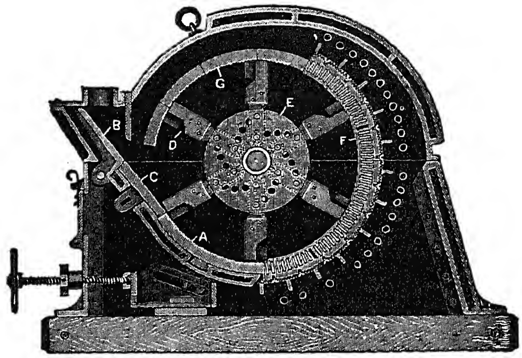


FIG. 435.—Williams hinged-hammer crusher.

TABLE 201.—WORK OF THE WILLIAMS HINGED-HAMMER CRUSHER

Material treated	Size of opening in cage	Capacity, tons per hour	Screen-analysis Per cent. Through mesh
Bituminous coal	$\frac{3}{64}$ in.	2.6	....
Bituminous coal	$\frac{3}{16}$ in.	99	....
Bituminous coal	$\frac{5}{8}$ in.	91	....
Dry clay	10-mesh	8–9	....
Shale[1]	16-mesh	8–10	....
Hydraulic cement	20-mesh	5–6	....
Limestone for cement works[1]	$\frac{1}{8}$ in.	8–10	93.75 through 40 81.25 through 60
	$\frac{1}{16}$ in.	6	93.00 through 40 87.50 through 60
	$\frac{1}{32}$ in.	4	98.44 through 30 96.87 through 40 93.75 through 60

[1] Other data for shale and limestone are given in Eckels, E. C., "Cements, Lime, and Plasters," Wiley, New York, 1905, p. 467.

The Aëro Pulverizer is used mainly to reduce coal to dust which is to be burnt in reverberatory clinkering or smelting furnaces. As seen in Fig. 436, it consists of four communicating chambers with increasing diameters in which revolve arms attached to a central shaft. The whole is enclosed in a single steel cylinder with a feed-opening at one end and a discharge at the other. In three of the chambers the arms, of lengths to fit the diameters, carry manganese-steel beaters which pulverize the coal, fed into the smallest chamber, their power becomes greater as the peripheral speed of the beaters increases with the diameters of the chambers. In the fourth chamber is a fan with wrought-iron blades which draws the coal dust through the chambers and delivers it into the pipe leading to the furnace to be heated. At the feed-end the amount of coal and air entering the pulverizer are regulated, and between the third grinding-chamber and fan-chamber is an auxiliary air-inlet controlled by a damper to admit additional air to the fuel dust with which it is thus intimately mixed. The machine is direct-connected with an electric motor. The standard sizes are the following given in Table 202.

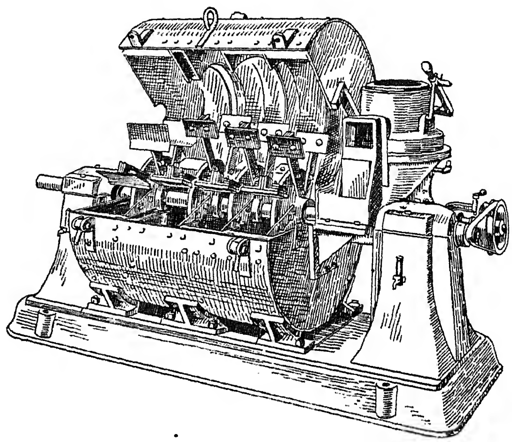


Fig. 436.—Aero pulverizer.

Table 202.—The Aëro Pulverizer

Size	Weight, pounds	Height, inches	Floor-space, inches	Normal capacity bit. coal, pounds per hour	Rev. per minute	Horse-power
A.............	1,800	28¾	61¾×27½	600	2,000	10
B.............	3,500	38½	77¾×29	1,000	1,600	15
C.............	4,200	51⅞	78¾×29	1,500	1,500	20
D.............	4,400	44¾	78¾×29	2,000	1,400	30
E.............	5,600	46¾	97×33	2,800	1,400	40

The paddles, of manganese steel, last about four months, in which time they have pulverized about 3,000 tons of coal to a dust of which 92 to 93 per cent. passes a 100-mesh sieve.

At the works of the Pennsylvania Steel Co., Lebanon, Pa., Fig. 437, a pulverizer, type *D*, requiring 30 h.p., was used, which pulverized 2,000 lb. of bituminous coal per hr. to be burnt in a nodulizing furnace; the coal fed con-

tained 2 to 3 per cent. H_2O; 80 per cent. of the dust passed through a 100-mes sieve. This pulverizer has been replaced by the Fuller-Lehigh Pulverizer Mil

274. Classification in General.—The pulp from gold- and silver-mills con tains sands, generally not coarser than 20-mesh, and slime. The particles ar therefore too small (<0.28 mm.) to be satisfactorily sized by screening, and hav to be classified by sorting. The usual aim of classifying is to separate san from slime by the sorting action of a current of water; classification of sand is sometimes extended further so as to separate quick-settling sulphide from slow-settling gangue. The free-settling of minerals from battery-pulp, say 20-mesh (0.69 mm.) down to 100-mesh (0.13 mm.) grains, is governed by the specific gravity, size and shape of the individual particles. A heavy or large mineral settles more quickly than one that is light or small, but a small particle of high specific gravity may settle at an equal rate with a large one of low specific gravity; further, a round grain settles faster than one that is long and narrow, and this faster than one that is flat. Fig. 438 represents the rate

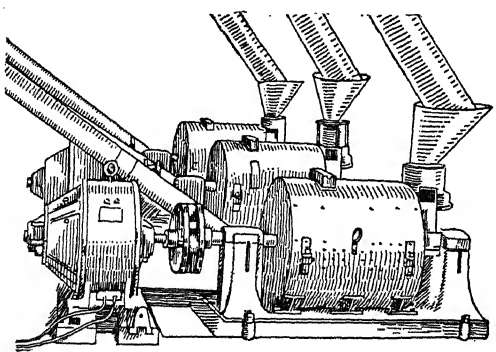


FIG. 437.—Aero pulverizer equipment at the nodulizing furnaces of the Pennsylvania Steel Co., Lebanon, Pa.

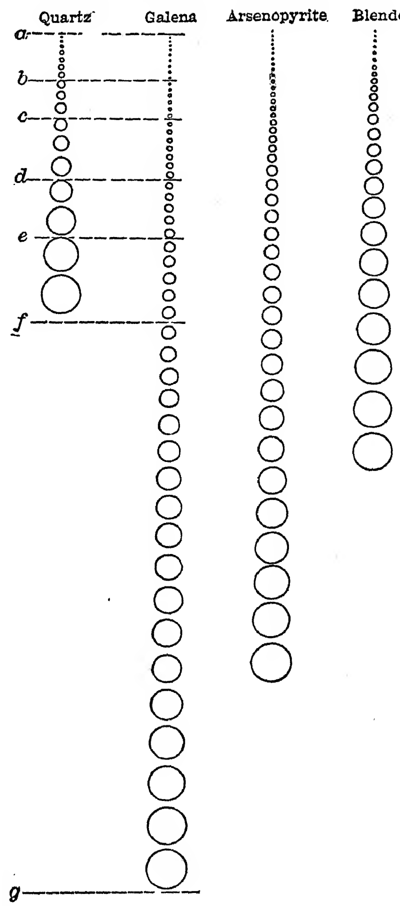


FIG. 438.—Graphical representation of free-settling.

of free-settling of the four minerals, quartz specific gravity 2.64, galena 7.59 arsenopyrite 5.63, and blende 4.05, within the range of sizes given above. Of the equal-sized minerals the heavy galena sinks the deepest; then follow arsenopyrite, blende, and the light quartz. The distances the minerals have fallen in unit time are equal to the velocities of rising currents that will just lift them. Thus grains above g will be lifted by a current of velocity ag, above f by one of the velocity af. With a rising current of the velocity af that part

of the sulphide lying below line f may be separated from a mixture of quartz and sulphide; currents weaker than af will give sorted products consisting of coarser quartz and finer sulphide which have to be cleaned by separate treatment. The velocity of a rising current sufficient[1] just to lift a mineral is expressed by $V=C\sqrt{D(\Delta-1)}$, in which $V=$ velocity in mm. per sec., $D=$ diam. in mm., $\Delta=$ specific gravity of grain, $C=$ constant which is 87 for quartz and 100 for galena. With sizes finer than 0.13 mm. for galena or 0.20 mm. for quartz, roughly finer than 100-mesh, the rate of settling of particles is influenced, in addition to diameter and specific gravity of-grains, by the colloidal state of some of the pulp[2] and the viscosity of the water. This changes the above formula to $V=K(\Delta-1)D^2$; the values of the constant K are 424 for quartz and 631 for galena.[3]

Classifiers are of two kinds: box classifiers in which the velocity of the stream carrying the suspended grains is simply retarded to allow the particles to settle, and hydraulic classifiers in which the sorting is assisted by one or more currents of clear (hydraulic) water rising from below into the flowing stream. In works the latter are installed before the former.

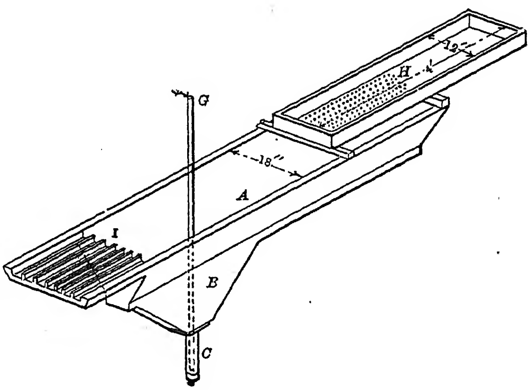


FIG. 439.—Gates classifier.

275. Hydraulic Classifiers.—The hydraulic classifier consists of one or more sorting columns, usually inverted four-sided pyramids or cones, into which the pulp is fed at the top, and the hydraulic water, coming from a pressure-box, is introduced at the apex through a suitable vertical tube. The sand settling in the vessel is met by a rising current of water and sorted; the heavy particles sink through the column, the lighter ones overflow into the next vessel where the process is repeated by a rising current of less velocity, and so on until the limit has been reached with a size of 0.15 to 0.10 mm., when the overflowing slime passes off to box classifiers. The sorted sands are then ready for further treatment. Perfect sorting is impossible, as the rising of the water and the falling of the grain are only approximately uniform.

The Gates Classifier, Figs. 439 and 441, is a V-shaped box A, 6 ft. long, 18 in. wide at the top and 4 in. at the bottom, 12 in. deep, with sides sloping 60°. The discharge-pocket B, with vertical sides and with ends sloping 60°, is 4 in. wide and 30 in. long. The sorting column, Fig. 441, consists of a pipe

[1] Richards, *Tr. A. I. M. E.*, 1907, XXXVIII, 235.

[2] Ashley, *Tr. A. I. M. E.*, 1910, XLI, 380.

[3] Sharwood, "Measurement of Pulp and Tailing," *Min. Mag.*, 1909, I, 227, 297.
Young, "Method of Testing Slime," *op. cit.*, 1910, III, 133.
Hyder, "Estimation of Pulp from its Specific Gravity," *Proc. Colo. Sc. Soc.*, 1910, IX, 417.

2 1/2 in. diam. and 8 in. long; it has at the lower end a rubber stopper *D* witl wooden nipple *E* and iron jet *F*; through the top passes vertical pipe *G* for hy draulic water. It is closed at the bottom and has eight rows of radial holes The pulp is received in box *H*, Fig. 439, perforated in part by 1/8-in holes to distribute the current and to hold back coarse particles; the tailings overflow through split launder *I*. At the plant of the Kennedy Mining anc Milling Co., Jackson, Amador Co., Cal., four classifiers treat in 24 hr., 93 tons of tailings from Frue vanners, and furnish 70 per cent. under-flow or spigot product which goes to waste and 30 per cent overflow which goes to canvas tables. The tailings carry 134.64 gal. of water per min. The hydraulic water required is 1/4 or 33.66 gal.; the spigot discharges 11.22 gal. per min., and the overflow 157.08 gal. water.

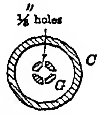


FIG. 440. Gates classifier.

Another form of V-shaped hydraulic classifier, working on the same principle as the Gates apparatus, is shown in plan and vertical section[1] in Figs. 442 and 443. It consists of three inverted pyramidal wooden boxes connected with each other in a series in which the preceding box is always smaller and shallower than the following. Each box has a baffle-board to deflect downward the inflowing pulp, and a perfectly level overflow; it has also a water-inlet and spigot-outlet controlled by dial-cocks, the water being supplied by a single main. The pulp upon entering the smallest box drops its heaviest grains. These sink through the rising current of water and pass off through the spigot while the overflow enters the next box. Here its velocity is retarded by the larger area of the box, and results in the falling out of the next-heavier grains which are met by a weaker current of hydraulic water. The same process goes on in the third box, when the leaving stream holds in suspension fine particles (slime) which can be settled only by means of a box classifier.

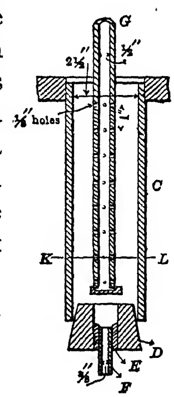


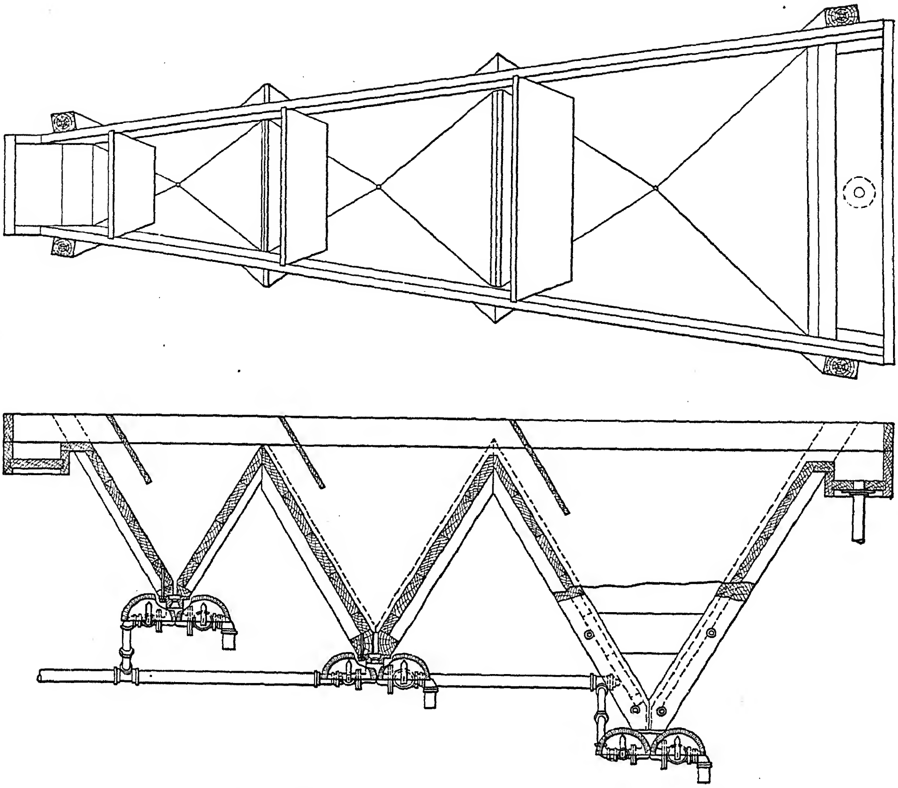
FIG. 441. Gates classifier.

The Allis-Chalmers Cone Hydraulic Classifier is shown in Figs. 444 to 446. Fig. 444 represents the single-cone classifier. It consists of a sheet-iron cone with cast-iron spigot-outlet. The characteristic feature of the apparatus is the central sub-level feed and peripheral overflow-launder. The suspended material is delivered below the surface of the water and has to rise in a current the velocity of which is retarded as the area of the vessel increases. Since a more efficient settling of particles is obtained in this way, conical classifiers with central sub-level inlets are replacing the older forms. Figs. 445 and 446 represent three conical classifiers connected in series in the same manner as the three V-shaped apparatus discussed above.

The Merrill Classifier[2] resembles the one first discussed.

[1] Allis-Chalmers Co., *Bull.* No. 1434, Sept., 1908.

[2] *Eng. Min. J.*, 1909, LXXXVII, 808; *Tr. A. I. M. E.*, 1904, XXXIV, 585; *Tr. Inst. Min. Met.*, 1911–12, XXI, 3; *Min. Eng. World*, 1912, XXXVII, 1090 (Clark-Sharwood); *Chem. Met. Min. Soc. So. Afr.*, 1910–11, XI, 323 (Smart).



FIGS. 442 and 443.—Three-compartment V-shaped hydraulic classifier.

The Richards Shallow-pocket Vortex Classifier(Figs. 447 and 449).—This consists of a shallow rectangular trough *e* with four pockets *b* and adjustable gates *c* above the pockets. The hydraulic water, admitted at *h*, receives a whirling motion before entering the sorting column *d* by being made to pass through the vortex *f*; its helical path prevents the formation of stray downward and upward currents which carry light grains into the spigot and heavy ones into the overflow. Fins *w* and *x* stop the whirling motion below the vortex *f* and at the top of the sorting column *d*; the spigot-discharge is at *g*. This classifier will handle, in 24 hr., about 60 tons of dry material that has passed a round hole 3/16 in. in diam., consisting of about 75 per cent. quartz and 25 per cent. pyrite, and forms a pulp of 10 parts water to 1 of solid.

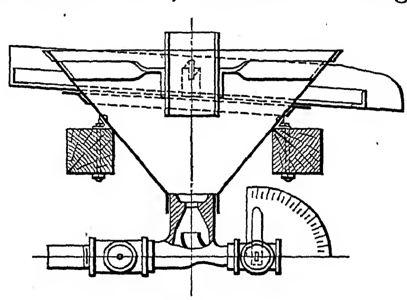
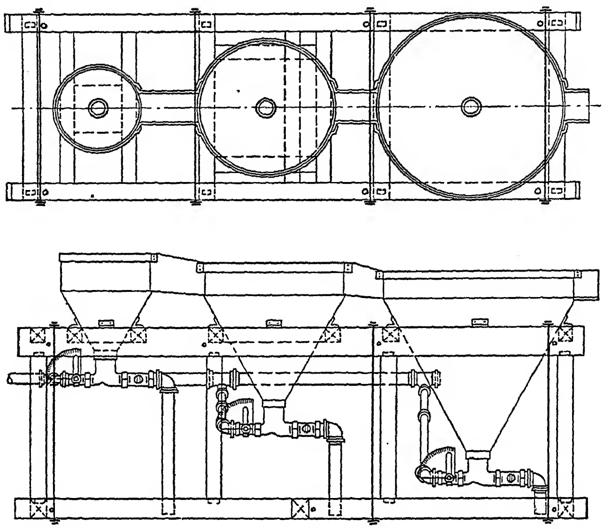


FIG. 444.—Allis-Chalmers single-cone hydraulic classifier.

The Dorr Classifier,[1] Fig. 450, is not a classifier in the usually accepted mean-

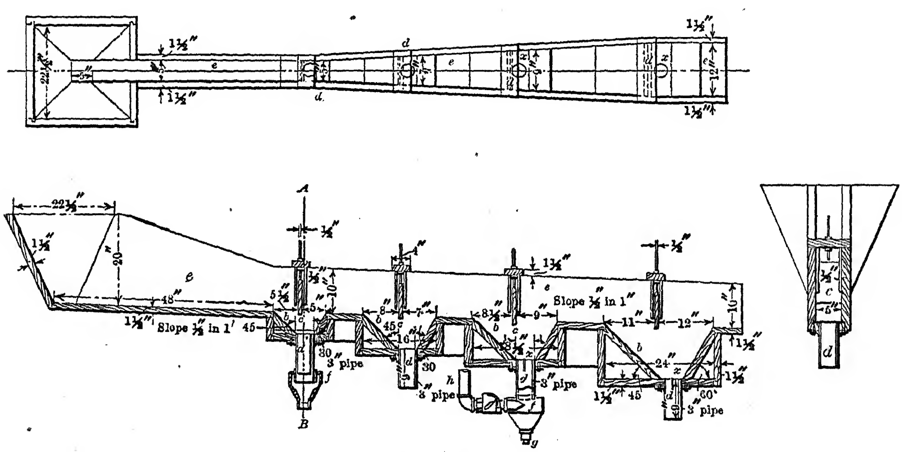
[1] *Eng. Min. J.*, 1911, XCII, 846 (Improved form).

ing of the term, although it aims to separate sands from slimes. It does this by raking the pulp up an inclined plane against a slight current of water, while the slime is carried away by the feed-water and some added wash-water. It



FIGS. 445 and 446.—Allis-Chalmers three-cone hydraulic classifier.

consists of a rectangular inclined (1 : 7) trough, closed at the lower and open at the upper end, in which two reciprocating rakes, each with 24 transverse blades, travel; the rakes are attached to longitudinal rods, and these are suspended



FIGS. 447–449.—Richards shallow-pocket vortex classifier.

from transverse axles supplied with rollers. When raking sand up the slope, the rollers travel on lower stationary tracks; when returning above the sand, they travel on latches which fall behind them. The sand meets in its upward travel

four spray-pipes, which wash off adhering slime, and after passing them is raked with some water (23 to 27 per cent.) into the sand-discharge launder; the tailings overflow at the lower end into the slime-discharge launder. At Terry, Black Hills, S. D., ore crushed with four parts alkaline cyanide solution through a 20-mesh screen gives pulp of which 60 per cent. is finer than 100-mesh; the classifier doubles the rate of leaching for the sands, and reduces the material coarser than 200-mesh from 25 to 5 per cent. At the De Lamar works, Idaho, 50 tons of 30-mesh material from a Chile mill with 6 parts solution : 1 part solid ore (1907) were treated in 24 hr.; the 30 per cent. sand recovered retains 1

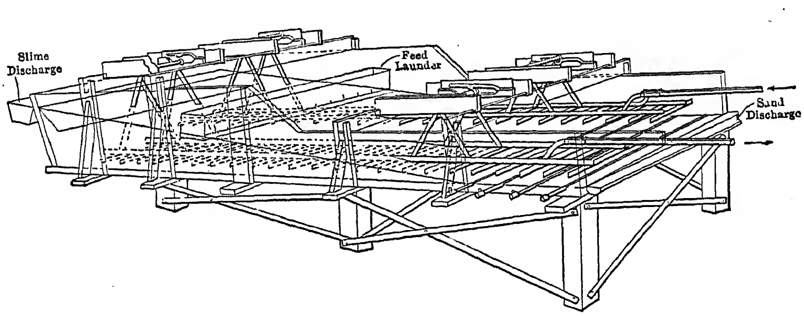


FIG. 450.—Dorr classifier.

per cent. slime, and of the 70 per cent. slime-overflow 25 per cent. is granular, of which 95 per cent. passes a 100-mesh sieve. At Real del Monté y Pachuca[1] the classifier furnishes the tube-mills with sands, which are not free from slime, containing 45 per cent. water, at the rate of 150 to 200 tons in 24 hr. if the water-jets are not used; or 75 tons free from slime when the jets are in operation.

The Esperanza Drag Classifier at El Oro,[2] the Nichols Separator Tank,[3] and the Forbes[4] are similar in principle. The Nahl apparatus[5] discharges slime intermittently.

276. Box Classifiers.—A box classifier usually has the same general form as the hydraulic classifier only the hydraulic water is omitted, and the discharge of the settled material is arranged differently. From a horizontal stream of pulp passing over a box filled with water, the particles will settle out according to the principles of free-settling given above; particles having a stronger settling power will fall out before those having a weaker. If the current passes over a series of boxes, of which the following one is in each case wider than the preceding one, a series of products will be obtained that are graded from coarse to fine. The process may be carried so far as simply to thicken the pulp that it may be handled economically, or the aim of the operation may be to clarify

[1] Rice, *Eng. Min. J.*, 1908, LXXXVI, 650.
[2] *Op. cit.*, 760.
[3] Wilson, *Mines and Minerals*, 1908, XXIX, 61.
[4] *Tr. A. I. M. E.*, 1912, XLIII, 471.
[5] *Eng. Min. J.*, 1911, XCI, 312.

the water. The Rittinger spitzkasten, the Callow tank and the Dorr thickener may serve as examples.

The Rittinger Spitzkasten, represented diagrammatically in Figs. 451 and

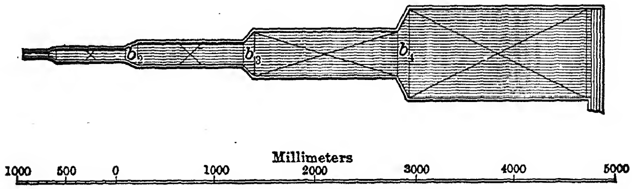


FIG. 451.—Rittinger Spitzkasten.

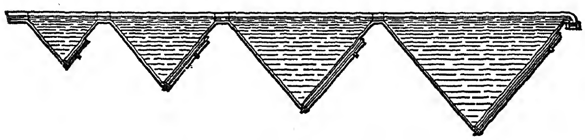


FIG. 452.—Rittinger spitzkasten.

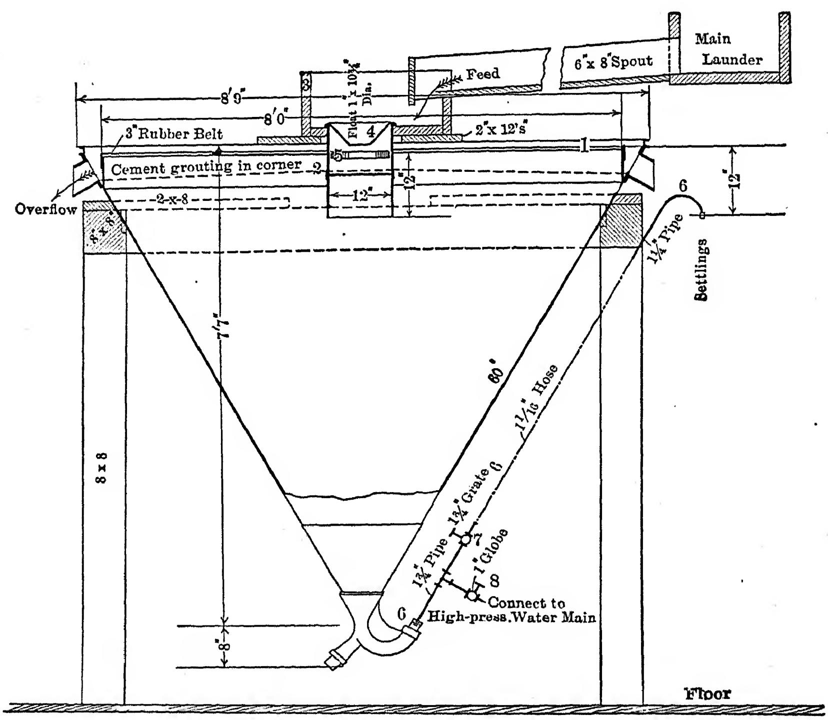


FIG. 453.—Callow tank.

452, consists of four V-shaped boxes connected in series so that the preceding overflows into the following. The boxes are so constructed that the width of one is double that of its predecessor, and that the length increases by an arith-

metical progression. Thus the widths of the four boxes are, *e.g.*, 2, 4, 8 and 16 ft., and the corresponding lengths 6, 9, 12 and 15 ft. The sides slope 50° or over from the horizontal, as an angle of less than 45° causes banks to form; the bottom of the feed-launder at the junction with the box and the overflow are on the same level; the slope of the feed-launder shows a range of 5° to 10°. The discharge of the slime is effected by a goose-neck; the outlet of this rising discharge, having an internal diameter of 1/4 to 3/4 in., is 3 to 3 1/2 ft. below the surface of the water for the first spigot, and 2 to 2 1/2 ft. for the last. A launder needs a section of 5 in. sq. for each cu. ft. of pulp passing through it per min.

The Callow Tank,[1] Figs. 453 and 454, is a conical settling tank with vertical central feed, peripheral overflow and goose-neck discharge. The tank, of No. 6

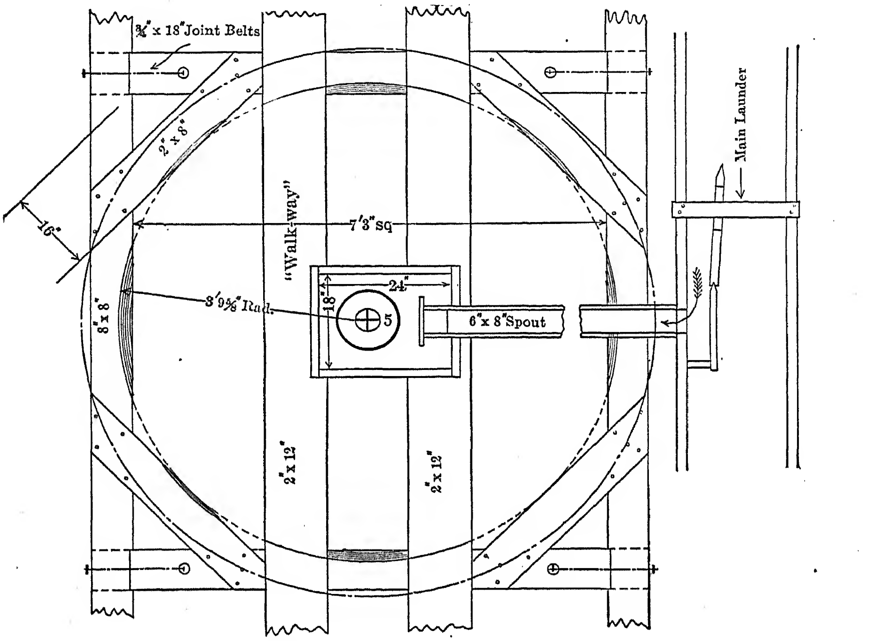


FIG. 454.—Callow tank.

steel plate, has an angle of 60° and is 8 ft. in diam.; the current of pulp from the main launder is broken by feed-box 3, directed by cone 4 into feed-pipe 2, which is 12 in. in diam. and reaches 12 in. into the water; the overflow-rim is made of rubber belting and delivers into the rim-launder which is 4 1/2 in. wide and deep; the goose-neck, of 1 1/4-in. hose and pipe, is reduced at the tip to 3/8 in. and discharges 12 to 16 in. below the water-level in the tank; it has a gate-valve 7 for shutting off the flow and a globe-valve 8 for admitting pressure-water to clear the passage in case of necessity. The tank weighs 650 lb. and costs $75. At Butte, Mont., a tank receives per min. 30 gal. pulp carrying 40 g. solids per gal. (1.06 per cent.) and delivers an overflow with 3 or 4 g. solids per gal. (prac-

[1] Utah Mining Machinery and Supply Co., Salt Lake City, Utah; *Min. Rep.*, 1904, L, 461.

tically clear water and an underflow with 300 g. solids per gal. (8 per cent.). The settling ratio is therefore 9.3 : 1, and the capacity 2 tons in 24 hr. This tank is not used in series as is the spitzkasten. The size of tank is varied to suit the conditions of the work to be done, be they separation of slime from fine sand, or water from slime and fine sand, or thickening of slime. The discharge can be continuous or intermittent.

The Dorr Continuous Thickener,[1] as seen in Fig. 455, is a flat-bottom tank in the center of which is suspended vertically a slowly rotating shaft carrying radial arms at its lower end. To the lower sides of these are fastened short pieces of angle iron at such an angle that when the shaft rotates the settled material is gradually moved toward the discharge-opening in the center of the tank. The shaft can be raised and lowered while running. The thin slime

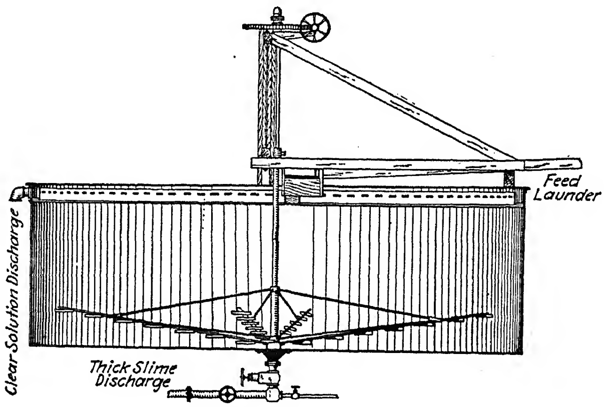


FIG. 455.—Dorr continuous thickener.

flows into the tank near the center just below the surface so as to cause no agitation; the thickened slime is discharged at the center either intermittently or continuously, and the clear water overflows at the periphery continuously. A tank 33 ft. diam., 10 ft. deep, requires 0.14 h.p. while running. At the Liberty Bell Gold Mine, Colo., 5 to 5.5 tons solution to 1 ton ore are thickened to 2.5 tons solution to 1 ton ore, the shaft making 4.8 rev. per hr.; at the San Rafael mill 4.5 sq. ft. settling area furnish in 24 hr. 1 ton thickened slime (75 per cent. $<$200-mesh, 45.5 per cent. solids); at the Mogul mill, 8 to 10 sq. ft. per ton thickened slime (98 per cent. $<$200-mesh, 30 to 40 per cent. solids); at the Liberty Bell the figures are 7.1 sq. ft. per ton with 29.5 per cent. solids; at the Homestake King, 12.6 sq. ft. per ton with 57 per cent. solids. The Forbes apparatus[2] is similar in principle. The Lamb thickener[3] has a slightly conical bottom.

277. Computation of Classifiers.—The computation of classifiers is based primarily upon the specific gravities of ore-particles and upon their rates of

[1] *Mines and Minerals*, 1909, XXX, 79.
[2] *Tr. A. I. M. E.*, 1912, XLIII, 471.
[3] *Eng. Min. J.*, 1911, XCII, 1190.

settling in still water. For any specific case the tonnage of pulp to be treated in 24 hr. including the percentage of solids in the pulp must be known. Of the solids the specific gravity and complete screen analysis will have to be determined. Thus, *e.g.*, the feed of a Richards shallow-pocket hydraulic classifier, Figs. 447 and 449, may contain particles ranging from 2.5 mm. down to the finest slime (0 mm.); the classifier has four pockets, the sands are conveniently grouped as 2.50 to 1.75, 1.75 to 1.15, 1.15 to 0.625, and 0.625 to 0.25 mm., while the overflow will retain particles 0.25 to 0 mm. which will go into settlers. In these (Figs. 451 and 452), provided a solid current of pulp is secured, it is necessary to allow the time required for a traveling particle of any desired size to settle from the top to the bottom of the tank before it reaches the overflow-end. Details of such calculations lead beyond the scope of this book. The reader is referred to the work of R. H. Richards.[1]

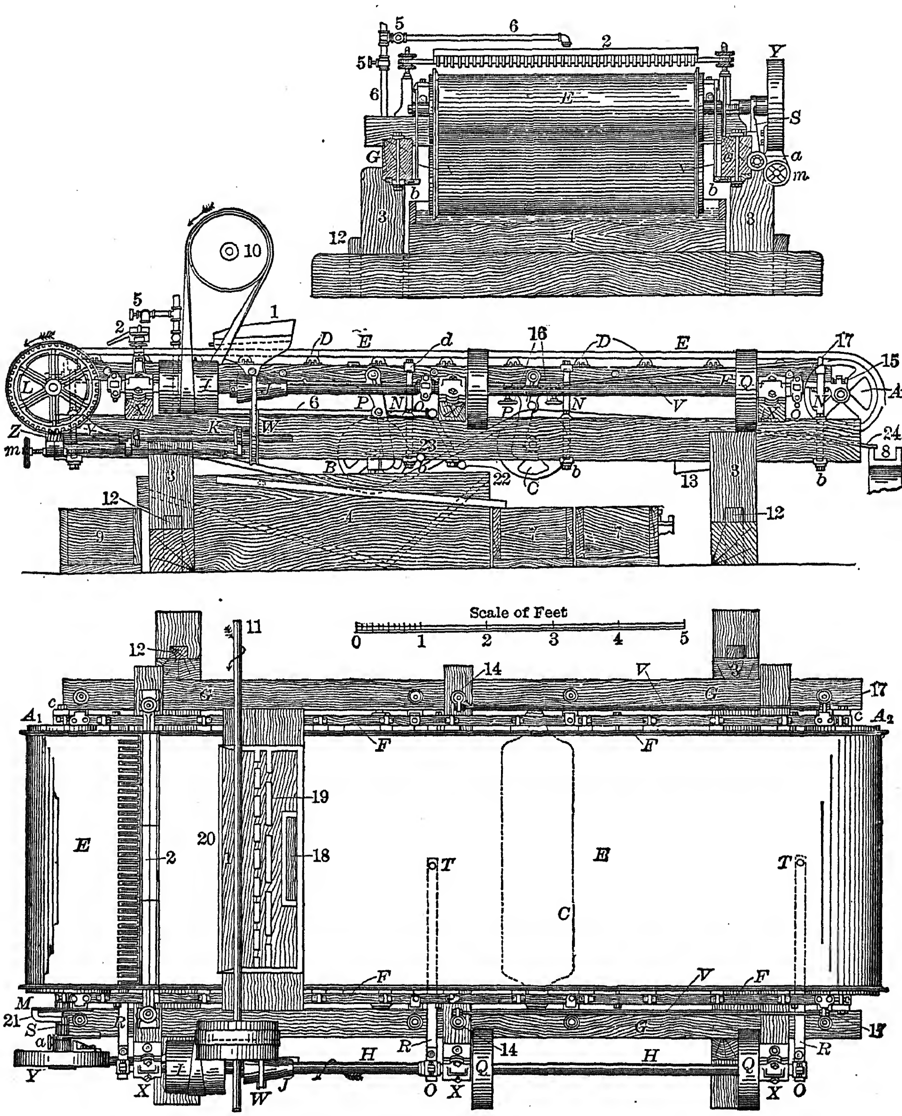
278. Concentrating Apparatus.—The apparatus used for concentrating the pulp from gold and silver mills may be divided into two classes, that which treats the pulp, *i.e.*, sand and slime, as it comes from the mill, and that which treats slime alone. The three leading machines of the first class are the Frue vanner, Johnston and Triumph concentrators, all endless-belt machines, and the Wilfley Table, a jerking machine. Theoretically these machines do better work with classified than unclassified pulp in most gold and silver mills. The difficulties of obtaining a uniform feed of classified material have been the cause of the unclassified pulp being treated directly, to separate heavy sulphide from tailing, and then to classify the tailing into sand, which may be waste or not, and slime, which is usually valuable and treated on slime-tables, the leading one being the canvas table.

Continuous slime-tables which do excellent work have so far been neglected, probably on account of the care they demand. Since the advent of leaching with potassium cyanide, the recovery of sulphides from slimes has lost some of its former importance.

THE FRUE VANNER, Figs. 456 to 458, is an endless rubber belt, *E*, with flanges on the sides, that travels up a gentle slope on horizontal transverse rollers, *D*, mounted on a wooden frame, *F*, that is shaken sidewise and causes the belt to vibrate at right angles to the direction in which it travels. The liquid pulp fed with water, at 1, near the head of the belt is separated in its downward flow into a concentrate which clings to the belt and passes over the head-rolls, A_1. The tailing is discharged over the tail-roller, A_2. Two longitudinal timbers, *G*, and three cross-timbers, *X*, form the main frame which is supported by four posts, 3, mortised into two heavy cross-sills. Its slope can be regulated by wedges, 13. The two longitudinal timbers each carry four adjustable toggle-blocks, *b*, for toggles, *N*, which are bolted to the shaking-frame, *F*, and form the support of the water distributor, *r*; one of the timbers carries the hanger, *S*, for the worm-wheel, *L*. The cross-timbers, *X*, carry the adjustable boxes for the crankshaft, *H*; the middle cross-timber has a lug, 14, at

[1] "Ore-dressing," 1903–1909, four volumes; "Text-book on Ore-dressing," 1909, one volume; both McGraw-Hill Book Co., New York.

either end for the guide-bolts, V, attached by lugs, 17, to the shaking frame in order to give it longitudinal stability. The bolts holding these lugs also serve to hold the boxes for the tail-roller, A_2. The shaking-frame, F, carried by eight toggles, N, is made of two longitudinal timbers and five cross-timbers strength-



FIGS. 456–458.—Frue vanner.

ened by cross-bolts. The longitudinal timbers carry at the ends the adjustable boxes A_1 and A_2 for the head- and tail-rollers. Near the center are suspended from long, and short, hangers, P, the dipping roller, B, and tightening roller, C. The position of the latter is controlled by hand-screw, 16; on top are

mounted the adjustable boxes of the twelve rollers, *D*, which support the belt, *E*; eight of them are set 12 in. apart, and the upper four at a shorter distance. All rollers are of galvanized iron. The belt, *E*, of two-ply rubber, is usually 4, but sometimes 6 ft. wide; the working plane is 12 ft. long; the flange 1 1/8 in. high, the surface smooth or corrugated by V-shaped grooves running crosswise. The angle of the V is 60° and the width of the groove small, 1/32 in.+. A smooth belt gives a clean concentrate; a corrugated belt a clean tailing. The slope of the belt traveling on the three or four upper rollers near the head (the cleaning plane) is greater than that of the rest of the belt (concentrating plane); with smooth belts the difference is 1/2 in., with corrugated belts 3/4 in.

The feed-distributor, 1, is attached to the shaking-frame by slotted bolts in order to permit adjustment of height and slope. The pulp is delivered to box 18, the bottom of which may be covered with an amalgamated copper plate to catch any stray amalgam as it passes by the distributing blocks, 19, into the sheet-iron gutter, 20, and thence through 1/4-in. holes 1 in. apart upon the belt just at the intersection of the cleaning and concentrating planes. The cast-iron water-distributor, 2, is carried by posts on the main frame, *G*, and can be leveled by lock-nuts. The wash-water arriving through pipe, 6, and regulated by the cock, 5, is distributed by six small brass spouts, 1 1/2 in. apart. The motive power is received by pulley, *I*, on the main shaft, *H*. The side-movement of the shaking-frame, *F*, with a throw of 1 in. is obtained through cranks, *O*, connecting rods, *R*, and bolts, *T*; the travel of the belt through cone-pulley, *J*, flanged pulley, *W*, worm-shaft, *K*, worm, *Z*, and worm-wheel, *L*. The latter is connected by crank 21 and spiral-spring connecting rod, *M*, to head-roller, *A*; the speed of the belt is regulated by hand-screw, *m*, which causes pulley, *W*, to move to and fro on a spline. The shaft, *K*, is suspended by hanger, *Y*; this can be revolved sufficiently by hand-stop screw, *a*, to raise pulley, *W*, from its belt and thus stop the travel of the rubber belt while the shaking of the frame continues.

Supposing the machine to be in motion, wash-water to have been turned on through 5, and pulp admitted into 18, the belt, which moves upward slowly, will be covered uniformly with pulp 0.15 to 0.20 in. deep, but varying in composition from concentrate at the upper to tailing at the lower end. The tailing flowing over tail-roller, A_2, is delivered on to apron, 24, and collected in launder 8; the pulp becomes richer along the concentrating plane until it has been changed into a concentrate on the cleaning plane beyond the pulp distributor, 1. The concentrate, which is allowed to retain 4 to 5 per cent. gangue, adheres to the belt and is removed by the belt dipping into tank 4, with drip- and overflow launders 9 and 7. It is also cleansed below and above with spray-water from the jets of pipes, 22 and 23; the belt now passes over tightening-roller, *C*, and returns beneath tail-roller, A_2.

In operating a vanner the following points have to be considered:

(1) The pulp must have the right degree of concentration. Coming direct from a gold or silver mill, it is usually sufficiently dilute; therefore the less wash-water is used the better. An excess of water makes clean heads and rich

tailings; with a lack, the reverse is the case. The amount of water fed per min. to the pulp on a 4-ft. belt is 3 to 5 gal. The wash-water used is 1 to 2 gal.; the pulp contains about 30 per cent. dry material. It is essential that all parts of the ore be covered with water, to prevent uncovered parts from flowing away with the tailings.

(2) The number of vibrations of a plane belt, with a throw of 1 in., is 180 to 200 per min. If the number is greater, the throw must be smaller and *vice versa*.

(3) The travel of a smooth belt is 24 to 36 in. per min.; that of a corrugated belt is faster.

(4) The slope of a smooth belt is 3 to 6 in. per 12 ft., that of a corrugated belt 2 to 3 in. more.

(5) Travel, slope, and pulp-water are interdependent; a steep slope needs a high speed of travel, a gentle slope a slow speed; a large quantity of water needs a gentle slope and a high speed.

The capacity of a 4-ft. vanner per 24 hr. is 4 to 6 tons of ore in battery-pulp that will pass a 50-mesh sieve, and 6 to 8 tons if it is a little coarser; hence a 5-stamp battery usually has two vanners. The power required is 1/4 to 1/2 h.p. One man watches eight vanners.

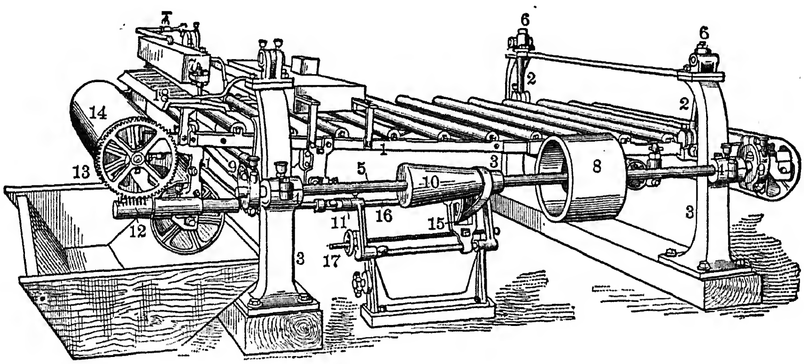


FIG. 459.—Johnston vanner.

THE JOHNSTON CONCENTRATOR,[1] Fig. 459, is a side-shake belt-machine. The shaking-frame, 1, is cast-iron and is hung from suspension links, 2, not parallel to one another, instead of being of wood and supported by toggles as in the Frue vanner. This makes the frame stronger and gives it a slightly undulating motion, and thus overcomes the tendency to bank up pulp at the edges, characteristic for the toggle-supported frame. Some differences in detail from the Frue vanner are, that the stationary frame consists of four hollow cast-iron posts, 3, bolted to two cross-sills which are held by adjustable tie-rods. This makes the bottom of the machine readily accessible and gives stability to the structure. The feed-distributor has slots (1/4 in. wide, 12 in. long, 10 in. apart) running parallel to the edge of the belt for distributing the pulp when this

[1] Risdon Iron Works, San Francisco, Cal.

has passed the distributing-blocks placed in rows 10 in. apart. The shaft of the flanged pulley, 15, serves to shift, by means of a hand-screw, the cone-pulley (10). The driving eccentrics, 9, are differently placed. The positions of the dipping- and tightening-rollers are changed, and with the former the position of the tank for receiving the concentrates.

The belt is 6 ft. wide and the working plane 12 ft. long; it receives 120 shakes per min.; treats 12 to 15 tons of ore in 24 hr., *i.e.*, one machine is sufficient for the pulp of a 5-stamp battery; and it requires 1/2 h.p.

The Triumph Concentrator, Fig. 460, is an end-shake belt-machine, *i.e.*, one which vibrates parallel to the direction in which the belt travels. The different parts are: 1, iron main-frame; 2, iron shaking-frame; 3, ten wooden toggles; 4, two connecting-rods for shaking motion and endwise stability; 5,

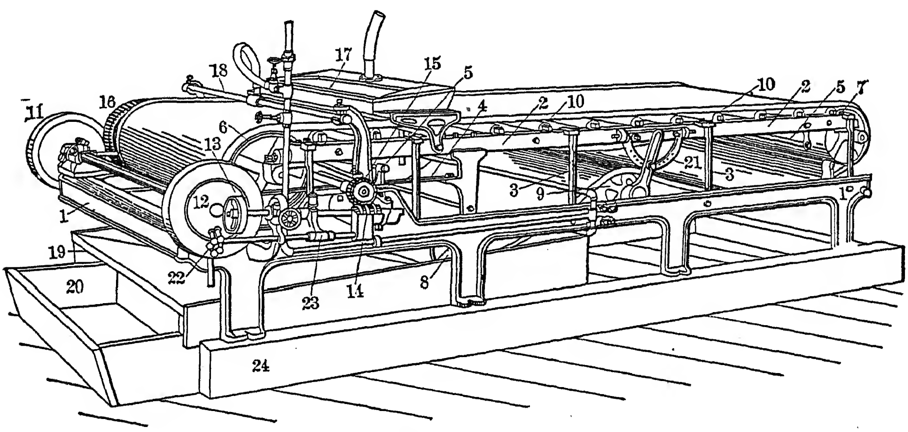


Fig. 460.—Triumph concentrator.

two rods for sidewise stability; 6, head-roller; 7, tail-roller; 8, dipping-roller; 9, tightening-roller; 10, twelve belt-supporting rollers; 11, power-pulley with shaft and eccentrics; 12, friction disc for imparting motion to belt through pulley 13, worm 14, gear 15, and pinion and gear 16; 17, pulp distributor; 18, wash-water distributor; 19 and 20, tanks for concentrates; 21, semicircular piece for raising and lowering tightening roller; 22, hand-screw and 23 arm for moving friction-pulley 13 from or toward center of disc 12, and thus decreasing or increasing the belt travel; 24, longitudinal timbers for support of main frame.

An end-shake vanner almost entirely avoids the formation of banks and gutters at the edge of the belt; it is better suited for fine pulp and makes cleaner tailings on coarse pulp than does the toggle-supported side-shake machine, but the concentrates are not as clean. Unclean fine concentrates are often cleaned on end-shake machines.

The number of shakes per min. is about 230, but the throw is less than 1 in.; the travel is 3 to 4 ft. per min., the capacity 3 to 4 tons of fine ore in 24 hr.; 1/2 h.p. is common in California.

The Wilfley Table,[1] Figs. 461 to 464, is a trapezoidal inclined end-jerk table with a surface partly covered by riffles. Its motion has two effects, one agitating which separates into layers the heavy and light grains of the pulp fed near the mechanism-end, and one jerking which throws them toward the

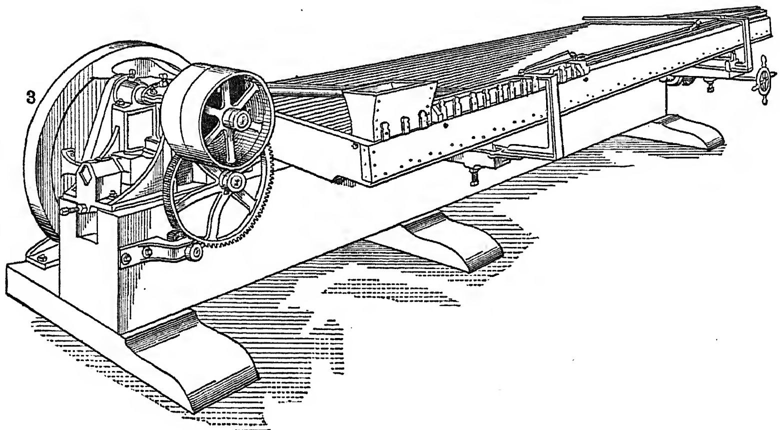


Fig. 461.—Wilfley table.

concentrate-end. The lighter grains rise to the surface and are carried down the slope toward the middling- and tailing-side by the surface-water, while the heavier grains held back by the longitudinal tapering riffles travel slowly toward the concentrate-end, and, coming to the plane-surface, are cleaned and

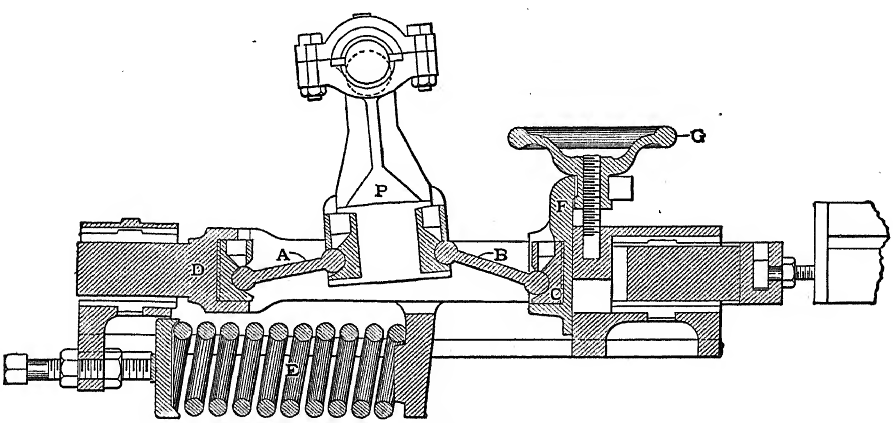


Fig. 462.—Wilfley table, motion.

sorted by wash-water fed on the upper side near the concentrate-end. The middling product is returned to the machine by a sand-wheel.

The leading parts to be considered are the foundation, tilting-frame, table, feed, elevator, and then the motion and work. The foundation consists of a

[1] Richards, *Tr. A. I. M. E.*, 1907, xxxviii, 556; 1908, xxxix, 303.

heavy timber 12×16 in. and 15 ft. long bolted to three wooden supports which are anchored in concrete to prevent any vibration. The tilting-frame (not shown), varies the slope of the table with which it is connected by four toggles. It rests with four rockers on two iron chairs that are bolted to the timber; a hand-wheel near the end of the table regulates its slope and with it that of the table.

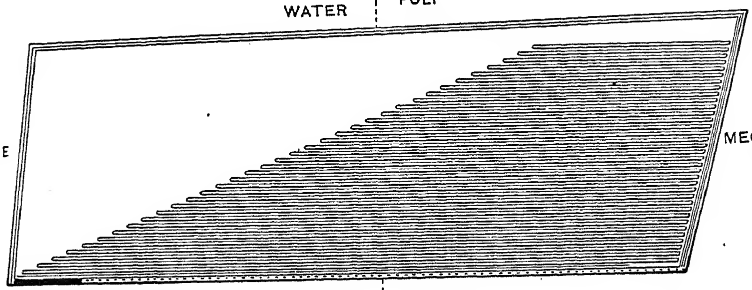


Fig. 463.—Wilfley table, top.

The table, Fig. 463 top and 464 bottom, consists of a light wooden frame which is covered by narrow strips of redwood screwed on diagonally. The cover is strengthened on the lower side by longitudinal iron-covered wooden ribs that are fortified by transverse rods; the top is covered with linoleum which carries 46 tapering riffle-cleats of sugar pine. At the mechanism-end these are 1/2 in. high and 1/4 in. wide and taper down to a feather-edge. Their length increases from the feed-side to the concentrate-end; thus at the feed-side the top cleat is 4 ft. long and at the discharge-side it extends the length of the table.

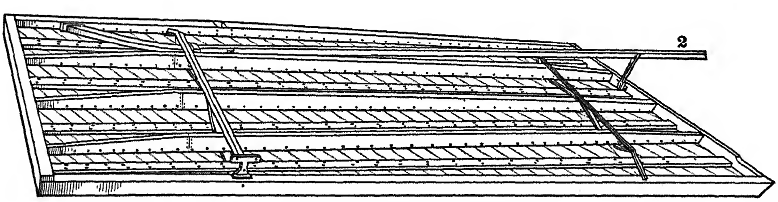


Fig. 464.—Wilfley table, bottom.

The feed of pulp and wash-water comes through a perforated launder, 4 ft. long, that is attached to the table and has a partition to deliver the pulp above the top riffle and the water the remaining length. The water for cleaning the concentrates arrives through a separate launder or spray-pipe. The middlings-elevator is a sand-wheel which returns this intermediary product through a trough to the feed-box. While true middlings will not be improved by re-treatment on the same machine, the usual middlings contain a slight amount

of tailings and concentrates and are thus cleaned. Another object of returning the small quantity of middlings generally made is to keep the table fully loaded and to thus correct irregularities in the feeding.

The machine is belt-driven. The motion, as shown in Fig. 462, is imparted to the table through pitman *P*, toggle *A*, connected with the table through yoke *D*, and toggle *B*, connected with abutment *C*, that rests in a sliding-piece *F*, and can be raised and lowered by hand-wheel *G*. The spring, *E*, takes up any lost motion. The jerking motion is obtained by the action of the toggles, as the movement is quickest when they diverge most from a straight line, and slowest when they are nearest a straight line. Thus the table has the greatest velocity at one end of the path of the pitman, and the smallest at the other. The strength of jerk is regulated by the position of the abutment *C*, by means of which the divergence of the toggles can be changed. The sand-wheel is driven from a counter-shaft which is rotated slowly by spur-wheel transmission from the pulley-shaft. In operating, the pulp has a thickness of bed corresponding to 10 to 20 grains maximum size, imagined to be one on top of the other, which reaches from the mechanism-end to a distance of 15 to 30 in. from the riffles and then gradually thins out to the thickness of simple grain on the riffle-free part of the table. Thus on the riffled or roughing plane the heavy mineral is separated from the light. On the riffle-free or cleaning plane the single heavy minerals cling more closely to the linoleum than does the remaining gangue which now is slowly removed. The table weighs 2,800 lb., makes 240 to 250 jerks per min., requires 5 to 20 gal. wash-water per min., treats 20 to 24 tons of ore in 24 hr., and requires 1/2 h.p.

Canvas Tables.[1]—These are inclined (1 1/2 in : 1 ft.) rectangular wooden tables covered with canvas (cotton duck), which work intermittently; they are fed automatically with pulp until charged (in about 1 hr.), then water-washed by hand to clean the concentrate, and lastly swept or preferably hosed to remove the finished product. The pulp fed evenly at the head flows over the table; the films of descending water roll the larger particles of gangue (quartz) downward faster than the smaller ones of sulphide which are held by the rough surface of the canvas; the waste and concentrate are collected in separate launders; the concentrates are further cleaned, preferably on a steep-slope end-shake vanner. As in canvas the warp projects higher than the woof, it is advisable to place a piece with warp running crosswise over the table; on account of the narrowness and length of the table, the opposite is not uncommon. The Gates Canvas Table,[2] used by the Kennedy Mining and Milling Co., Jackson, Amador Co., Cal., is represented in Figs. 465 to 468. There are two rows of 13 tables which treat the overflow from four Gates classifiers (Figs. 439 and 441); two of the tables are so-called extra tables (see below). The pulp passes through the distributing troughs, enters the distributors (four only are shown), and spreads over the tables. A distributor has a small dam perforated by holes, 2 in.

[1] Storms, *Min. World*, 1911, XXXIV, 581, 681, 783.
Martin, *Min. Sc.*, 1911, LXIV, 28.

[2] Hutchinson, *Min. Sc. Press*, 1896, LXXIII, 277.

apart, to correct any irregularities of flow. A table is 12 ft. wide, 10 ft. long (in reality 12 ft. sq., but 2 ft. are covered by the distributor), and slopes

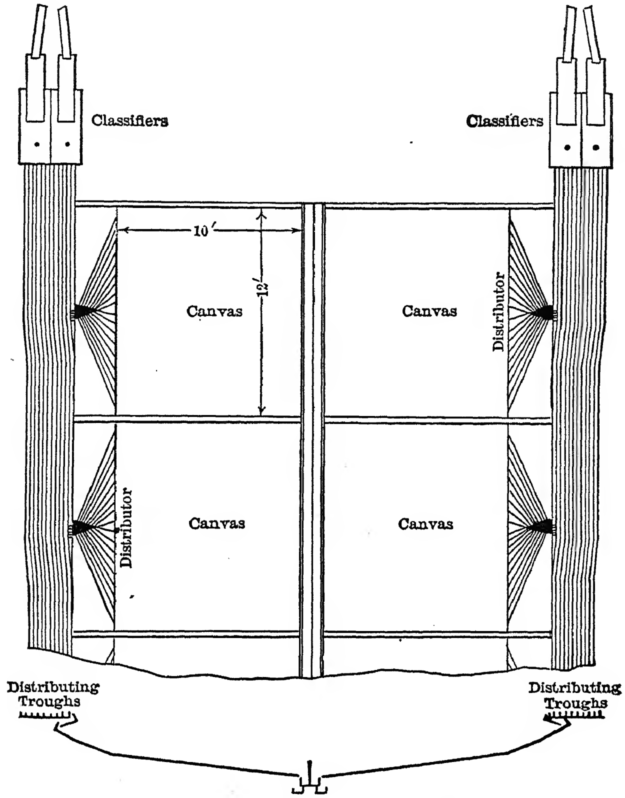
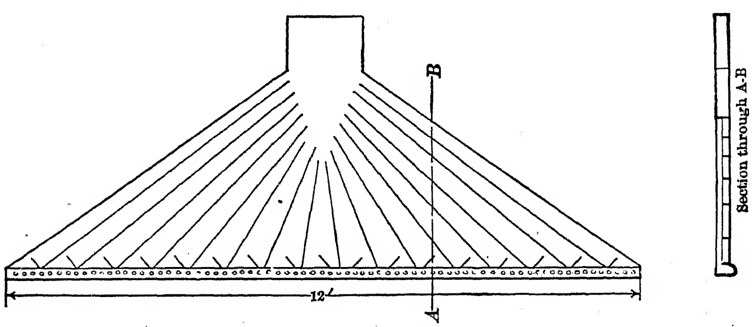


FIG. 465.—Gates canvas table.



FIGS. 466 and 467.—Gates canvas table.

1 1/2 -in. : 1 ft. A single piece of No. 6 duck is laid on the table with warp crosswise; it lasts eight months, but every 4 to 6 weeks it is moved up a short distance to change its position in relation to the joints of the boards which

wear out the duck. At the end each table has a tilting tail-board to direct the concentrate into a central launder and the tailing into lateral launders. The attendant makes the round of the tables once an hour; he switches the feed from two regular tables on to two extra tables, turns on water to remove some of the tailing, turns the tail-board, washes the concentrate into its launder with a flat, broom-shaped jet, tilts the tail-board back to its original position and shifts back the feed; he then repeats these operations with the next pair of regular tables. The 26 tables treat 30 tons pulp (dry weight) assaying $2.25 in Au per ton, yield 2 tons concentrate worth $30 per ton to be concentrated on an end-shake vanner to 1,000 lb. sulphide worth $100 per ton. Other recent examples are the tables at the Central Mill of the North Star Co., Cal.,[1] Goldfield, Nev.,[2] and the Giant Mine, Rhodesia.[3] The Darrow-Hambric Table with revolving feed has been described by Martin.[4]

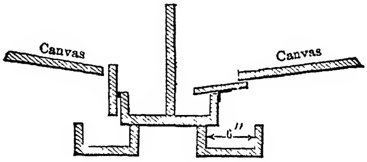


FIG. 468.—Gates canvas table.

279. Briquetting in General.[5]—Briquetting of coal was discussed in §108 to

[1] Wolcott, *Eng. Min. J.*, 1909, LXXXVII, 439.

[2] Kirby, *Eng. Min. J.*, 1912, XCIV, 207.

[3] Cooke, *J. Chem. Met. Min. Soc. So. Afr.*, 1908, IX, 152; *London Min. J.*, 1909, LXXXV, 366.

[4] *Eng. Min. J.*, 1912, XCIV, 542.

[5] Franke, G., "Handbuch der Brikettbereitung," Enke, Stuttgart, 1910, Vol. II.

IRON ORE. *General:* Louis, *Cass. Mag.*, 1905, LXXV, 1098; Colby, *J. I. and St. I.*, 1906, III, 369; Weiskopf, *Stahl u. Eisen*, 1904, XXIV, 275; Wedding, *op. cit.*, 1906, XXVI, 276; German Commission, *op. cit.*, 1908, XXVIII, 321, 1909, XXIX, 240 (Goebel); Elbers, *Am. Mfr.*, 1904, LXXIV, 731; 1905, LXXV, 19; Mould, *Iron Tr. Rev.*, Oct. 23, 1902, p. 45; Vogel, *Tr. A. I. M. E.*, 1912, XLIII, 381; Landis, *op. cit.*, 375; Franke, *Stahl u. Eisen*, 1910, XXX, 1060, 1340 (Venator); 1341 (Tigler). White, *Iron Age*, 1911, LXXXVII, 374 (Specifications).

Gröndal Method: Magnuson, *Oest. Zt. Berg. Hüttenw.*, 1903, LI, 180 (Plant of Pitkäranta); Hansell, *Iron Age*, 1911, LXXXVII, 611 (Sydoranger); Woodbridge, 1911, XCII, 261; Kroupa, *op. cit.*, 1904, LII, 589; Louis, *J. I. and St. I.*, 1904, I, 40; Zeidler, *Stahl u. Eisen*, 1905, XXV, 321 (Plant of Kertsch); Weiskopf, *op. cit.*, 1906, XXVI, 78; Gröndal, *op. cit.*, 1908, XXVIII, 1194; 1911, XXXI, 537, 1097; Bennie, *Electrochem. Met. Ind.*, 1907, V, 135; Francke, *Glück Auf*, 1908, XLIV, 1417, 1453; *Seventeenth Ann. Rep.*, Bureau of Mines of Ontario; Johannsen, *Jernkont. Ann.*, 1908, p. 400; Hansell, *Iron Tr. Rev.*, 1909, XLIV, 364; *Tr. A. I. M. E.*, 1912, XLIII, 394.

Others: Bumby, *West Scotland Inst.*, 1901–02, IX, 147; Edison, *Iron Age*, Oct. 28, 1897, p. 7; Hale, *op. cit.*, Dec. 11, 1902, p. 8; Ronay, *op. cit.*, 1905, LXXV, 1098; Ruthenberg, *Iron Tr. Rev.*, Sept. 14, 1905, p. 19; Bennie, *J. Can. Min. Inst.*, 1907, X, 261; Schumacher Process, *Tr. A. I. M. E.*, 1912, XLIII, 387 (Richards); *Iron Age*, 1910, LXXXVI, 1330; Schwartz, *J. I. and St. I.*, 1910, II, 9; Auzies, *Rev. Met.*, 1912, IX, 35; Ronay Process, *Iron Age*, 1910, LXXXVI, 1330; Weiss Process, Holzhüter, *Stahl u. Eisen*, 1911, XXXI, 1539; *Iron Age*, 1911, LXXXVIII, 1185; *Rev. Mét.*, 1912, IX, 44; Hale Process, *Iron Age*, Dec. 11, 1902, p. 8 ($CaCl_2+Na_2CO_3$)

Nodulizing: Witte, *Stahl u. Eisen*, 1910, XXX, 755 (Fernie Works); *Iron Age*, 1910, LXXXVI, 504; Baker and Wolle, *Met. Chem.* Eng., 1911, IX, 607; Little, *Iron Age*, 1911, LXXXVIII, 586 (Mayari); Editor, *Iron Age*, 1905, LXXVI, 583; Colby, *J. I. and St. I.*, 1906, III, 358; Heberlein, *Stahl u. Eisen*, 1913, XXXI, 244.

Dwight-Lloyd: Gayley, *Tr. A. I. M. E.*, 1911, XLII, 180; *Eng. Min. J.*, 1911, XCII, 50, 115; *Iron Age*, 1912, LXXXIX, 73 (Birdsboro, Pa.); Klugh, *Tr. A. I. M. E.*, 1912, XLIII, 364.

NON-FERROUS ORES, *General:* Hofman, *Min. Ind.*, 1892 to date, under "Lead"; Hofman,

111; here will be treated the briquetting of other finely-divided material, ore and metal scrap, that it may be better suited for treatment in coarse-ore roasting and in blast- and cupola-furnaces.

280. Briquetting Ore.—The raw material is finely-divided ore (Mesabi hematite, purple ore), concentrate (magnetite, pyrite, etc.), or flue dust collected in various pyrometallurgical operations. The mode of operating, which in the first place must be cheap, depends upon the use or omission of a binding-material, which, in turn, is governed by the character of the raw material and the destination of the briquette; the two together furnish the means of deciding upon the apparatus. A briquette should be porous so that gases can act chemically upon it, but at the same time strong that it may not be disintegrated by atmospheric conditions when stored, or by compression, attrition, water-vapor and heat to which it may be exposed in a furnace.

281. Briquetting Ore without Binder.—There are some ores which have binding properties, such as certain argillaceous iron ores, "blue billy," which retains some Na_2SO_4, and many slimes from concentrating plants or gold and silver mills. Their treatment is the same as that of a non-binding ore to which a binder has been added. With non-binding ores compression has been tried, but upon heating, the briquette falls to pieces; the only practicable method is that of sintering or fusing. Here it is convenient to distinguish between non-ferrous and ferrous ores.

Non-ferrous ores are frequently smelted in reverberatory furnaces where the fine state of division does no harm. Readily fusible oxide ores that are to be smelted in the blast-furnace have been agglomerated and even fused in special reverberatory furnaces, but the operation is expensive and may even be carried on at a loss in case the metal is volatile, as with lead ores; it has therefore become obsolete and has been replaced by briquetting with a binder. The same holds good for sulphide ores that are to be smelted raw in the blast-furnace. With sulphide ores that have to be roasted the finishing temperature is sometimes carried to the fritting-point, as is the case with some lead ores and mattes. But here the modern process of blast-roasting (§176) has already replaced to a large extent sintering and fusing furnaces and promises to extend its field of usefulness.

Of the many processes and apparatus proposed for sintering oxide iron ore, three have so far shown vitality: the Gröndal for magnetic oxide, the Nodulizing and Dwight-Lloyd for oxides in general. The difficulty met in sintering is the maintenance of a correct temperature, as with magnetite the difference in heat between sintering and fusing is 200 to 250° C.,[1] with purple ore 150°, and with flue dust only 100°. If the temperature is too low, the ore comes out raw; if

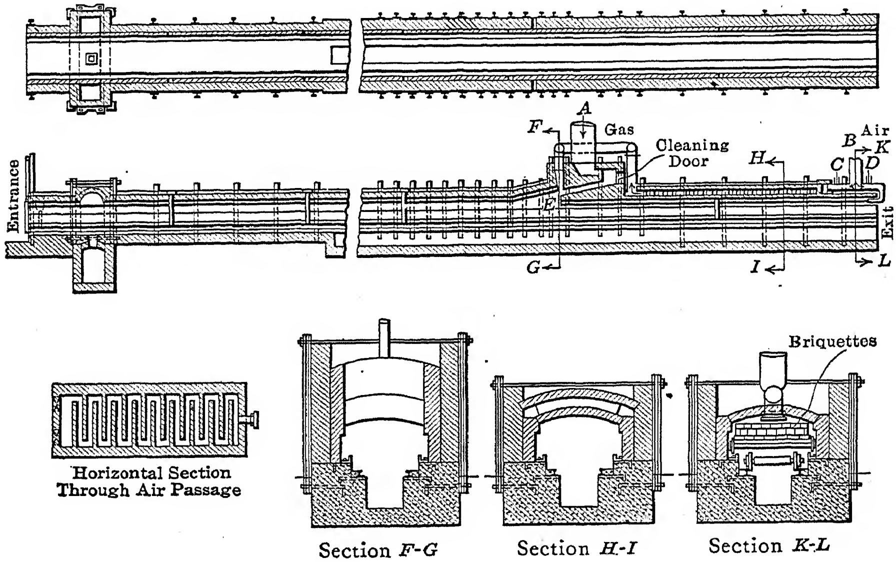
"Metallurgy of Lead," 1899, p. 398; Schorr, *Eng. Min. J.*, 1902, LXXIV, 673; Irwin, *Eng. Mag.*, 1901–02, XXII, 889.

METAL SCRAP. Ronay, *Brass World*, 1911, VII, 41; Siepke, *Metallurgie*, 1912, IX, 123; Moldenke, *Foundry*, 1911, XXXVIII, 278; Leyde, *Stahl u. Eisen*, 1909, XXIX, 1881; Leber, *op. cit.*, 1910, XXX, 1759; Messerschmitt, *op. cit.*, p. 2063; Schoemann, *op. cit.*, 1911, XXXI, 1045.

[1] Wedding, *Stahl u. Eisen*, 1906, XXVI, 77.

too high it is fused, and fusion injures the furnace as well as the product. In the treatment of Fe_2O_3 which just begins to sinter at 1,375° C.[1] and fuses at 1,560° C.[2], being in part decomposed, a reducing agent has to be added to the charge to change some of the Fe_2O_3 into Fe_3O_4 which melts at 1,527° C.; further it may be necessary to add some fluxing material, but usually the ore contains sufficient gangue to form some silicate or ferrite which will assist in sticking together the single particles.

THE GRÖNDAL PROCESS AT HERRÄNG, SWEDEN.—Magnetite concentrates (Fe_3O_4, 88.74; Fe_2O_3, 1.34; MnO, 0.65; MgO, 1.01; CaO, 2.30; Al_2O_3, 0.79; SiO_2, 5.70; P_2O_5, 0.006; S, 0.17; Cu, 0.015; total, 100.721; Fe, 65.20; P, 0.0026) are stored in bins holding 1,000 tons, allowed to drain for a week after which



FIGS. 469–474.—Gröndal furnace for sintering briquettes.

they retain 7.5 per cent. H_2O; they are then compressed into briquettes (6×6×3 in. weighing 10 lb. each) with an open 2-stamp battery with stamps weighing 1,600 lb. and dropping 6 1/2 to 7 1/2 in., each briquette receiving three blows; the process requires 3 h.p.; each stamp makes 8 to 12 briquettes per min.

The briquettes are transferred by hand with trowels on to an iron flat-car, Figs. 469 to 474, 40 by 80 in., taking two layers of 84 briquettes on edge or 1500 lb. A car has a groove at one end and a rib at the other so as to form a tight joint, and a deep flange at the sides which travels in a channel sealed with sand to exclude air. The wheels have roller-bearings and are lubricated with graphite. The sintering is carried on with iron blast-furnace gas (CO, 22; CO_2, 15) in a flue-shaped furnace 148 ft. long by 4 ft. 4 in. wide by 5 ft. 3 in.

[1] Hofman-Mostowitsch, *Tr. A. I. M. E.*, 1909, XL, 807.
[2] Kohlmeyer, *Metallurgie*, 1909, VI, 323.

high, the roof rising in the highest part of the furnace to 8 ft. 3 in.[1] In the furnace three divisions may be distinguished; the head division, 60 ft. long, at which the train of cars loaded with green briquettes enters and in which the briquettes are warmed and roasted; the central division, 33 ft. long, in which the briquettes are sintered at a temperature of 1,300° C.; the tail division, 55 ft. long, in which the sintered briquettes are cooled and at the end of which the cars leave the flue. The air for combustion enters at the head and underneath the air-tight horizontal partition formed by the platforms of the cars. It cools the bottoms of the cars, rises at the tail-end, and returns by the compartment above the cars, travels over and cools the sintered briquettes, becomes preheated, and meets the heating-gas. The products of combustion leaving the sintering division, travel over the green briquettes, dry, warm and roast them, and, having been cooled to 150° C., pass off into the stack situated near the head-end. One car with sintered and cooled briquettes is withdrawn at the tail-end every half-hour, the train is moved forward, and a car with green briquettes introduced at the head-end. In firing with producer gas, coal equal to 3.75 per cent. of the weight of sintered briquettes is required. Three tunnel-furnaces produce 100 tons of briquettes in 24 hr. The following analysis shows how effective is the elimination of S: SiO_2, 5.50; Fe_2O_3, 85.93; FeO, 3.96; Al_2O_3, 0.76; Mn_2O_3, 0.63; CaO, 2.23; MgO, 0.97; P_2O_5, 0.006; S, 0.010; Cu, 0.007; total, 100.003; Fe, 63.16; P, 0.0026.[2] The briquettes are stacked and then broken in a crusher. The process is in operation also at Pitkäranta, Sweden, and Kertsch, Russia. Its drawback is the cost which in Germany, according to Weiskopf,[3] at its lowest is $1.00 to $1.25 per ton, whereas it ought not to exceed $0.75.[4]

THE NODULIZING PROCESS.—Here finely-divided oxide of iron, with not >5 per cent. C., is fed in at the throat of an inclined revolving steel cylinder, about 100 ft. long and 7 ft. not >12 ft. in diam., that is fired from the lower end with fuel dust, oil or gas.[5] In its descent through the furnace the fine ore loses its S in the upper part and sinters to nodules in the lower part, whence it is discharged continuously into the boot of an elevator which delivers the hot nodules into a water-sprayed upright receiving cylinder that is emptied at intervals from below into a car. The flux used is crushed limestone or lime-forkings from partially slaked lime. The addition of lime beside furnishing a base for the formation of slag which assists in agglomerating the particles, may prevent nodules of magnetite from becoming glazed.[6] One per cent. of organic matter (tar), sprayed into the ore while being fed, diminishes the amount of

[1] In new installations the improved form by Ramén is taking the place of the older construction of Gröndal.

[2] Other analyses of raw ore, concentrates and briquettes are given: *Stahl u. Eisen*, 1908, XXVIII, 1195.

[3] Paper by Hansell, *op. cit.; Stahl u. Eisen*, 1913, xxxiii, 276, 319.

[4] *Op. cit.*

[5] With fuel dust, which gives a short flame, rings of sintered ore do not form as readily in the furnace as is the case when oil or gas is used.

[6] *Stahl u. Eisen*, 1908, XXVIII, 1194.

flue dust and reduces Fe_2O_3 to Fe_3O_4 which sinters at a lower temperature. With magnetite it is not used; with hematite it has been given up in several instances.

The process has been found successful with finely-divided hematite, "blue billy," magnetite and franklinite concentrates, and flue dust.

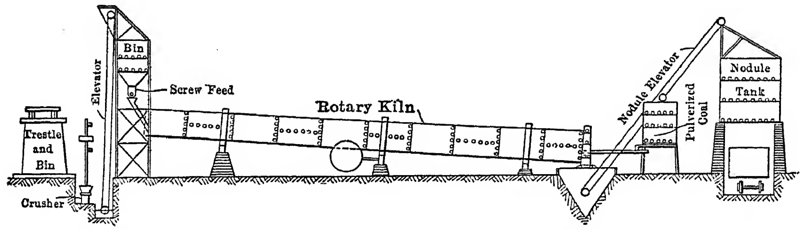


FIG. 475.—Nodulizing plant, National Metallurgic Co., Newark, N. J.

An elevation and plan of the Newark, N. J., plant of the National Metallurgic Co. is given in Figs. 475 and 476. The ore, mainly burnt pyrite (cinder), arriving on the elevated track and stored in bins below, is crushed if necessary, elevated to the cinder bin, and discharged by a screw-conveyor into the feed-

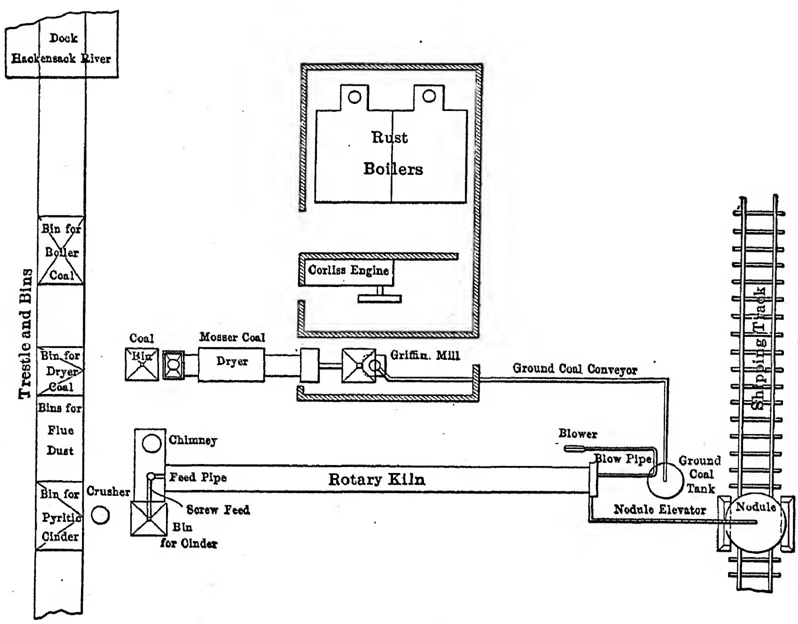


FIG. 476.—Nodulizing plant, National Metallurgic Co., Newark, N. J.

pipe. At the end of this pipe tar, kept hot in a tank built around the chimney, is dropped into the ore. The rotating furnace is fired with fuel dust, the hot nodules collected in the brick bin are raised by the nodule-elevator to the internally-sprayed nodule-tank and drawn into a car. The cylinder is 100 ft.

long, 7 ft. diam. and drawn to 6 ft. at the feed; the ore passes through it in 1 1/2 hr. Cinder from Newfoundland with Fe, 54.29; S, 3.25; SiO_2, 4.33, gave nodules with Fe, 59.29; S, 0.074; SiO_2, 9.80.[1] The fuel dust is prepared from slack coal. This is dried in an inclined revolving cylinder,[2] the fuel gases passing first around and then through it.

The dried coal is ground in a Griffin mill, Fig. 414, to dust of which 90 per cent. passes a 100-mesh sieve, conveyed to a storage tank and fed into the burner. A 150-h.p. Corliss engine furnishes the power.

At the works of the Pennsylvania Steel Co., Lebanon, Pa., a magnetite concentrate (80 per cent. $<$30-mesh, Fe 60 to 64, S 1, Cu 0.2) is delivered by a roller-feed to the nodulizing cylinder which is 100 ft. long, 7 ft. diam., has a pitch 3/4 in. : 1 ft., and is lined with 9 in. of fire-brick (60 per cent. Al_2O_3). It is supported by four tires on friction rollers and makes two r.p.m.; the downward thrust is opposed by the rollers being placed at an angle, and by two sets of vertical rollers bearing against the tires. In 24 hr. the furnace treats 125 to 150 tons of concentrates; the S is reduced to 0.05 per cent., 80 per cent. of the nodules are $>$1/4 in. in size; the fuel consumption is 150 to 200 lb. dust per ton of nodules. The cost of nodulizing with labor at $1.30 is $0.75 per ton.

The following partial analyses of raw and sintered flue dust, Table 203, from a central Western blast-furnace plant represents an average for the month of November, 1908.

TABLE 203.—ANALYSES OF RAW AND SINTERED FLUE DUST.

Flue dust	SiO_2	Al_2O_3	Fe	Mn	CaO	MgO	Fixed C	P	S
Raw......... .. .	8.06	3.10	52.64	0.45	1.95	0.46	10.06	0.053	n.d.
Sintered........ .	11.25	4.73	60.05	0.53	2.80	0.70	2.81	0.065	0.205

THE DWIGHT-LLOYD PROCESS.—The Dwight-Lloyd straight-line sintering machine, described in connection with Fig. 272, was developed in the blast-roasting of sulphide ores. Within the last five years its sphere has been extended to the sintering of fine iron-bearing oxide material. As the latter contains no heat-furnishing constituent, this has been supplied by mixing coal with the ore. Experiments have shown that oxidized iron requires about 8 per cent. C to furnish ample heat for the production of a sinter. An excess over this amount is wasteful and harmful in that it prolongs unnecessarily the time required for sintering, raises the temperature from the desired sinter-point to that of fusion, and overheats the grates and other parts of the machine. Flue dust from an iron blast-furnace with Fe 46, SiO_2 9, C 14 to 17 per cent. gave without any additions a hard porous clinker; a magnetite concentrate with Fe 57 per cent. and SiO_2 9 to 10 per cent. required an addition of about 10 per cent. coal. At Birdsboro,

[1] Colby, *J. I. and St. I.*, 1906, III, 368.
[2] *Ibid.*

Pa., 8 per cent. C furnished all the required heat. The high percentage of C in blast-furnace flue dust has been reduced to the required amount by mixing the dust with iron ore or mill cinder. The S-content of "blue billy," averaging about 1.25 per cent., is reduced in the process from trace to 0.04 per cent. The ore-bed with oxidized iron is from 5 to 7 in. deep, while with blast-roasting of sulphide ore a depth of 4 in. is customary. The machine for sintering oxide ore is the same as the one used in blast-roasting. The daily capacity of a machine with pallets 42 in. wide is 100 tons. The cost of installation of a 100-ton machine is $12,000, and the cost of sintering $0.50 per ton charge. Klugh[1] in his paper gives photomicrographs which bring out the porous character of the Dwight-Lloyd sinter. The effect of the sinter on the smelting-power of the iron blast-furnace has been very favorable.

282. Briquetting Ore with Binder. NON-FERROUS ORES.—This mode of operating is very common with plants smelting lead, copper and other non-ferrous ores in the blast-furnace, but rare with iron-blastfurnace works. The raw material of the former has a much higher value than any iron ore and can therefore stand a more expensive treatment; besides, agglomerating by heating causes more loss by volatilization with lead ores than is permissible, and has not proved to be economical with copper and similar ores. Briquetting finely-divided sulphide ores for coarse-ore roasting furnaces has been only partially successful, as most sulphides decrepitate upon heating and cause briquettes to fall to pieces. Further, since only part of the water from a sulphide briquette can be moved by air-drying, the elimination of the remainder during the first stage of the roast also assists the disintegration.

The behavior of the different raw materials in briquetting is not uniform. Thus, blast-furnace flue dust which contains carbonaceous matter is difficult to mix intimately with the binder, and requires more or a stronger bond and a higher pressure than do the raw sulphides; on the other hand, with oxide or roasted sulphide and with flue dust from roasting-furnaces, the union is easily accomplished. A thorough mixing and kneading of ore and binder is of prime importance. When this is done, a comparatively low pressure will furnish a satisfactory brick. A brick will be stronger if it is made of coarse instead of fine particles, and strongest if coarse-, medium- and fine-sized are present in the right proportions. A brick made up of coarse ore (not over 1/2 in.) would require much bond in order to have the requisite strength, but it would dry easily and be porous; a brick made of fines alone could be molded with a small addition of bond, but would be too dense to behave well in the subsequent treatment; when practicable, coarse, medium and fine ores are mixed in proportions to furnish a brick that will require little bond and have at the same time sufficient interstitial spaces for the moisture to pass off readily and for the shrinkage, caused by drying and baking, to progress evenly. It is the usual practice simply to air-dry briquettes and then to charge them into the furnace; further treatment in drying-chambers at temperatures ranging from 100 to 180° C. greatly increases the hardness.

Kiln-burning, when applicable, gives the strongest brick. A brick to harden

[1] *loc. cit.*

in baking must contain either some clayey material, or some fusible component, such as lead or metallic sulphide, which will frit the particles together; coke has been added to a mixture containing little S in order to furnish the heat necessary for agglomeration. The burning has been carried on in heaps (see below, Port Pirie); ordinarily it is accomplished in up-draft kilns similar to those in common use with red brick manufacturers. "The brick are set up in arches about 30 courses high, the arches being 20 ft. long and the open portion about 9 courses high. Six or more arches are built side by side. A kiln 80 ft. long, 10 ft. wide and 20 ft. high will hold about 100,000 brick. Fires are built in the arches, cord wood being commonly used. It takes from 3 to 4 cords to fire a kiln. The kiln begins to burn after a few hours' firing, which now must cease. It burns for 2 or 3 days and is pulled down after it has been given some time to cool."[1]

Many binders have been suggested. Flue dust containing soluble metallic sulphates will harden after having been moistened. The most effective is $ZnSO_4$ of which 3 to 4 per cent. is known to have been sufficient; $FeSO_4$ and dilute H_2SO_4 have proved satisfactory; clay, pan[2] and concentrator[3] slimes have been used as has the waste molasses[4] of sugar plantations; waste sulphite liquor[5] has been recently advocated. The common binder is burnt lime which has been slaked and allowed to age, *i.e.*, to remain for some time under water, when it is stirred into water to form a creamy liquid. Incorporating the milk of lime into the ore by hand and making briquettes in the same manner, requires 8 to 10 per cent lime. Hand-work has about been given up, machinery having taken its place; with this an addition of $4\pm$ per cent. lime is sufficient. The hardening of the lime is due mainly to its combining with CO_2, but the simultaneous formation of a hydrosilicate is probable under the right conditions (see below, Schumacher process).

The leading machines used in briquetting are the ordinary brick-press, the presses of Chisholm, Boyd & White of Chicago, and of H. S. Mould of Pittsburgh, and the augur-machine of Chambers Bros., Philadelphia. Before taking them up, a few examples of briquetting by hand may be given.

HAND-BRIQUETTING AT PORT PIRIE, N. S. W.[6]—Slimes from the concentrating plant (PbS 24, ZnS 29.40, FeS_2 3.38, Fe_2O_3 4.17, FeO (in garnets) 1.03, MnO (in garnets and rhodonite) 6.66, Al_2O_3 5.40, CaO 3.40, SiO_2 22.90, Ag 0.060; total 100.48) are run from settling tanks in a semi-fluid condition into railroad cars, allowed to settle, and then dumped along the side of the track to form a layer 9 to 12 in. thick. After air-drying for several days they are cut with spades into blocks; these are further air-dried for 4+ days until they have become sufficiently hard to permit being handled, when they are loaded into

[1] Hofman, "Lead," 1908, p. 404.

[2] Church, *Eng. Min. J.*, 1885, XL, 125; *Tr. A. I. M. E.*, 1886–88, XV, 611.

[3] See Port Pirie below.

[4] Hofman, *op. cit.*, p. 405.

[5] *U. S. Dep't. Agriculture, Circular* No. 92, Washington, 1910.

[6] Horwood, *Tr. Austral. Inst. Min. Eng.*, 1903, IX[1], 106.
Delprat, *op. cit.*, 1907, XII, 15; *Eng. Min. J.*, 1907, LXXXIII, 317, 517.

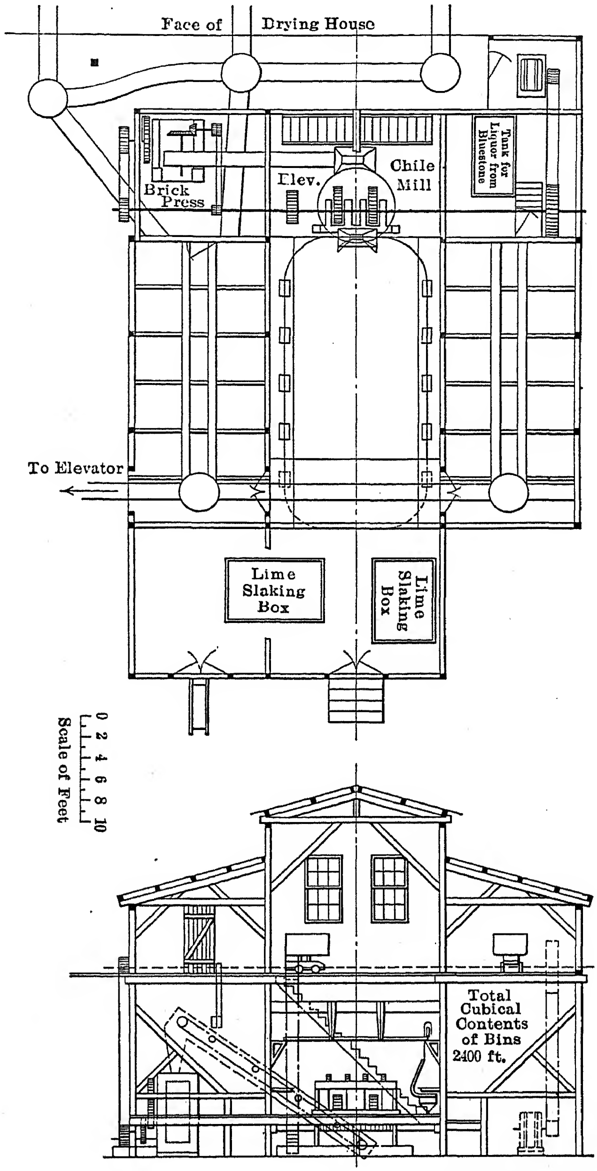
cars and hauled to the roast-yard. In handling, about 10 per cent. fines are made which are pugged with water and used as a cover for the roast-heaps. The blocks are built into heaps with brick channels, 12 ft. apart, along the bottom; the sides are plastered with pugged fines, and draft-holes are left open at the top. A heap is 200 to 250 ft. long, 20 to 22 ft. wide, 6 ft. 6 in. high, and holds 1,000 to 1,500 tons of brick. A wood fire is kindled in the brick channels and kept going for 1 to 2 days to start the roasting. A heap burns 10 to 15 days. When burnt, the loose part of the cover is removed and pugged with other fines. The heap-roasted slime forms a hard porous clinker with 6.5 to 8.5 per cent. S; temperature-measurements showed that sintering began at about 800° C. The loss in metal is high, that of Pb under 12.5 per cent., that of Ag under 5 per cent. In one instance, slime assayed before roasting Pb 17 per cent., Zn 16 per cent., S 12.5 per cent., Ag 17 oz. p. ton, and after roasting Pb 14 per cent., Zn 12.5 per cent., S 7 per cent., Ag 15.8 oz. p. ton.[1] The cheapness of the method compensates for the loss; annually 100,000 tons of slime are treated in this way.

Hand-briquetting at Santa Fé, N. M.[2]—Sands (30-mesh and smaller) and slimes from concentrating gold- and copper-bearing battery-pulp are mixed by hand with 10 per cent. freshly slaked lime and 1 to 2 per cent. wood-ashes, and then kneaded in a mechanical mixer. Bricks, weighing 14.5 lb. dry, are made in a hand-lever machine, transferred to four-wheel drying trucks, 10 by 16 ft., holding 280 brick, and run to the drying-yard, part of which is covered with a shed. From 600 to 900 brick are made with a press in 24 hr.; they remain three or four days on the trucks, are stacked in the yard 8 or 10 high, allowed to dry for one week, and then removed to the stock-pile on the feed-floor of the blast-furnace, forming heaps 30 or 40 bricks high. The breakage in handling is about 10 per cent.; the total cost of bricking is $1.50 per ton of brick.

Brick Press at the Selby Smelting and Lead Co., San Francisco.[3]—Plan and elevation of the briquetting plant are shown in Figs. 477 and 478. A perspective view of a four-mold Boyd brick-press is shown in Fig. 479. The ore arrives in cars on the upper floor, is discharged into steel bins which are on either side of the building and have three or four compartments. The floor of the bins projects 2 1/2 ft. forming a platform, and has holes 12×18 in. in front of the compartments; the holes are covered with a coarse screen to keep out pieces of iron. The charge is made up by shoveling the materials into two hopper-shaped buckets of 7 cu. ft. capacity suspended from an overhead trolley; the lime bond from the slaking boxes, amounting to 3+ per cent., is added last; the water of the slaked lime is generally sufficient to moisten the mixture which has the consistency of slightly watered brasque. The mass is worked in a 7-ft. wet-pan, having stationary edge-rollers, until it assumes a uniform color, and is then discharged by means of long-handled shovel into the boot of a 14-in., 8-ply rubber-belt elevator. The elevator has steel buckets at

[1] No mention is made of change in weight.
[2] Collins, *Tr. Inst. Min. Met.*, 1902–03, XII, 58, ill.
[3] Hofman, "Lead," 1908, pp. 400 and 402.
Bennett, *Eng. Min. J.*, 1904, LXXVIII, 425.

6-in. intervals which deliver the pulp to the hopper of the press through a canvas spout leading to the molds. The frame of the press, Fig. 479, consists of two cast-iron housings bolted together at the top and locked by the mold-table at



FIGS. 477 and 478.—Briquetting plant, Selby Smelting and Lead Works, San Francisco, Cal.

the bottom. It carries at the back the bearings for the shafting and forms the guides and upper supports for the two inner side-bars. These are connected at the top by a steel pin journaled in them, and at the bottom by bolts passing

through the mold-table. The toggles receive their movement at the middle joint from a connecting-rod operated by a crank-shaft, and exert a total pressure of 600 tons.

The cross-head with the upper steam-heated plungers is guided by the side-bars; the lower steam-heated adjustable plungers rise and fall. The press has a capacity of 20,000 to 30,000 brick of ordinary size in 10 hr. and requires 30 h.p.

In operating, the feed from the canvas spout is automatically shut off and the tops of the molds are smoothed when these have been filled; the mixture

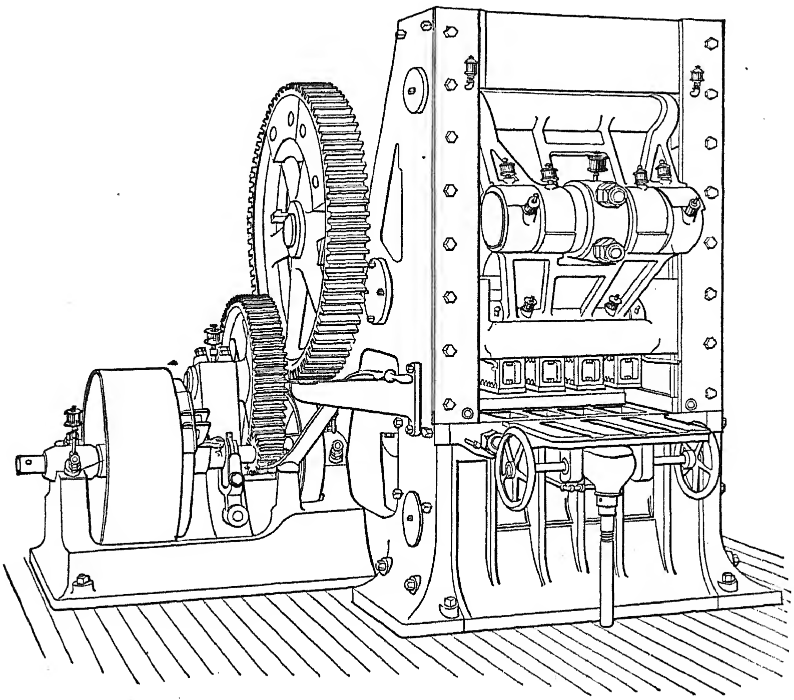


FIG. 479.—Four-mold Boyd brick press.

is then pressed by the straightening of the toggles; when these have been opened again, the lower plungers raise the bricks to the tops of the molds; the bricks are then pushed forward mechanically and removed by an attendant. The bricks are loaded on to cars, run into the dry-house, placed on shelves and air-dried for 3+ days. The plant has a daily capacity of 75 tons, is provided with a 50-h.p. induction motor, and requires 12 men, not counting those necessary to deliver the materials to the bins.

THE CHISHOLM, BOYD & WHITE PRESS.[1]—(Figs. 480 to 482).—The bri-

[1] Hofman, "Lead," 1908, p. 403.
Editor, *Eng. Min. J.*, 1899, LXVII, 563; 1901, LXXI, 278.
Editor, *Min. Sc. Press*, 1902, LXXXV, 21.
Circular published by the manufacturer.

quetting device consists of two parts, the auxiliary apparatus for preparing the mixture, Fig. 480, and the press proper with belt-conveyor, Figs. 481 and 482, to carry off the briquettes. The former comprises a lime-slaker, ore-feeder and conveyor-mixer; the total weight is 12,500 lb. The lime-slaker is a steel

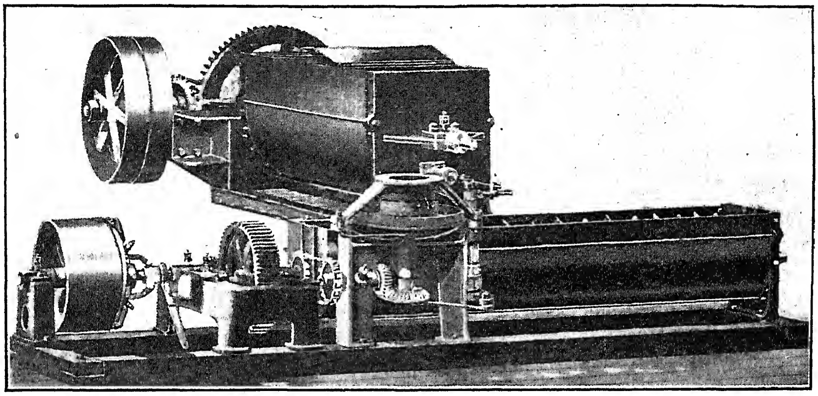


FIG. 480.—Chisholm, Boyd and White briquetting press.

trough, about 7 ft. long, divided into two chambers by a sheet of perforated metal placed crosswise acting as a strainer; a horizontal shaft, 4 in. sq., with detachable blades extending through it is driven by bevel gearing and tight and loose pulley; at the discharge-end is an adjustable brass pump, 6 in. in diam., which is driven in connection with the ore-feed and delivers the milk of lime to

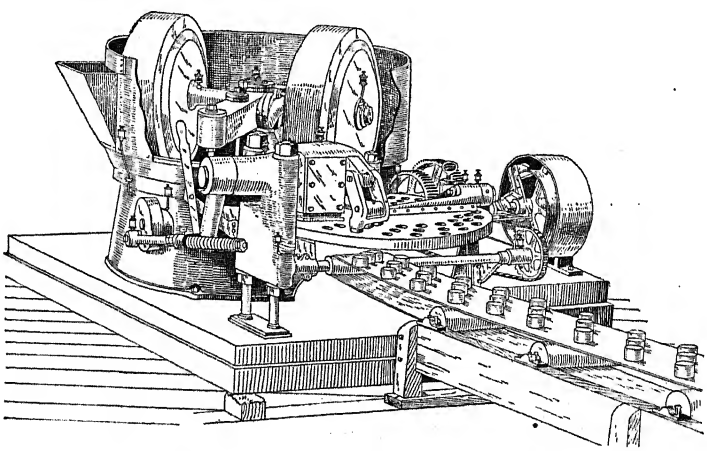


FIG. 481.—Chisholm, Boyd and White briquetting press.

the conveyor-mixer. The machine occupies a floor-space 4×12 ft. and requires 3 to 5 h.p. The automatic ore-feeder, supported by the frame of the conveyor-mixer, feeds by means of a revolving disc (carried on an upright shaft) through an adjustable telescopic ring. It is driven from the shaft of the conveyor-

mixer by a sprocket-wheel and link-belt. The mixer which receives measured amounts of ore and milk of lime is a trough of boiler-plate, 12 ft. long, with a 4-in. square steel shaft carrying detachable blades, set at an angle of 30° and spaced 3 in. apart. These pug the mixture and deliver it at the discharge-end either into a conveyor, if the press is on the same level, or into a chute if it is on the floor below. The mixer occupies a floor-space 17 ft.×4 ft. 6 in., is driven by a friction-clutch pulley and requires from 5 to 10 h.p.

The main parts of the press are a roller-mill, a molding disc, a re-press and an ejector. The cast-iron pan of the roller-mill, mounted on a circular cast-iron frame, is 7 ft. in diam.; it is lined on the path of the roller with white-iron plates 2 in. thick, and on one side it is cut out to admit the molding disc. The rollers

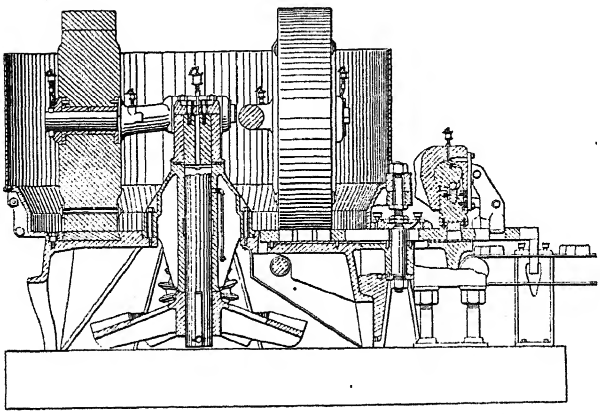


FIG. 482.—Chisholm, Boyd and White briquetting press.

are 48 in. in diam., 12-in. on the face, have chilled-iron tires 4 in. thick, and weigh 6,000 lb. each; they are connected by 5-in. crank-shaped steel arms to the cross-beam wnich is keyed to the central 6-in. steel spindle, the latter being driven from below by bevel-gearing. The cross-beam carries two pairs of plows which turn the mixture from both sides of the rollers into their path.

The mold disc is a cast-iron plate, 5 ft. in diam. and 2 1/2 in. thick, with center just outside of the pan. It has three rows of circular holes, 48 in all, forming molds 3 or 4 in. in diam. at the top and slightly larger at the bottom. Underneath the molds, in the path of the rollers, is a heavy steel plate which forms the bottom and supports the mold disc while the rollers are passing over it. The movement of the disc is effected by a radial pawl-arm journaled at the center and attached at the outer end to a connecting-rod with crank gear. The arm has a round steel pawl with hardened toe which drops into one of the holes. When the arm is moved forward the pawl draws the disc around 15°, and drops into the next hole. The movement of the rollers and disc is so adjusted that the rollers will travel repeatedly over a set of molds during their passage through the pan and thus firmly compact the briquettes. This is carried further by re-pressing.

The re-press, attached to the base-frame, consists of two heavy cast-iron

frames which are held together by 3-in. steel bolts and enclose part of the mold disc. Through the upper frame extends a 6-in. steel shaft with roller and toggle for operating three re-press plungers. The re-press is actuated by a connecting-rod provided with a relief-spring; this same motion operates the ejector, so that at the same time three briquettes are re-pressed, and three ejected on to the belt-conveyor. While re-press- and ejector-plungers enter the molds, the mold-disc remains stationary. The machine occupies a floor-space 10×12 ft., is 7 ft. high, and weighs 30,000 lb. It is driven by a friction-clutch pulley, 30 in. in diam. and 12 in. face, making 265 r.p.m., and requires 25 to 30 h.p. when in operation, and 40 for starting. The capacity of the machine at Anaconda[1] was 125 tons in 24 hr.; eight men including one foreman were required to operate it and to handle the briquettes. The cost of briquetting ranges from $0.65 to $1.50 per ton.

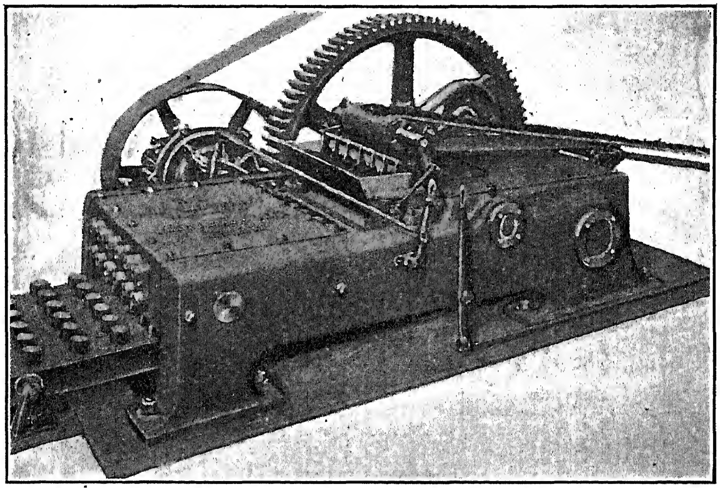


FIG. 483.—H. S. Mould Co. briquetting press.

THE H. S. MOULD CO'S PRESS[2] (Figs. 483 to 485).—This machine, as the preceding one, consists of the auxiliary apparatus for preparing the mixture, viz., the lime-slaker, lime-pump, dust-feed and mixer; the feed-belt which conveys the mixture to the plunger-press; the press proper; and the delivery-belt for the briquettes. One line-shaft furnishes the power for the entire plant. There are two lime-slakers which are used alternately; they are connected at the bottom by screened openings (to be closed by a cut-off) with the central chamber which forms the reservoir for the lime-pump. The shafts of the slakers are driven independently by fast and loose pulleys, the shaft of the slaker to the right extending across both vats. The pump has an adjustable stroke

[1] Austin, *Tr. A. I. M. E.*, 1906, XXXVII, 460.
[2] Editor, *Eng. Min. J.*, 1901, LXXII, 724.
Editor, *Iron Trade Rev.*, Nov. 14, 1901, p. 61
Editor, *Min. Rep.*, 1902, XLV, 49.

and is driven by a chain from one of the mixer-shafts, while the other actuates the ore-feed, the discharge of which is regulated by a rod and handwheel.

The mixer has two shafts provided with pugging arms. It is driven through gearing by a friction-pulley connected by belting with the line-shaft. The feed-

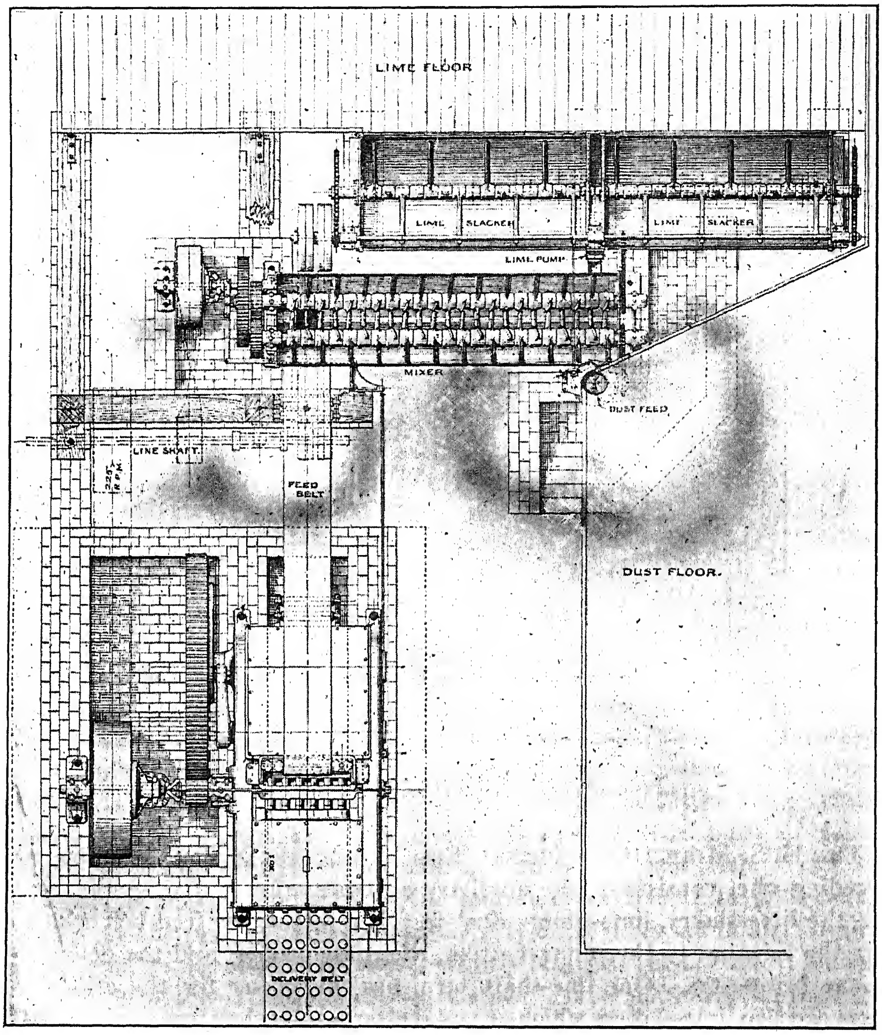


FIG. 484.—H. S. Mould Co. briquetting press.

belt, supported by four rollers, receives its motion from a pulley on the pulley-shaft of the mixer. The press is of the plunger type, working on the same principle as the open-mold Exeter press used to dry-press some lignites (Figs. 104 and 105). As the plunger is drawn back, a certain amount of mixture falls into the mold and is compressed by the forward stroke; thus the mold becomes filled

with briquettes, every new briquette being compressed against the briquettes in the mold, of which one is ejected by every forward stroke. The mixture to be suited for the press should not contain more than 7 per cent. water. The machine shown has six plungers 3 in. in diam. and makes briquettes 3 in. long. With 25 r.p.m. it produces 150 briquettes, or 1,080 cu. ft. molded material. Its rated capacity is 100 tons in 10 hr., and it requires 5 men to operate it. The wear on the chilled-iron bushings and the pistons is considerable. The plunger-pressure necessary, in one case, was 25 lb. for flue dust, 15 for mixed charge, and 10 for fines.

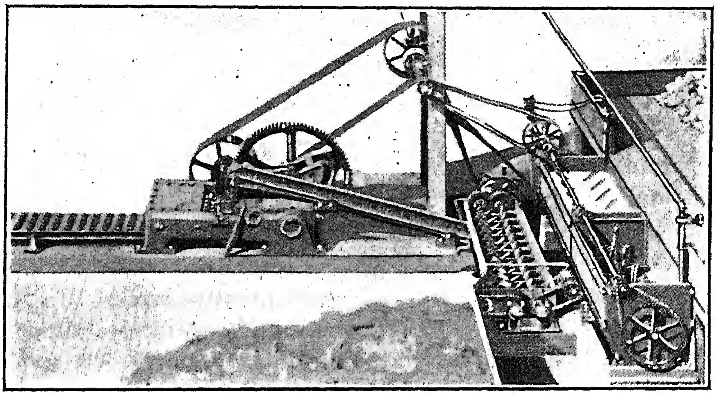


FIG. 485.—H. S. Mould Co. briquetting press.

THE AUGUR END-CUT BRICK MACHINE (Chambers Bros., Philadelphia).—This machine is shown in Fig. 234, and its mode of working discussed in §159. At Anaconda[1] the briquette mixture is made up of 1/3 screened first-class ore, 1/3 concentrate and 1/3 concentrator slime; to this is added 5 per cent. washed coke from the ash-pit droppings of the reverberatory smelting-furnaces, as it makes the briquette porous and replaces part of the coke necessary in the blast-furnace. The materials are drawn from storage-bins in right amounts and delivered by conveyor-belts to the hopper of the machine; the necessary water is introduced at the same time. The machine produces in 24 hr. with nine men per shift, 840 tons of briquettes weighing 5 to 10 lb. each. They are conveyed by one 10-in. rubber belt to a second one which runs parallel to the long side of a building, steam-heated in winter; this contains 36 drop-bottom storage hoppers, 2 ft. 6 in. wide at top, 2 ft. wide at bottom, 3 ft. 4 in. long and 2 ft. 6 in. deep. Deflecting-boards scoup the briquettes into the hoppers, each of which holds about 1 ton. From the hoppers the briquettes are drawn into charge-cars passing underneath. The briquettes, it will be seen, are not handled and are used when only imperfectly dried. While they crumble somewhat, they hold together sufficiently for smelting with the rest of the charge.

[1] Austin, *Tr. A. I. M. E.*, 1906, XXXVII, 460.

Briquetting Iron Ores.—A raw material of low value, such as finely divided iron ore, cannot usually bear the cost of briquetting with a binder, further, the grade of the ore is reduced. Some of the leading binders used and suggested are clay, lime, lime and silica, iron blast-furnace slag, cement, soda, sodium, calcium and magnesium chlorides, etc. Organic binders, such as coking coal, rosin, tar, pitch, oil residuum are too expensive.

Since the bricking and baking of clay-bearing iron ore has been successful, the addition of clay to iron ore appears promising. Briquettes can be made in that way, but the addition of a refractory bond which has to be slagged later on is hardly economical. In exceptional cases where the iron ore is so high-grade slag-forming material has to be added to the charge to insure having enough slag to cover the iron, the use of clay may be advantageous. Thus, in Pittsburgh, low-grade argillaceous Pennsylvania limonite used to be added to the rich Lake Superior iron ores to furnish the amount of slag and Al_2O_3 necessary for the satisfactory working of the blast-furnace. Briquettes of fine rich iron ore and clay would have served the same purpose.

The use of lime with pure iron ore cannot have a satisfactory effect; if the ore contains finely divided SiO_2 the case is different, it resembles the combined use of CaO and SiO_2 in the manufacture of lime-sand brick.[1] In the original Schumacher process,[2] lime and quartz are ground separately, some lime being added to the quartz to absorb the water this may contain. The two are incorporated into the ore, the mixture is briquetted, and the briquettes are steamed until they become hard. The hardening is said to be due to SiO_2 and $Ca(OH)_2$ combining to a hydrous silicate. The amounts of lime and quartz needed vary with the character of the ore, but are small: 91 per cent. purple ore, 4 to 5 per cent. lime and 3 to 4 per cent. quartz gave good results at one plant; another required only 1.5 per cent. quartz; 88 per cent. flue dust, 10 per cent. lime and 5 per cent. quartz worked satisfactorily at a third plant. The briquette presses used weigh 22 tons, exert a pressure of 4,300 to 5,700 lb. per sq. in. and make 11,000 12-lb. briquettes in 10 hr.

The new Schumacher process[3] consists in mixing flue dust with from 5 to 10 per cent. $MgCl_2$- or $CaCl_2$-solution ($=0.25-2.00$ per cent. $MgCl_2$ or $CaCl_2$), which causes the mixture to set and harden in from 15 to 60 min. At the works of the Cambria Steel Co., Johnstown, Pa., a mixture of flue dust and fine Mahoning ore is moistened with a 30-per cent. $CaCl_2$ solution, compressed to cylindrical briquettes 8 in. in diam. and 8 in. high. A chemical reaction takes place in the briquettes, causing them to become warmed. The briquettes contain 27 per cent. of voids (absorbing 11.5 per cent. water) and have a compressive strength of 445 lb. per sq. in. The cost of briquetting is said to be less than $0.30 per ton.

[1] Eckels, E. C., "Cements, Limes and Plasters," Wiley, New York, 1905, p. 130. Verein der Kalksandsteinfabriken, E. V., "Der Kalksandstein," Berlin, 1909.

[2] *Stahl u. Eisen* 1908, XXVIII, 321, 1190.

[3] Richards, *Tr. A. I. M. E.*, 1912, XLIII, 387.

Granulated iron blast-furnace slag[1] with cementing properties is used in amounts up to 10 per cent. by the Scoria Co.[2] The slag is steamed, crushed and mixed with the ore in an edge-roller mill, the mixture briquetted and the briquettes steam-hardened.

Cement which crumbles[3] when heated alone to about 450° C. appears to give good briquettes especially when these are steam-hardened. At Le Creuzot[4] briquettes of purple ore with from 3 to 6 per cent. cement made under a pressure of 7,000 lb. per sq. in. and steam-hardened, resisted a temperature of 480 to 540°C. under a pressure equivalent to that which they would have to endure after having passed through three-quarters the height of a blast-furnace. Brick made with lime fell to pieces. Lime, soda and salt have been used at South Chicago.[5]

The briquetting of iron ore and coking coal, and then coking the briquettes, appears promising, but the amount of ore which even a good coking coal can bind, and furnish a strong coke, is comparatively small; further, the reduction of the Fe_2O_3 in the blast-furnace will be effected by solid C, which is wasteful; on the whole, a good coking coal is generally too valuable to be used for such purposes; the same is generally the case with the other organic bonds.[6] A recent example of briquetting with coal is that of the Alexandrowski Iron Works in the Ural Mts.[7] where the coal could not stand over 7 per cent. ore, if the coke was to retain its original crushing strength.

283. Briquetting Metal Scrap.—The process of compacting chips, borings, etc., of brass, bronze, cast-iron, cast steel, etc., without a binder originated with A. Ronay of Buda-Pest. The divided metal particles suitably prepared are placed in a mold and subjected to a hydraulic pressure of about 30,000 lb. per sq. in. The briquette is allowed to remain under pressure sufficient time to expel air and water, which would have a tendency to weaken the bond, and thus to permit the particles to come into the necessary intimate contact. The particles before being charged into the hopper of the press are dried, if necessary, and freed from dust and dirt by the sucking action of an exhaust-fan. Turnings may have to be passed through disintegrating rolls to reduce their bulk. A press exerts its force both from above and below the charge. In charging cast-iron borings into the cupola, the loss in metal is, according to Moldenke,[8] 50 per cent.; if they are boxed, the loss ranges from 8 to 12 per cent.; with briquettes the loss has been brought down to 6 per cent. In melting pig iron straight, the loss should not exceed 3 per cent.; in charging as much as 80 per cent. briquettes, the loss is about the same as with pig iron.

[1] *Stahl u. Eisen*, 1908, XXVIII, 321, 1190.
[2] *Stahl u. Eisen*, 1908, XXVIII, 324; 1909, XXIX, 240.
At the Krupp works, of Rheinhausen, *Iron Age*, 1913, XCI, 901.
[3] Bumby, *J. West Scotland Iron and Steel Inst.*, 1901–02, IX, 14.
[4] *Stahl u. Eisen*, 1902, XXII, 457.
[5] *Iron Age*, 1902, Dec. 11, p. 8.
[6] Rosin used by Edison, *Iron Age*, 1897, Oct. 28, p. 7.
[7] *Stahl u. Eisen*, 1906 XXVIII, 475.
[8] *loc. cit.*

B. METALS—ALLOYS

284. Mechanical Treatment of Metal in General.[1]—The shaping of metal, which is the object of this class of operations, is accomplished by mechanical pressure that is greater than the elastic limit of the metal. During the application of the pressure the crystals glide along cleavage-planes in the direction of the strain (flow of metal, §12), or they, as well as the original cleavages, are broken. A rearrangement takes place, the manner of which depends upon the temperature of the piece and the duration of the work. A metal must be ductile, if it is to be shaped mechanically; the larger the difference between its elastic limit and tensile strength, the greater is the deformation it will endure. As a rule, both the elastic limit and the tensile strength of a metal are lowered when it is heated; in some instances the two are raised for a short period when the temperature rises above normal. The lowering of the elastic limit by a rise of temperature permits working a metal with an application of less power than at ordinary temperature. If the elastic limit is lowered more quickly than the tensile strength, the range within which the metal can be worked is increased. The variations of these two mechanical properties due to temperature are not always regular. This explains the fact that some metals can be worked only within a small range of temperature. Working a metal cold causes a rise of both the elastic limit and the tensile strength. As the rise with the former is greater than with the latter, the difference between the two properties decreases as the work progresses, and the metal becomes brittle and may break. By annealing the metal before the danger-line is approached, *i.e.*, heating to a temperature approximately fixed for each metal, the original difference between elastic limit and tensile strength is restored, and the metal has regained its original ductility. If the work is carried on at this annealing temperature, the operation cannot have any influence upon the ductility. It will, however, increase the specific gravity and tensile strength in that it compacts the metal by increasing the cohesion and adhesion of the crystals, and by closing possible blow-holes. In some cases it is desired to bring the elastic limit and tensile strength of a metal nearer each other in order to increase the stiffness as, *e.g.*, cold-rolling steel springs or steel shaftings.[2]

The mechanical pressure exerted for shaping a metal may be applied in three ways: (1) Slowly and releasing it almost as soon as the metal has yielded to it, as in rolling or wire-drawing. (2) Gradually and allowing it to remain active for a short time when the metal has yielded, as in pressing. (3) Quickly and releasing it before the metal has fully yielded, as in hammering.

[1] Hauer, J. von, "Hüttenwesen-maschinen," Felix, Leipsic, 1876, Supplement, 1887.

Jordan, S., "Album to the Course of Lectures on Metallurgy," Baudry, Paris, 1877.

Guettier, A., "Le Forgeron Mécanicien," Bernardt & Co., Paris, 1887.

Fischer, H., "Die Bearbeitung der Metalle," Baumgärtner, Leipsic, 1890.

Ledebur, A., "Lehrbuch der Mechanisch-metallurgischen Technologie," Vieweg & Sohn, Brunswick, 1905.

[2] Goerens, *Ferrum*, 1912–13, I, 65, 137.

285. Rolling of Metal in General.[1]—The apparatus used for shaping metal by rolling is called a rolling-mill. The first was built in the 16th century. Its essential feature is a set (or train) of at least two rolls, usually mounted with axes parallel one above the other and revolved at approximately the same speed in opposite directions. The friction between the rolls and a piece of metal causes the former to grip the metal and draw it in, when it will be reduced to a thickness equal to the distance between the rolls and increased in length, but not much in width. The flow of metal caused by rolling is well illustrated by the curved lines of Hollenberg[2] shown in Fig. 486. He bored holes into a bar in the direction in which the rolls were to exert their pressure, inserted pieces of wire, rolled the metal, and then planed it down. In cold-rolling the curvature of the wires along the surface of the bar was the greater, in hot rolling along the center. For the same reason, a bar with the inside hotter than the outside will show convex ends after rolling, and concave (fish-tail) ends when the outside is hotter, as the hotter metal flows more quickly than the cooler.

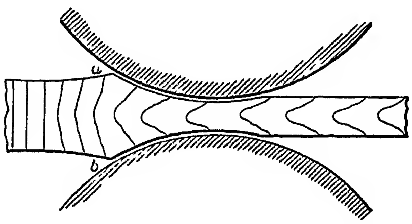


FIG. 486.—Flow of metal in rolling.

In special rolling processes, such as the Mannesmann[3] for rolling weldless tubes, the rolls move in the same direction, but are set at an angle to one another.

Rolls are either plane or grooved. With plane rolls only plates or sheets can be rolled. If the metal is to receive a special shape by rolling, grooves are formed in the upper and lower roll to furnish together the desired sections. The space between two plane rolls or between the two grooves of a pair of rolls is called a "pass." A piece of metal having traveled through the space is said to have made a pass. As the amount of mechanical work to be done is great,

[1] Daelen, Hollenberg, and Dickman, *Verh. Beförd. Gewerbefleisses*, 1869, XLIII, Supplement; Tunner, P., and Pearse, J. P., "Roll-turning," Van Nostrand, New York 1869; Blass, *Stahl u Eisen*, 1882, II, 189, 283; Angström, C. A., "Walzen- und Kalieber-construction für Eisenwalzwerke," *Oest. Jahrb.*, 1891, XXXIX, 353; Beckert, Th., "Leitfaden der Eisenhüttenkunde," Springer, Berlin, 1900, Part III; Geuze, L., "Laminage der Fer et de l'Acier," Béranger, Paris, 1900; Brovort, A., "Das Kaliberieren der Walzen," Felix, Leipsic, 1903; Ledebur, A., "Lehrbuch der Mechanisch-metallurgischen Technologie," Vieweg, Brunswick, 1905; Kirchberg E., "Grundzüge der Walzenkalibrierung," Ruhfus, Dortmund, 1905; Puppe, J., "Experimental Investigations on the Power Required to Drive Rolling Mills," Griffin & Co., London, 1910; Harbord, F. W., and Hall, J. W., "Steel," Lippincott, Philadelphia, 1911, Vol. II.

[2] *Stahl u. Eisen*, 1883, III, 121.
Puppe, *op. cit.*, 1909, XXIX, 162.

[3] Leobner, H., "Ueber das Mannesmann'sche Röhrwalzverfahren," Richter, Hamburg, 1897.
Siemens, *Brit. Assoc. Rep.*, 1888, p. 804; *Engineering*, 1888, XLVI, 291.
Wedding, *Tr. A. I. M. E.*, 1890–91, XXIX, 384; *J. I. and St. I.*, 1890, II, 551.
Reulaux, *Iron Age*, 1890, XLVI, 128.
Gordon, *Eng. Min. J.*, 1890, XLIX, 676.
Editor, *Iron Age*, 1907, LXXX, 553.
Gruber, *Stahl u. Eisen*, 1910, XXX, 1449.
Mayer, *op. cit.*, p. 105.
Springer, *Iron Age*, 1910, LXXXVI, 612.

it is impossible to obtain a final reduction by a single pass; the general scheme of rolling is therefore to have a succession of decreasing passes which reduce the thickness of the bar and, as stated above, at the same time increase mainly the length.

Rolling-mills are two-high, three-high, or universal. A two-high mill is either non-reversing, when the direction of travel of the rolls remains unchanged, or it is reversing, when the direction is changed. In a non-reversing (pull-over, pass-over) mill, Fig. 487, the metal is fed only from one side (the roller's) and returned, after having made a pass, from the other (the catcher's) over the top of the roll. The loss in time and the labor of lifting are avoided in a reversing mill. Here the engine is reversed after the metal has made one pass, and the latter run through the mill in the opposite direction. The gain in time and labor is coupled with the drawback that reversing an engine, especially when at full

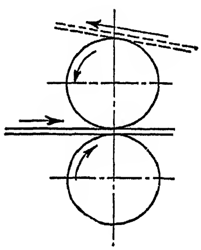


FIG. 487. Two-high rolls.

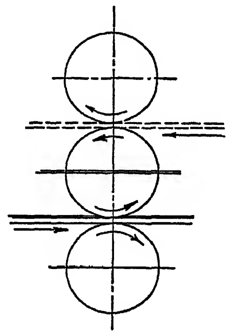


FIG. 488. Three-high rolls.

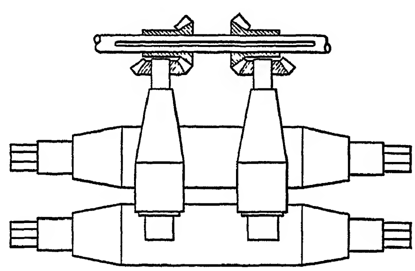


FIG. 489.—Universal rolls.

speed, subjects it to severe strains. This difficulty is overcome by the continuous three-high mill invented by John Fritz in 1857. Here, Fig. 488, the top and bottom rolls travel in directions opposite to that of the middle roll. The disadvantages are, that power is necessary to raise the metal above the middle roll, and that, with the exception of some plate-mills, the rolls are not adjustable. The universal mill,[1] which was devised by Daelen in 1855, aims to even the edges of a flat piece while it is being rolled. It consists, Fig. 489, of a pair of horizontal reversible rolls backed by a pair of vertical adjustable rolls which are also reversing; with the former the thickness of the metal is reduced in the usual way and its length increased; the latter compress it sidewise and thus control the spreading of a piece and give it an even edge. Universal mills have been made three-high[2] with a pair of vertical rolls on either side of the horizontal rolls.

286. Examples of Typical Rolling-mills.—A few examples of rolling-mills used in working iron and steel may serve to illustrate the leading types.

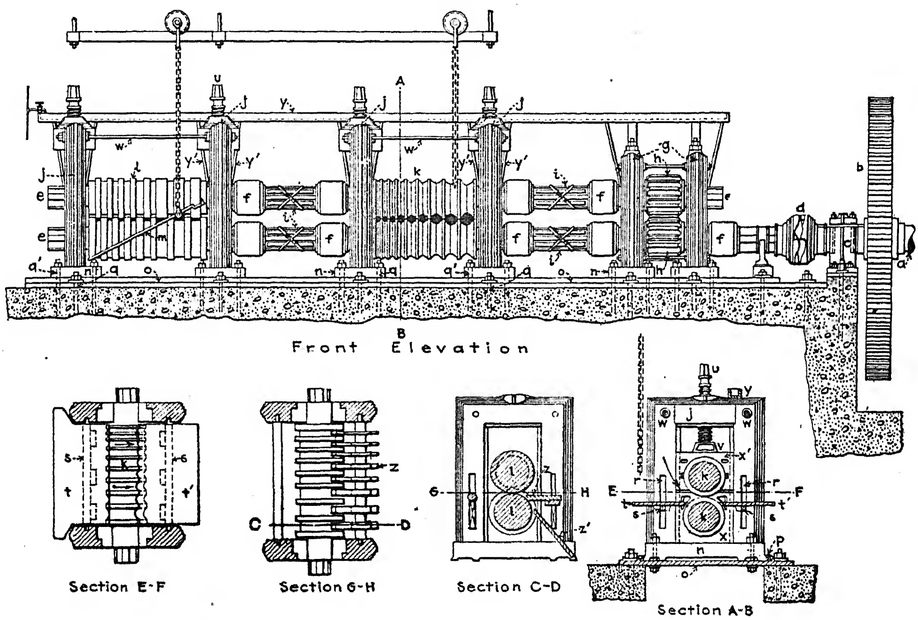
TWO-HIGH NON-REVERSING TWO-STAND ROD-MILL.—Figs. 490 to 494 represent a two-high non-reversing two-stand mill for rough-rolling a squeezed

[1] Reddy, *Iron Trade Rev.*, 1910, XLVII, 1115.

[2] Reddy, *Iron Trade Rev.*, 1910, XLVII, 317.

or hammered puddle-ball into muck-bar, and for finishing the muck-bar into finished bar.

The engine-shaft, *a*, is coupled with the lower of a pair of straight-tooth spur wheels, *h*, held in position by the spur-wheel housing, *g*; their shafts are connected with those of the first pair of rolls, *k*, by star-shaped connecting-spindles, *i*, and coupling-boxes, *f*; and these are joined in the same manner to the second pair of rolls, *l*. The rolls are held in their correct positions by the



FIGS. 490–494.—Two-high non-reversing two-stand rod mill.

a, Engine-shaft; *b*, fly wheel; *c*, pillow-block for engine-shaft; *d*, coupling with star-shaped spindle; *e*, star-shaped coupling (wobbler) with four driving points; *f*, coupling-box; *g*, spur-wheel housing; *h*, straight-tooth spur-wheels; *i*, star-shaped connecting-spindle; *i′*, flutes with wooden strips strapped on to maintain distance between coupling-boxes; *j*, roll-housing; *k*, roughing-rolls with Gothic passes; *l*, finishing-rolls with box-passes; *m*, hand-lever with fulcrum attached to chain; *n*, base of housing; *o*, bed-plate of train; *p*, wedges; *q* and *q′*, anchor-bolts; *r*, recesses; *s*, cramp-bars; *t* and *t′*, fore-plate and rest-plate; *u*, screw for upper bearing-box; *v*, breaker; *w*, tie-bars; *x*, lower bearing-box resting upon bottom of standard; *x′*, upper bearing-box resting on block carried by lower box; *y* and *y′*, water-trough and -pipe; *z*, guide; *z′*, guard.

roll-housings, *j*; these are secured at the bottom to their bases, *n*, and held at the top by tie-bars, *w*; the bases are bolted, *q′*, to the bed-plate, *o*, and this in turn is anchored, *q*, to the foundation. The trough, *y*, with descending pipes, *y′*, furnishes the water necessary for cooling the journals of the rolls. Each pair of rolls is grooved by a succession of decreasing passes; the top-rolls are not adjustable. The lower bearing-box, *x*, rests on the bottom of the standard and the upper box, *x′*, on blocks carried by the lower box; the upper box is held down by the screw, *u*, to be turned with a key having one or more curved handles

(not shown); the screw bears on a breaker, *v*, a small hollow casting which breaks and gives way in case a piece too large or too cool is put between the rolls. The fore-plate, *t*, Figs 491 and 494, in front of the roughing rolls, *k*, serves to support the piece when it is presented to the rolls; it is often provided with ribs[1] in front of the collars of the rolls to guide the piece and protect the collars from injury; the rest-plate, *t'*, forms the support of the piece as it leaves the rolls. Instead of having ribs on the fore-plate, special guides, *z* (Figs. 492 and 493), as in front of the closed box-passes of the finishing-rolls, *l*, are provided to direct the piece into the grooves.

Fig. 493 shows a guard, *z'*, which peels out a piece, should this become wedged in a pass and wrapped around the lower roll (collared) instead of passing out straight. In order that all collaring shall take place on the lower and not on the upper roll, the grooves and collars are placed in the lower roll; this arrangement also permits the piece to enter the groove more easily. If the arrangement were reversed, the guard would have to be suspended (hanging guard) and held in its position against the groove by a counter-weight.

The mill intended for rolling muck-bar and finishing-bar has rolls 18 to 20 in. in diam., makes 50 to 60 r p.m., and requires about 60 h.p.

TWO-HIGH REVERSING TWO-STAND (ROUGHING AND FINISHING) PLATE-MILL.—The illustrations, Figs. 495–497, represent two pairs of plate-rolls driven by a reversing engine; one pair serves for roughing, the other for finishing the plate. Each pair of rolls has stationary feed-tables with live rollers. The plate reduced to the required size in the roughing rolls is delivered by the live rollers on to a pair of rails over which it is pulled mechanically on to the feed-table of the finishing-rolls. The engine-shaft is coupled to the lower of a pair of pinions with herringbone or double-helical teeth.[2] The lower pinion is connected with the lower rolls of the two stands by the usual spindles and coupling-boxes. In the roughing train the upper roll needs considerable adjustment which brings the connecting-spindle much out of line. In order to overcome this difficulty, the ends of the spindle are formed into a spherical wobbler, and the pinion- and roll-housings are placed sufficiently apart so that the angle which the spindle makes with the horizontal does not exceed 15°. The bearing-boxes of the lower roll rest on the bottoms of the standards; the movable boxes of the upper rolls are forced by bell-crank levers and counter-weights against the screws passing through the tops of the housings. The adjustment is accomplished by a worm-gear; the adjustment screws carrying spirally-toothed wheels are rotated by two endless screws on a single shaft which is turned by handwheels. In the finishing-stand there is no balancing of the upper roll. Its box rests upon that of the lower roll, and the sheet upon entering the mill, lifts the upper roll, and this drops down again after the pass has been made; the height to which it may be raised is regulated as in the roughing roll. The live rollers on either side of a stand are driven by a single reversing engine through a line-shaft, spur-gears, counter-shaft and bevel gears.

[1] Munker, *Stahl u. Eisen.*, 1911, XXXI, 1620; *Iron Age*, 1912, LXXXIX, 245.

[2] Day, *Iron Age*, 1911, LXXXVIII, 1364.

TWO-HIGH REVERSING BLOOMING-MILL (Figs. 498–501).—This form of mill permits roughing down (blooming, cogging) ingots with one set of rolls to all sizes of squares and flats within the range for which the mill has been constructed. The main differences between this mill and the roughing-stand of the preceding reversible plate-mill are the form and adjustment of rolls, and the

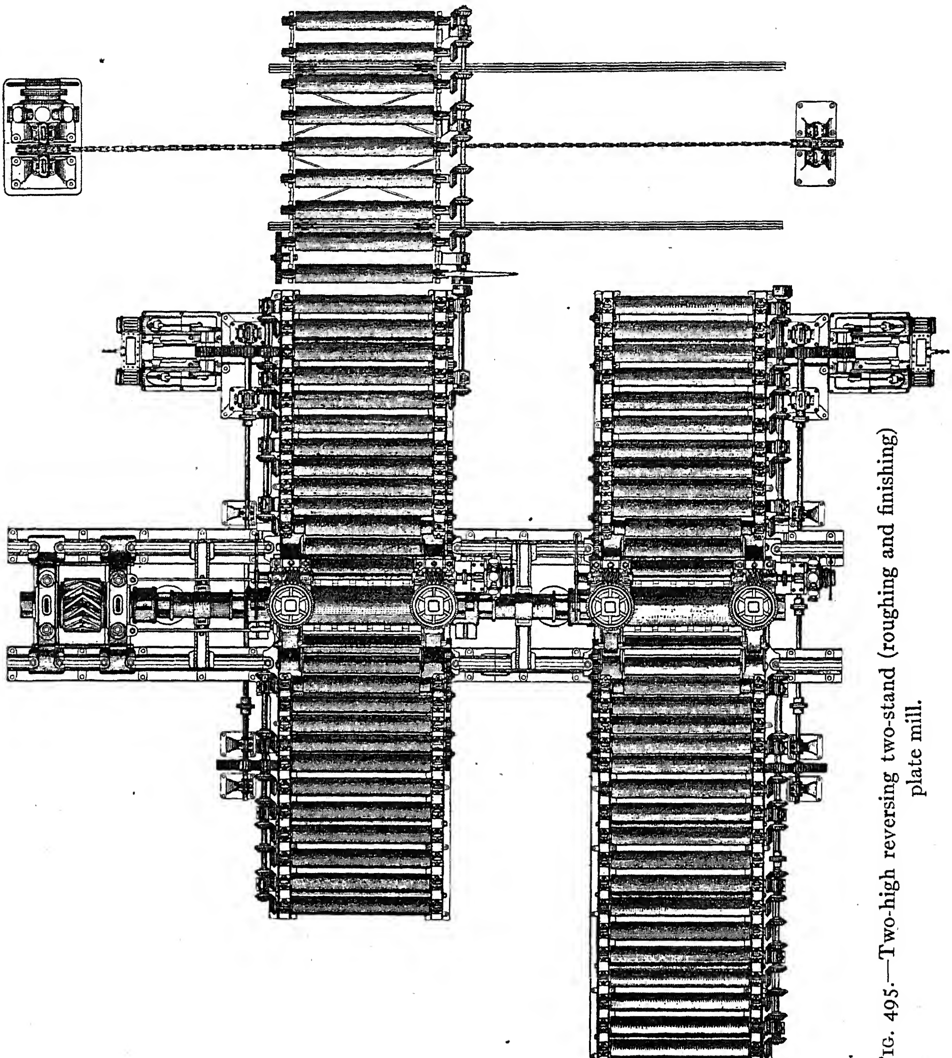


FIG. 495.—Two-high reversing two-stand (roughing and finishing) plate mill.

manipulation of the ingot. The blooming-rolls have four box-passes, a wide one at either end and two narrower ones near the center. The upper rolls are adjusted by means of a hydraulic screwing-gear. A rack is moved to and fro by a direct-acting horizontal hydraulic cylinder that is attached to the tops of the housings of the helical driving-wheels. It gears with a pinion the shaft of

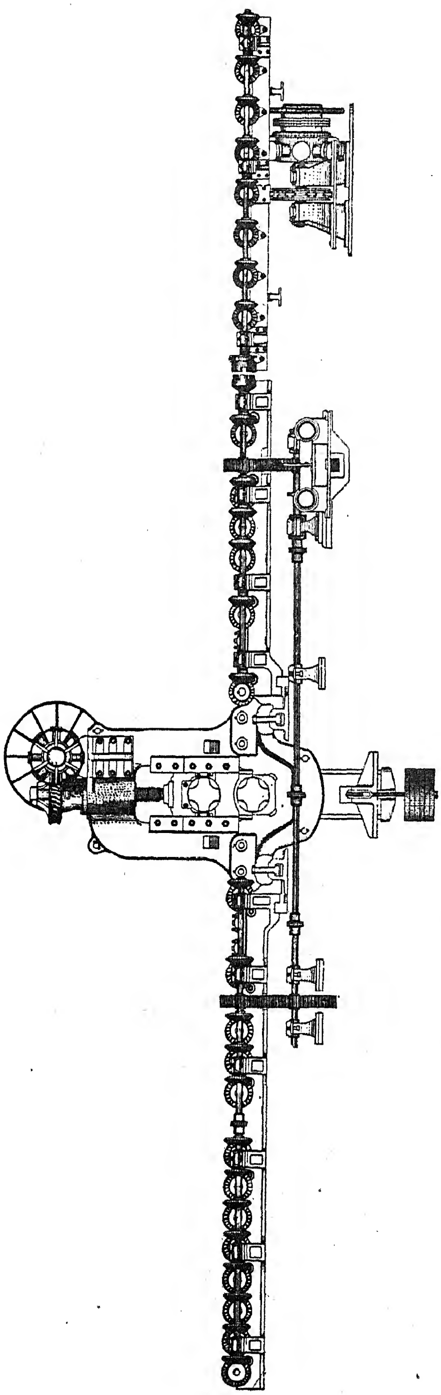


FIG. 496.—Two-high reversing two-stand (roughing and finishing) plate mill.

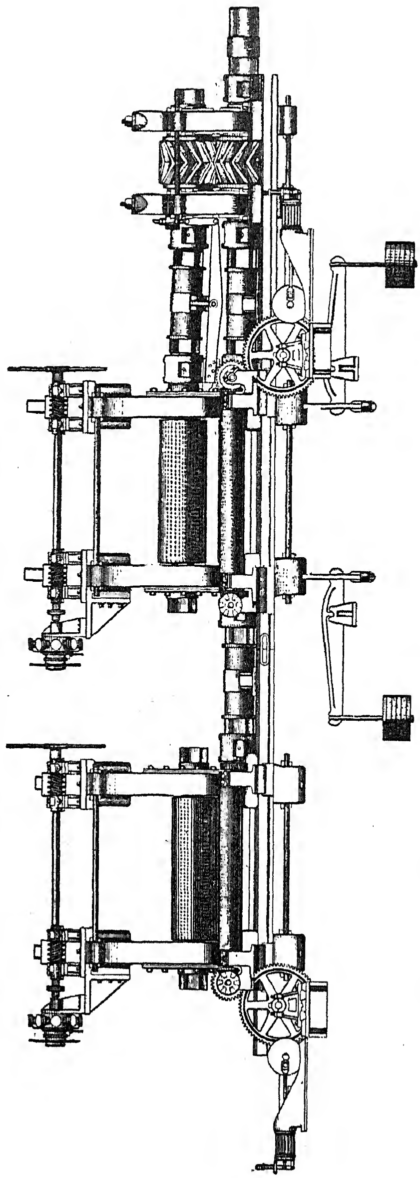


FIG. 497.—Two-high reversing two-stand (roughing and finishing) plate mill.

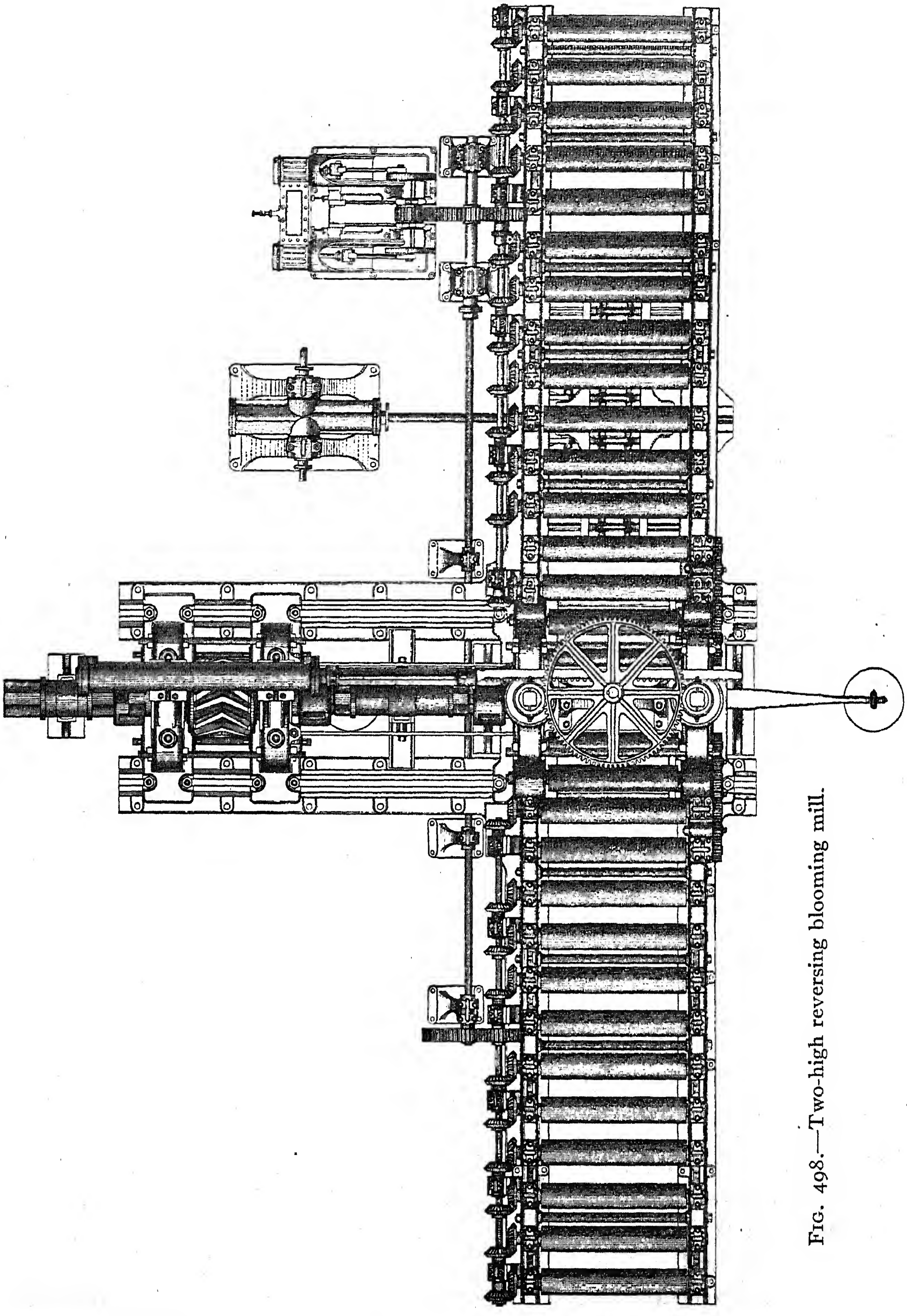


Fig. 498.—Two-high reversing blooming mill.

which carries a large cog-wheel meshing with pinion-wheels keyed to the adjusting screws. The manipulator has four fingers (go-devils), for pushing the ingot over the live rollers from one pass to another, and for tilting it. The fingers, Fig. 501, are bolted to a carriage which is moved to and fro parallel with the axes of the rolls by a horizontal hydraulic cylinder. The platform upon which the carriage travels is supported by a piston moving up and down in a vertical hydraulic cylinder, the platform being steadied by a deep flange on the lower side which moves telescope-fashion in a socket. Thus the four fingers can be moved to and fro, and up and down between the live rollers.

A 40-in. mill is generally used; the rolls are 30 and 34 in. in diam. and 96 in. long. The size of passes ranges from 24 to 4 in. (the r.p.m. varies from 10 to

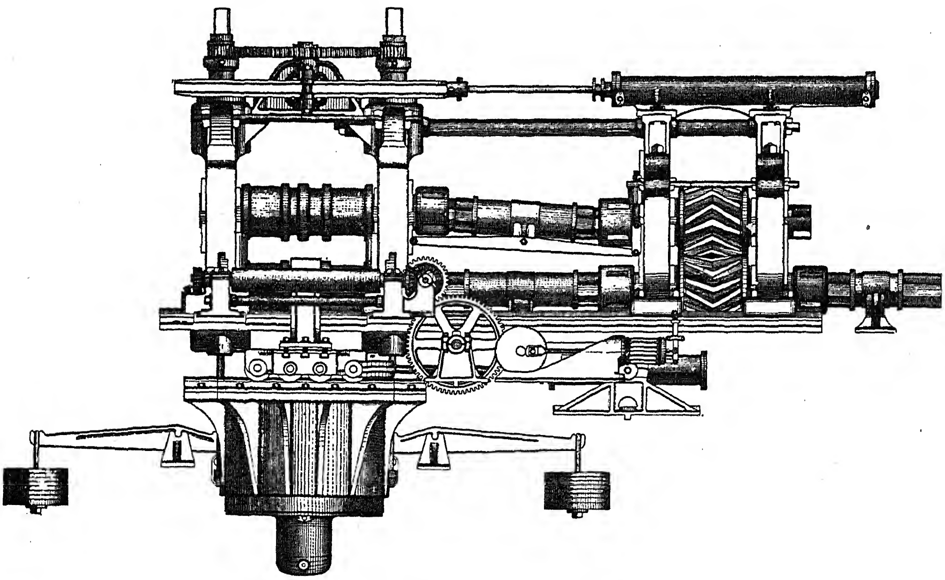


FIG. 499.—Two-high reversing blooming mill.

125). The hydraulic cylinder for operating the screws has become obsolete, having been replaced by a 50-h.p. motor. The mill-table rollers are driven by a 14×14 in. reversing engine; the peripheral speed of the rollers is from 300 to 400 ft. per min. The capacity of the mill is 1,200 tons of billets 4×4 in., or 1,800 tons of slabs and blooms. The horse-power is 6,000. Three men (roller, engineer and manipulator) operate the mill.

THREE-HIGH BLOOMING-MILL (Figs. 502–504).—The rolls in this mill are fixed, and the mill produces blooms of a given size. The engine-shaft is coupled with the central helical driving wheel, and its shaft is connected with that of the middle roll which is then driven direct, while the upper and lower rolls are driven by the spur-wheels above and below the central wheel. The bearings rest one upon the other and are held down by the top pin. The manipulators have the same form as in the preceding mill. The tables, with the live

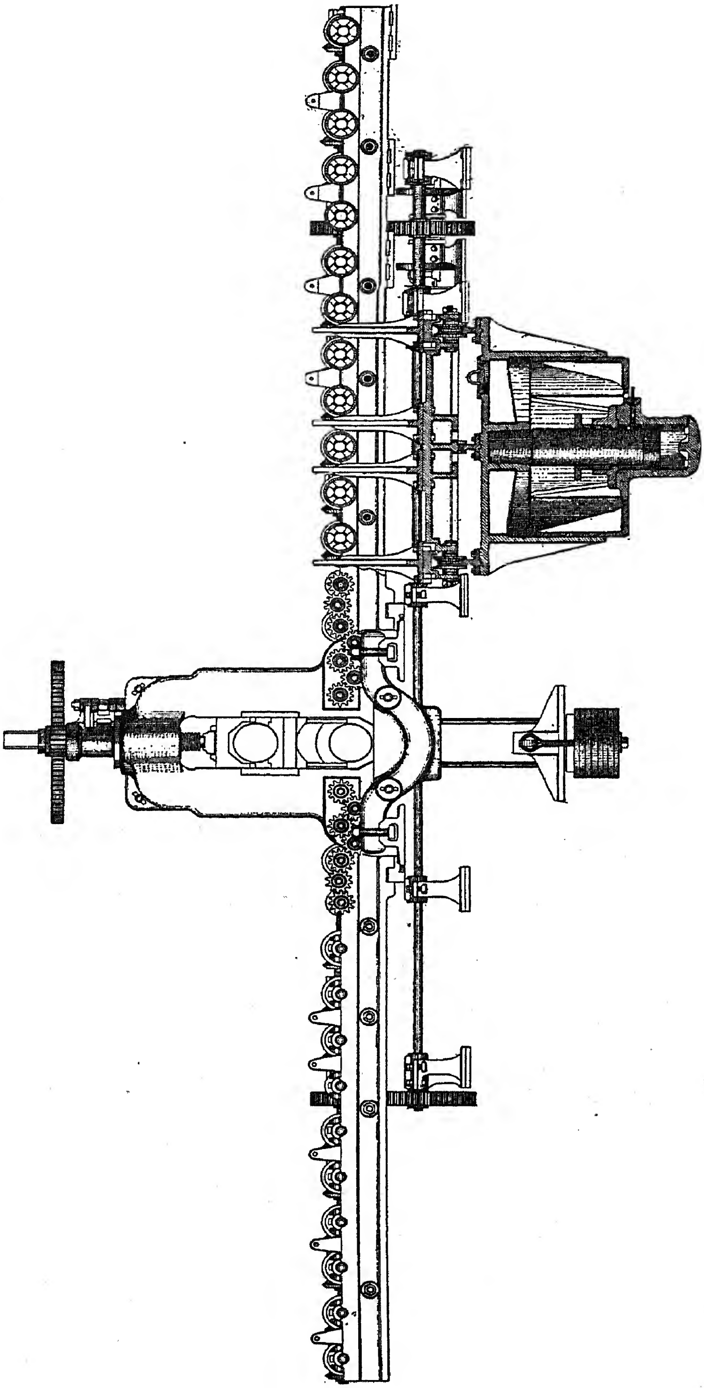


FIG. 500.—Two-high reversing blooming mill.

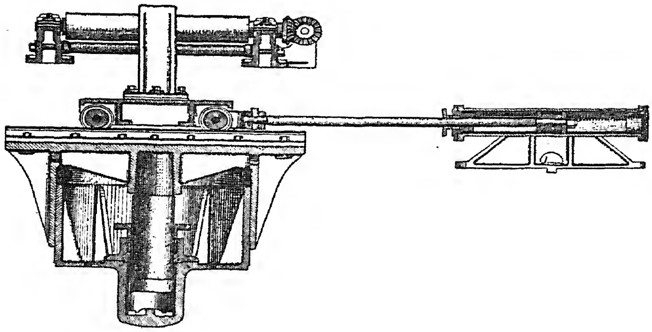


FIG. 501.—Two-high reversing blooming mill.

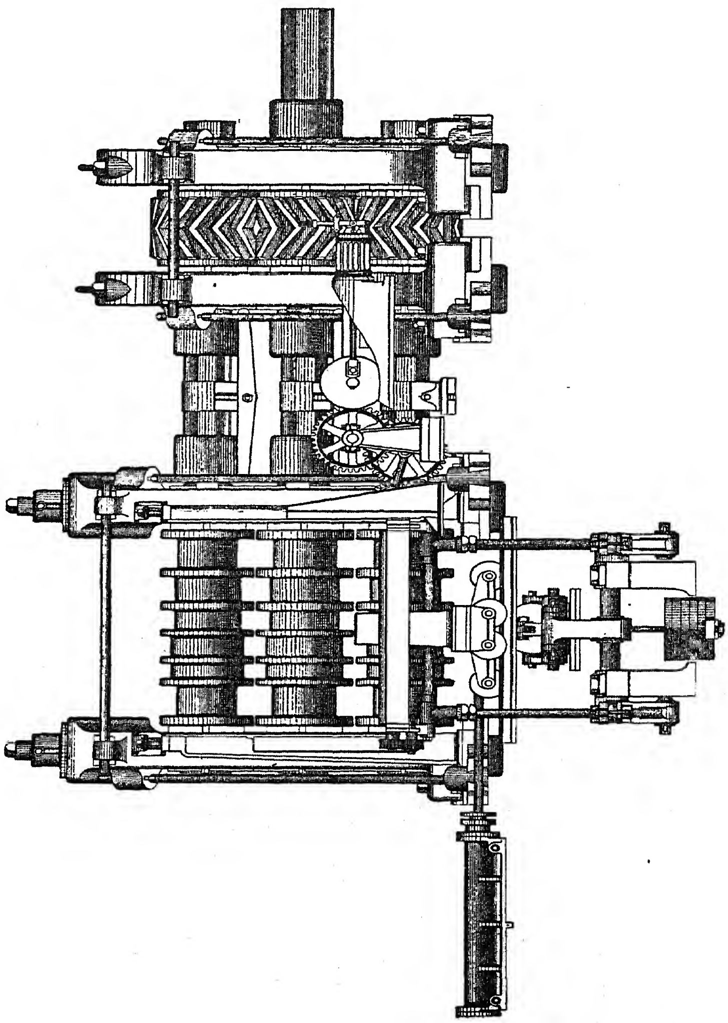


FIG. 502.—Three-high blooming mill.

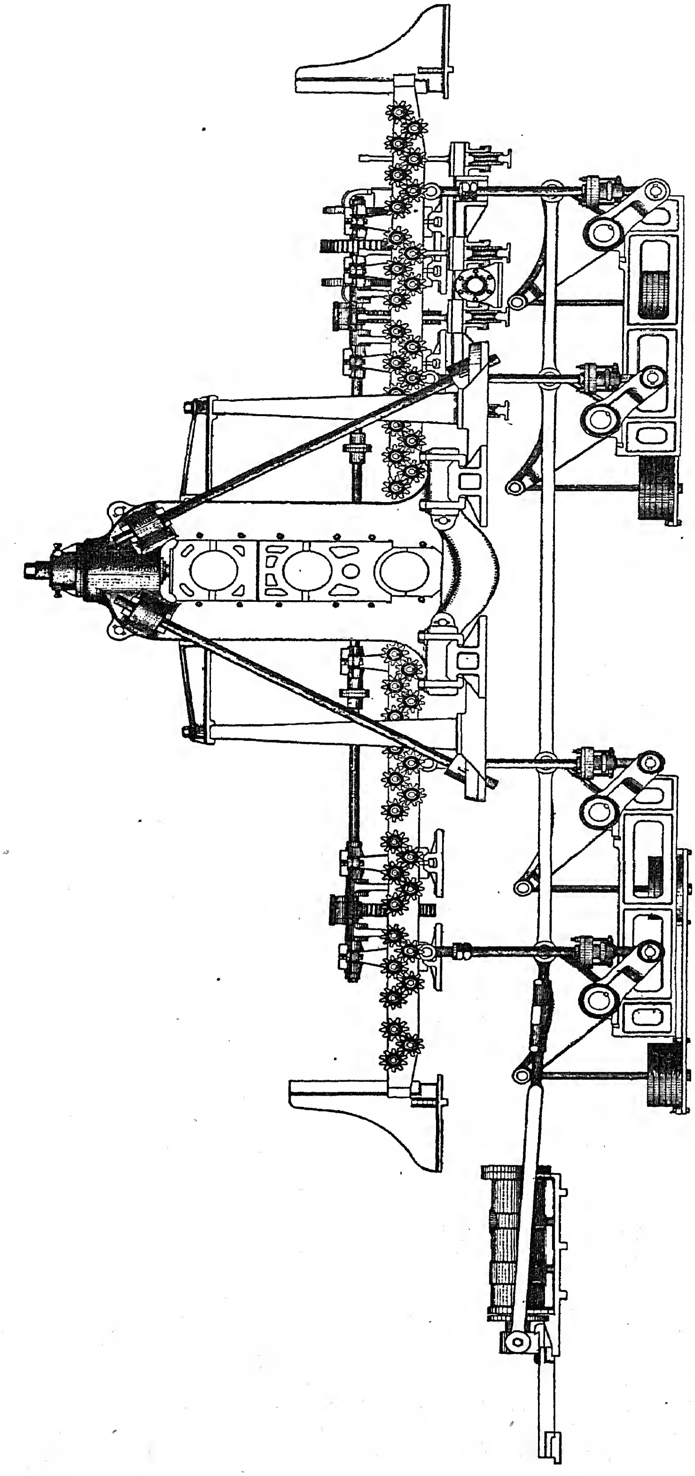


Fig. 503.—Three-high blooming mill.

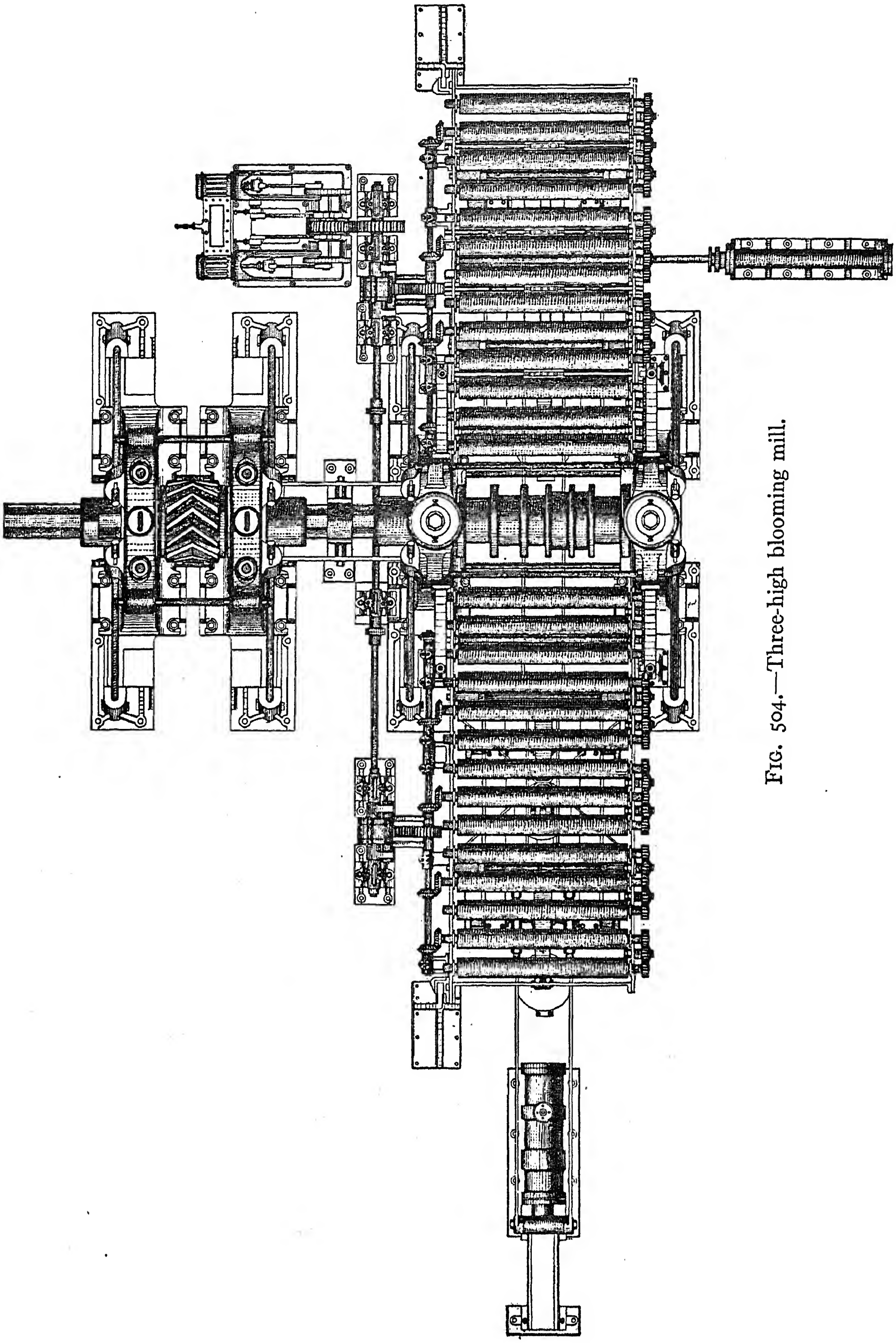


FIG. 504.—Three-high blooming mill.

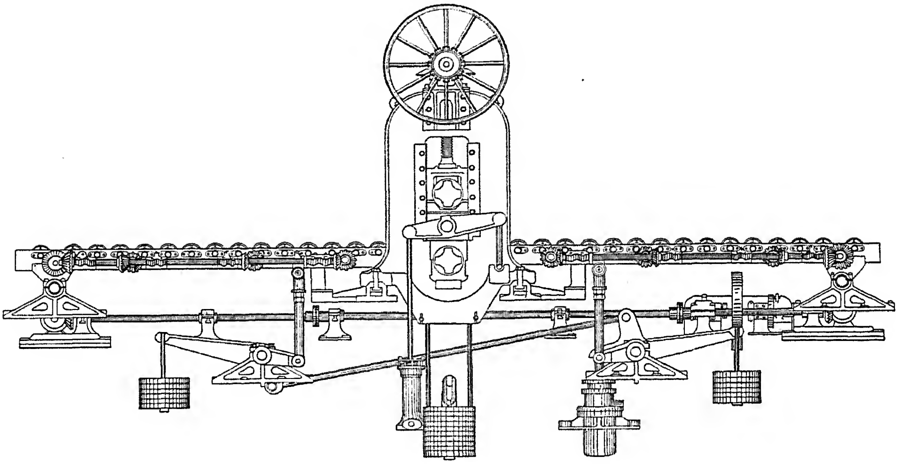


FIG. 505.—Three-high (Lauth) plate mill.

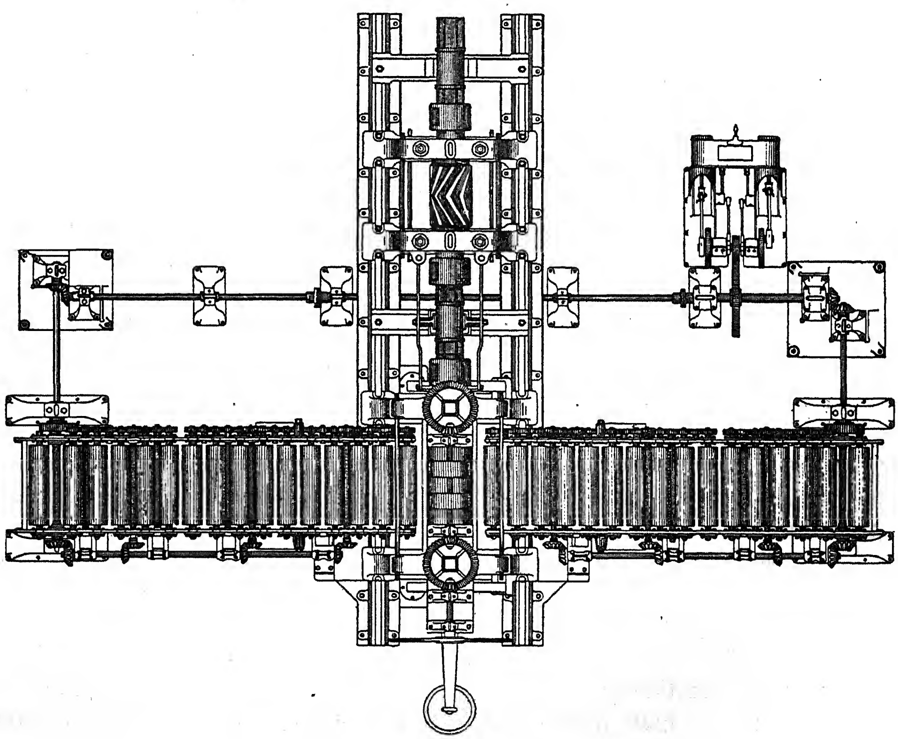


FIG. 506.—Three-high (Lauth) plate mill.

rollers and ingot tumbler at either end, are raised and lowered by the steam or hydraulic cylinder on the left. The live rollers are rotated by bevel-gearing; the main shaft sets going through spur-gear nine rollers, each of which communicates the rotary motion to two or five rollers on the opposite side.

A 40-in mill of this type is generally used in a rail-mill. The diameter of rolls is 30 and 34 in.; the length 74 in.; the size of passes ranges from 18 to 9 in.; the rolls are of cast steel; they make 65 to 70 r.p.m. The lifting cylinder of the tables has a plunger 12 in. in diam. and a stroke of 5 ft. 6 in.; the table rollers are driven by a 12×12 in. engine; the hydraulic manipulator cylinder has a 6.5-in. plunger with a 6-ft. stroke. The capacity of the mill is 2,000 tons of 9×9 in. rail blooms in 24 hr.; three men operate the mill; total h.p. required is 6,000.

THREE-HIGH (LAUTH) PLATE-MILL (Figs. 505–507).—This is a standard type of large-size plate-mill. The characteristics are a fixed large bottom-roll; a balanced top-roll of the same diameter which is adjusted by top-screws that

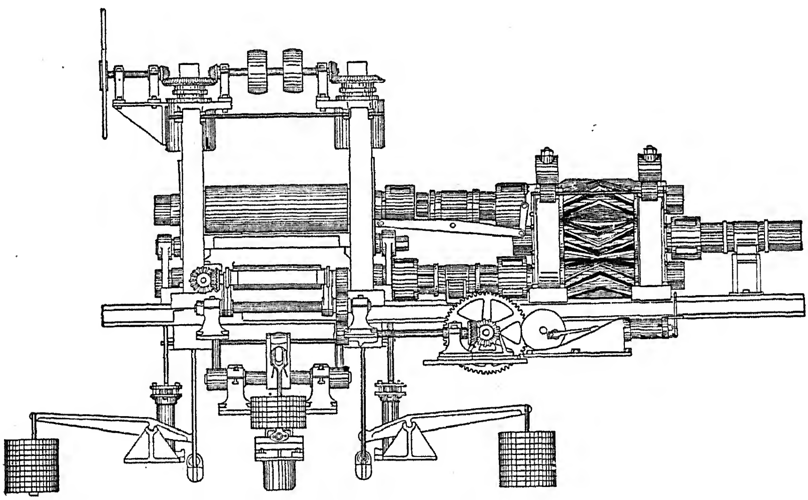


FIG. 507.—Three-high (Lauth) plate-mill.

are rotated from one side by a handwheel through bevel gears; a small center-roll, about two-thirds the diameter of the larger rolls, which is raised and lowered by a hydraulic or steam cylinder, so as to receive alternately from the upper and lower rolls support for its entire length against the bending stresses set up when the plate is being rolled. The lifting-tables with live rollers are similar to those in the preceding mill excepting that there are no manipulators. The center-pinion is coupled to the engine-shaft, and is usually smaller than the top- and bottom-pinions. The pinion- and roller-housings are some distance apart to keep the angle of the inclined connecting-spindle within the fixed limit of 15°. The mill combines the advantage of a small-diameter roll, *i.e.*, of increasing the length of a plate more quickly than a large diameter, with the strength of the large diameter roll.

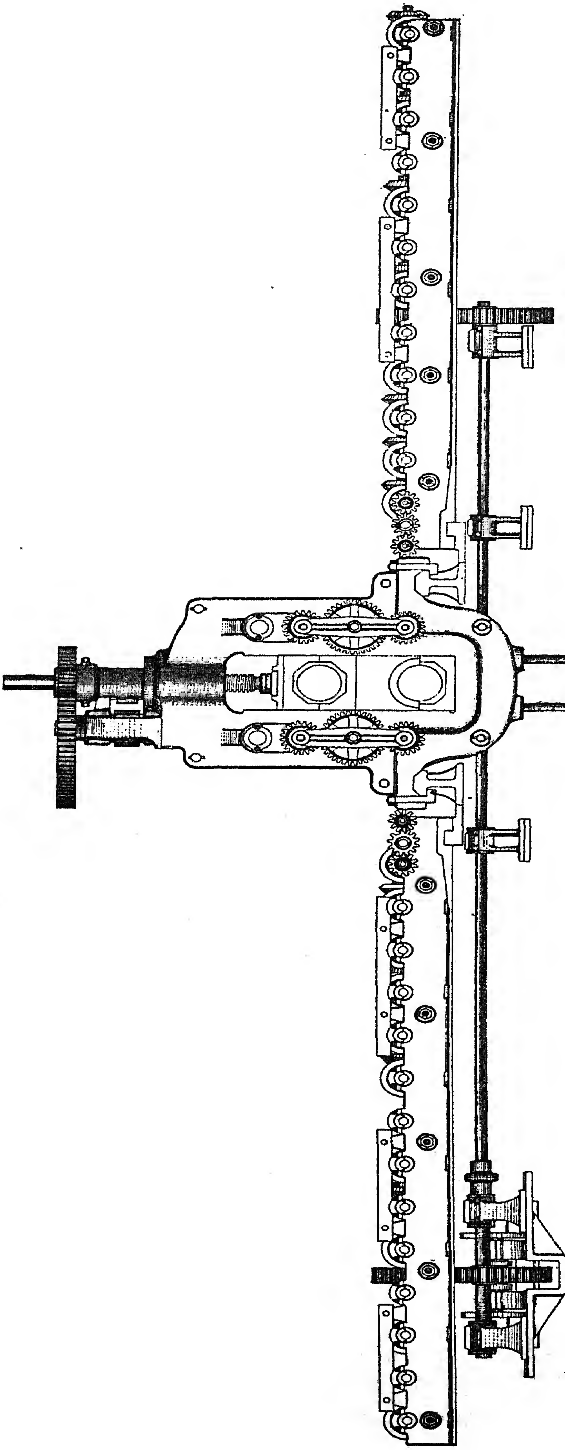


FIG. 508.—Two-high universal mill.

The diameter of the large rolls is usually 34 in., that of the small 13.5 in.; the rolls are chilled cast-iron; they make about 80 r.p.m. The hydraulic cylinders for the middle rolls have 12-in. plungers. The hydraulic lifting cylinder

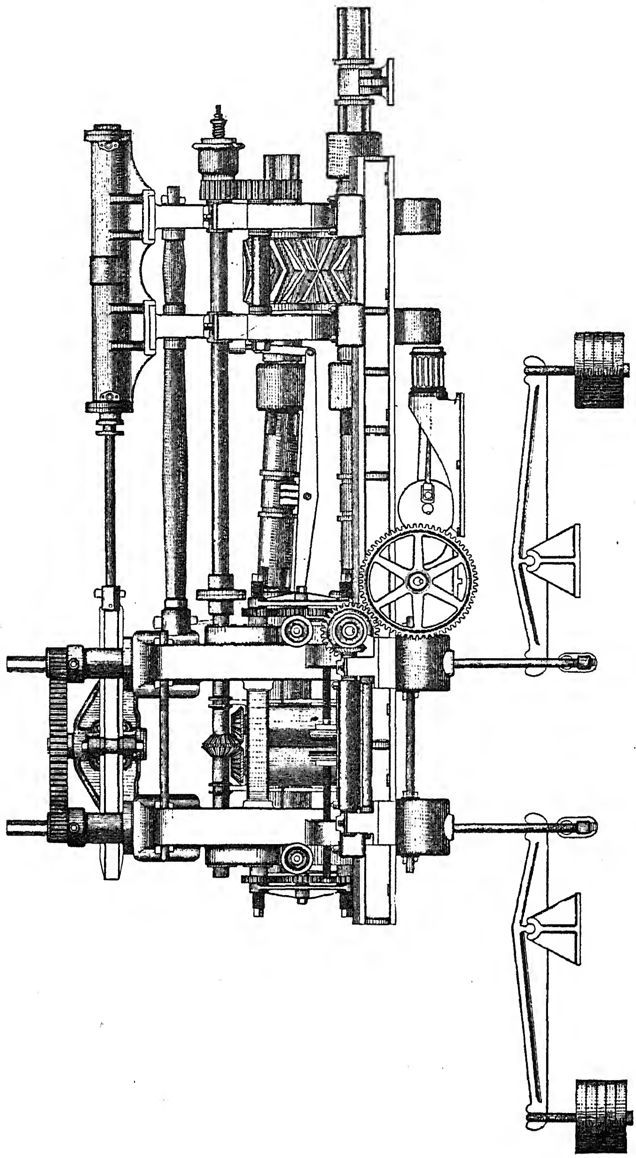


FIG. 509.—Two-high universal mill.

of the table has a 12-in. plunger with a 5-ft. stroke. The rollers are driven by 14×14-in reversing engines; the peripheral speed of the rollers is from 300 to 350 ft. per min. The capacity of the mill is 14,000 long tons per month, the power required 6,000 h.p.; four men operate the mill.

Two-high Universal Mill (Figs. 508–510).—This mill serves principally for rolling steel plates of special sizes which are long, comparatively narrow (8 to 42 in.) and thin (0.25 in.). The plates are finished with square and straight

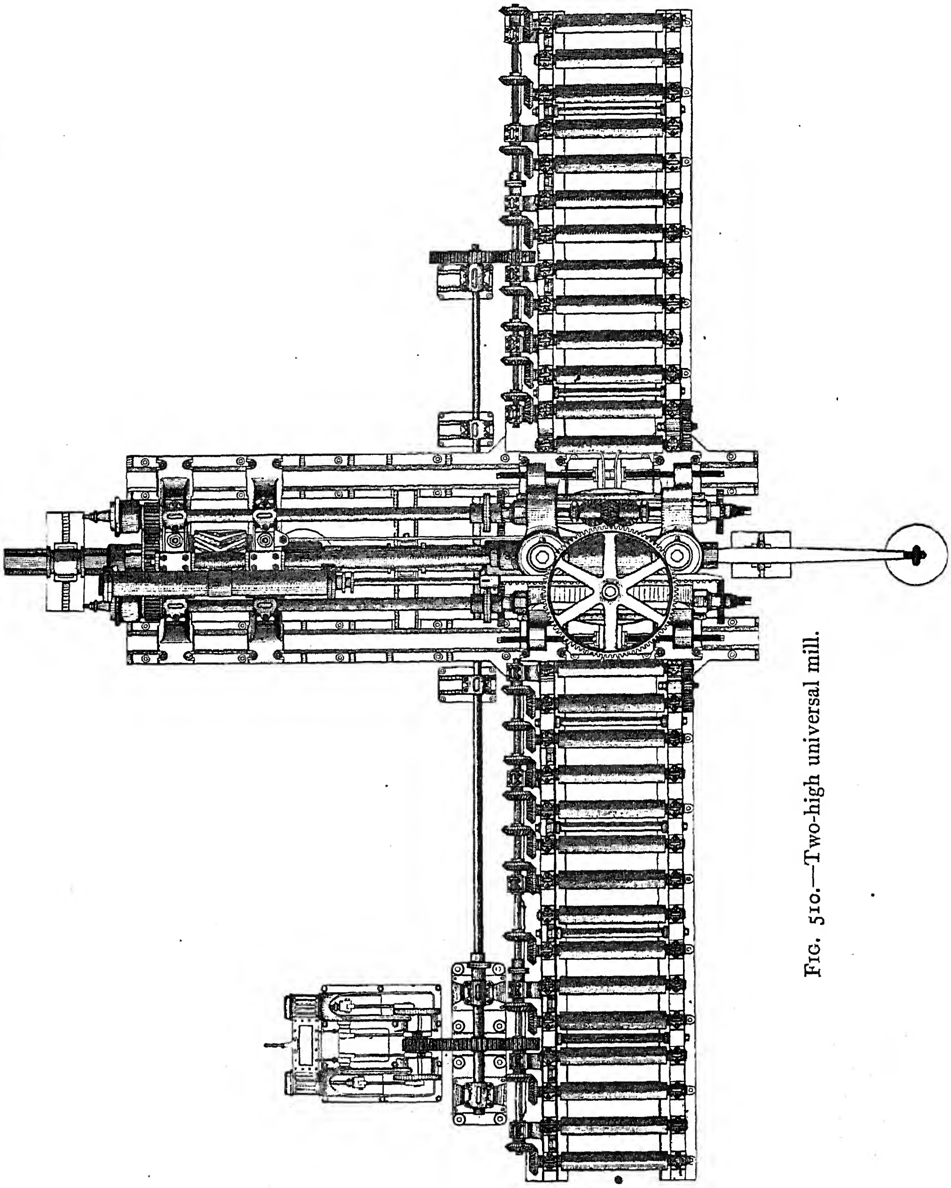


Fig. 510.—Two-high universal mill.

edges so as not to require any shearing. As is the case in the ordinary two-high plate rolling-mill, the top horizontal roll is balanced by stay-rods, levers and counter-weights, and adjusted by screws, while the bottom-roll is journaled in

fixed bearings. The vertical rolls are driven by bevel gearing set in motion by a shaft having at one end a spur-gear meshing with another keyed to the shaft. The speed of the vertical rolls is guided by that of the horizontal rolls. The vertical rolls must travel faster than the horizontal, because the piece increases in length and therefore leaves the rolls at a higher speed than it entered them. This difference in speed, which depends upon the reduction a piece receives in rolling, is taken into account in the design. The actual speed given is usually somewhat greater than the theoretical. In order to make the reduction by a pass independent of the speed of the vertical rolls, the latter are often driven independently of the horizontal rolls. The adjustment of the vertical rolls to enlarge or narrow the space between them is not clearly shown in the illustrations. It appears to be as follows: The short horizontal driven shaft carrying the bevel gear at the right end is hollow and has grooves; the long solid driving-shaft passes through the driven shaft and has splines which fit into the grooves. The vertical rolls can separate from and approach each other and at the same time be driven from one main shaft. The bearings of the vertical rolls are carried in guides and adjusted by screws having gearings at the ends which mesh with central spur-wheels. One or the other of the two sets of gearings is set in motion by means of a lever, and thus the vertical rolls brought closer together or farther apart. The side-guides of a piece, Figs. 509 and 510, follow the lateral motion of the vertical rolls, being directed by fingers (eccentrics).

The horizontal rolls are 31 in. in diam., 60 in. long. The maximum opening is 60 in. The vertical rolls are 17.5 in. in diam. and the maximum opening 56 in. Both horizontal and vertical rolls are of chilled cast-iron. The maximum speed of the horizontal rolls, driving the finishing pass, is 120 r.p.m. The mill is driven by a double 55×60 in. reversing engine; the steam pressure at the engine is usually 125 lb., hence the I.H.P. is 5,000. The top-roll is raised and lowered either by a hydraulic cylinder having a plunger 12 in. in diam. and a stroke of about 11 ft., or an electric motor of 50 h.p. The steam engine for the mill-table rollers is 14×14 in. and the speed about 150 r.p.m. An ingot weighing 8,500 lb. is rolled in 17 passes to a plate 48 in. ×0.25 in. ×200 ft. The capacity of the mill is 15,000 tons per month; five men run the mill.

287. Power of Rolling-mills.—In the six examples of rolls given, hydraulic- or steam-power is used for all mechanical purposes. In the most recent mills electricity[1] is used as motive power excepting for the driving of the rolls, where steam has held its own; electrically driven rolls are, however, coming into use. The power required to drive rolls has been arrived at so far mainly by precedent. With the introduction of electricity as motive power, more exact knowledge as to power consumption has become necessary. Knesche[2] carried along

[1] Ortmann, *Stahl u. Eisen*, 1909, XXIX, 1820.
Wiley, *Iron Age*, 1909, LXXXIV, 12.

[2] Iron Age, 1908, LXXXII, 239; *Eng. Mag.*, 1910–11, XL, 41, 233.

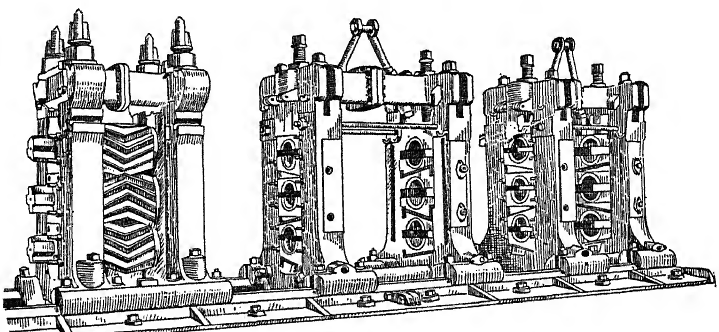
some investigations, and Ortmann[1] published an abstract of the first report of the German Commission[2] investigating the subject.

The formula $\frac{(S-S')L}{E}$ is the expression for the volume of material expelled from the original length per unit of energy employed. In it S is the original section, S' the reduced section, and L the original length, all in mm., and E the energy consumed in kg. m. Table 204 gives the values of cu. mm. of steel displaced by 1 kg. m. of energy at different temperatures.

Table 204.—Values of Steel (Cu. Mm.) Displaced by 1 Kg. M. Energy at Different Temperatures

Square ingots to	At temperatures, deg. C.			
	1,300	1,200	1,000	900
Flats	100	45	20	18
Rounds	80	50	....	....
T-girders	85	60	20	10
Rails	....	70	20	....

288. Parts of Rolling-mills. 1. Housings and Foundations.—There are two kinds of housings, Figs. 511 and 512, one for the pinions, the other for the rolls. Passing over the former, the housings for rolls were formerly made



Figs. 511 and 512.—Pinion- and roll-housings of a 23-in. rail mill.

exclusively of cast-iron, had closed tops, and were secured by wedges to flat bed-plates; Figs 490 and 491 are of this type. At present, cast-iron is often

[1] *Stahl u. Eisen.*, 1909, xxxix, 1.

[2] Puppe, *op. cit.*, p. 1204; further, Puppe, *Stahl u. Eisen*, 1910, xxx, 1619, 1824, 1871; 1912, xxxii, 6; *Iron and Steel I.*, Carnegie Scholarship Memoir, 1910, ii, 27.

Herrmann, *Stahl u. Eisen*, 1911, xxxi, 1706.

Katona, *Tr. A. I. M. E.*, 1904, xxxiv, 542.

Roberts, Iron Age, 1912, lxxxix, 17.

Wheatley, *Eng. mag.*, 1912, xliii, 33.

replaced by cast steel, at least in the larger mills; further, the housing is usually made with an open top and tied by a steel cap that is secured by bolts (Fig. 512). This arrangement facilitates changing of rolls, as these can be removed quickly as a whole and replaced by a spare set. The modern housing has grooved feet fitting on or into projections of the bed-plates in which are cut grooves to receive I-shaped bolts (Figs. 496, 500, 503, 511 and 512). This method makes keeping in line and shifting of housings a simple matter. A rectangular housing has a minimum cross-sectional area of 0.25–0.35 D^2 (diam. of roll), the ratio of width to depth ranges from 1 : 1.3 to 1 : 1.5; the face is made 1 1/8 D, the cap at its

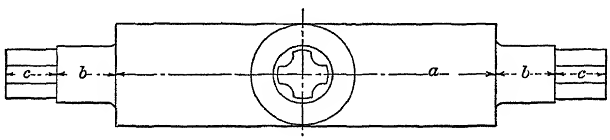


FIG. 513.—Plain roll.

weakest point has a cross-sectional area of 0.4–0.5 D^2; the top-screw is 1/4–1/3 D.

The foundation for the housing is of stone, brick, or cement concrete. It consists of a solid block on which are erected with set-offs two parallel walls to carry the two bed-plates; the walls are from 3 ft. thick by 6 ft. deep with rod-mills to 6 by 10 ft. with heavy plate-mills; in the passage, from 2 to 4 ft. wide, between the walls there are left open recesses for the lower ends of the anchor-bolts.

ROLLS.—In a roll, Fig. 513, there may be distinguished three parts, the body or barrel *a* with which the rolling is done, the neck *b* on which the roll revolves in the bearings, and the star-shaped coupling (pod, wobbler) *c* through which

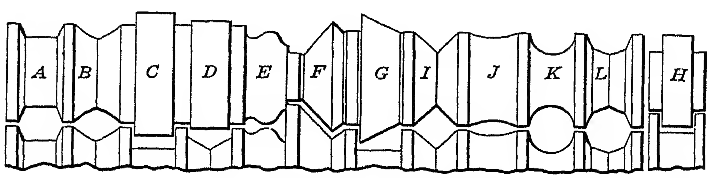


FIG. 514.—Forms of passes.

the roll is driven. As stated in §285, the body of a roll is plane when only plates or sheets are to be rolled. If a piece is to be rolled into a definite shape, at least one roll of the pair must be grooved.

The combination of the groove of one roll with groove, projection, or plane surface of another roll forms a pass. Passes have various forms, as shown in Fig. 514. Passes *A* and *B* are open, as the grooves are evenly distributed between the top and bottom roll; *A* is a box-pass, *B* a Gothic pass. Passes *C* and *D* are closed passes, as the grooves are cut deep enough in the lower roll to receive the projection (or former) of the upper one which closes the pass. By thus sinking the whole pass into the lower roll the height of the pass can be

changed by raising or lowering the upper roll without opening the pass. The width of the projection is made at least as great as its height. Passes *E* and *F* are half-open passes; they are closed on one side and open on the other; pass *F* appearing to be open is, however, half-closed, as the boundary-lines are too unevenly distributed between top and bottom to class it as open. As an open-pass roll is less deeply cut than a closed one, it is stronger, but it cannot shape the piece so accurately, as there is greater tendency to "finning," *i.e.*, the metal squeezing out between the crevices of the upper and lower rolls. This is because the metal flows outward sidewise more readily from the middle of the side than from the corner. The remedy for the evil is to roll the fin back into the piece before it grows too large by an edging pass, *i.e.*, turning the piece 90° so as to bring the fin against the solid body of the roll. Finning is prevented by making each successive pass enough wider than the preceding one so that the fin cannot reach the crevice between the rolls. In order to avoid this widening of passes, the piece is turned 90° when the former width becomes now the depth (thickness), and the flattening out is begun again. With Gothic or diamond passes, since the piece must be turned 90° at each pass, the successive passes must be so proportioned that the depth (height) of one pass shall be a little smaller than the width of the next. Pass *G* is an unbalanced pass. It deflects the piece horizontally and increases the friction of the rolls in their bearings. The piece on leaving the rolls tends to turn toward the least reduced side, and if it is straightened by a guard it is likely to tear. According to the shape of the passes, there are distinguished Gothic passes (*A*), flat passes (*C* and *H*), diamond passes (*I*), oval passes (*J*), round passes (*K*), polygonal passes (*L*), and special shapes (rails, angle-irons, etc.). The angles in grooved rolls are not as sharp as shown in the sketches, being rounded off somewhat; this is especially the case with the passes at the beginning of an operation, the rounding-off growing less as the rolling nears the finishing passes.

According to the kind of work that the rolls are intended to do, there are roughing and finishing passes, welding, drawing, shaping, flatting, edging, adjusting passes, etc., and the rolls are called roughing rolls, finishing rolls, etc.

In balanced passes the piece ought to emerge straight from the rolls; it may, however, be deflected vertically and horizontally. Vertical deflection is caused (1) by the friction of the collar against the piece, in which case the piece tends to adhere to the deeper-grooved roll; (2) by one roll having a greater circumferential speed than the other, in which case the piece follows the slower roll; (3) by having made the diameter of the groove of the top roll greater than the diameter of the groove of the lower roll, in which case the top surface of the piece is drawn out more than the bottom surface, and the bottom surface curls downward; (4) by the piece being unequally heated, in which case it tends to bend toward the hotter side. Causes 1 and 2 are corrected by guards. It is desirable to have these act in one direction, and if possible downward. In a two-high mill the deeper groove should be placed in the lower roll, and the higher circumferential speed given to the upper roll. In a three-high mill, the deeper grooves should be placed in the upper and lower rolls, and the middle roll given

the higher speed. An ordinary guard will then be required on the lower roll, and a suspended guard on the upper. Horizontal deflections are caused by unbalanced rolls and by unequal heating. Corrections are made by guides.

Rolls are made principally of cast-iron, occasionally of steel. The cast-iron may be soft, medium-hard or hard, depending upon the work that is to be done. Soft rolls should be made of tough gray iron cast in loam molds; medium-hard rolls of mixtures of gray and white iron and cast in chill molds coated with loam in order that the surface may become only partly chilled; hard rolls of mottled iron cast in bare or slightly coated chill molds. All rolls that have to do roughing work (blooming-rolls) should be soft; the grooves are often purposely roughened to assist in gripping the metal. Rolls with shallow grooves for medium-sized work are made of medium-hard iron provided that turning down to the

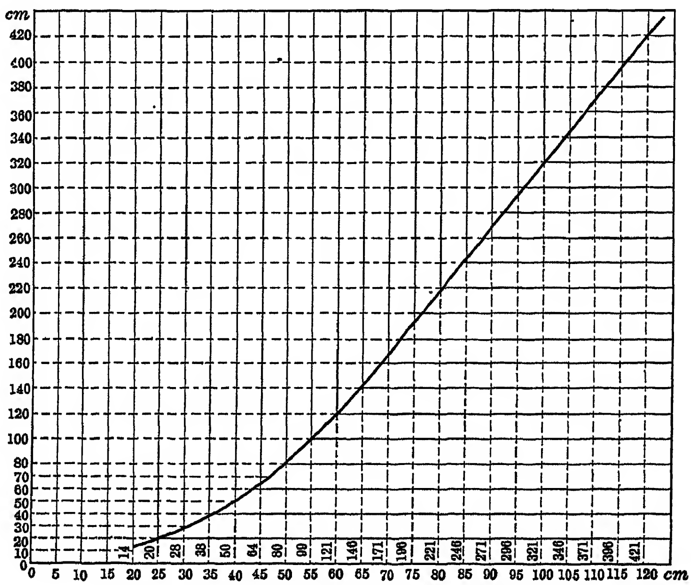


Fig. 515.—Curve for plotting length of roll.

approximate form of the casting does not penetrate the chilled surface. Finishing rolls are made of hard iron. Steel rolls were formerly confined to cold-rolling of soft metal; at present steel is used with hot metal, when the roll must be strong and not too thick, or when it has sharp corners that would crumble if the roll were of cast-iron. The steel is medium-carbon, 0.50 to 0.75 per cent.; usually it is cast, sometimes it is forged; nickel-steel rolls are used in some instances. The steel of rolls may, of course, not be chilled, as the hot metal passing through them would draw the temper.

The diameter of rolls varies with the work they have to do, from say 7 in. with wire-mills to 48 in. with blooming-mills. As the power required for rolling is proportional to the square root of the radius of a roll and to the thickness of a bar, the diameter of a roll is made as small as is consistent with safety.

The length of a roll will be governed by the bending stress set up in it. Neglecting the weight, the length may be considered as depending upon the resisting moment, and this varies with the cube of the diameter. Brovort[1] gives the curve shown in Fig. 515 to find the length of roll for a given diameter. It is based upon examples taken from practice; the original scale given in cm. has not been changed into in., as the curve representing empirical data would have to be re-drawn, and for this the data are not given. Taking a roll 80 cm. (=31.5 in.) in diam., first find the place on the abscissa; the corresponding ordinate cuts the curve at 221 cm.; now the section modulus $\frac{D^3\pi}{32}=\frac{80^3\times 3.14}{32}=50{,}176$; dividing this by 221 gives the desired length, 227 (=89 in.). In general, the length of a roll is rarely less than two or more than four times the smallest diameter.

The diameter of the neck =0.58 D (diam. of roll) with small, =0.55 D with medium, =0.53 D with large rolls; the length is equal to the diameter. The diameter of the wobbler is 3/8 to 5/8 in. smaller than that of the neck, the length 1 1/2 in. greater than the diameter of the neck.

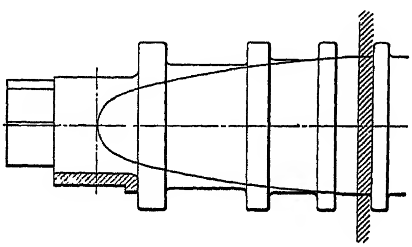


FIG. 516.—Disposition of grooves in roll.

In a grooved roll the center section will be the weakest. In order to make the roll strong, deep grooves will be placed near the ends and shallow ones near the center, thus approaching the theoretical circular beam of uniform strength which is the paraboloid, shown in Fig. 516.

The speed given to a roll depends mainly upon the size and temperature of the piece that is to be rolled. A large piece exerts a greater shock upon entering the rolls than a small one, as the force of the blow caused by it is proportional to the square of the surface-speed of the rolls; hence, considering the power required and the wear and tear upon the machinery, large rolls are run more slowly than small ones. Also, the slower the speed, the less power will be required to induce a flow of metal and the more effective will be the deformation. As regards temperature, a large piece is never so hot as a small one, and the finishing temperature must be high enough for the metal to be soft. Thus slabbing-mills make 20 to 30 r.p.m., three-high blooming-mills 50, rail-mills in the finishing passes 100, and rod-mills from 60 to 1,200 r.p.m.

PINIONS.—The teeth of pinions have to be strong to enable them to stand strains and especially sudden shocks. Formerly they were made straight, Figs. 490 and 491, running across the full width of the face as in cog-wheels. In order to give them the requisite strength their number had to be small, but this is not compatible with smooth running. At present, therefore, herring-bone or double-helical teeth,[2] Fig. 511, are used with which there are no shocks, as the rolling motion is continuous. Pinions are made almost exclusively of cast steel;

[1] *Op. cit.*, p. 2.

[2] Day, *Iron Age*, 1911, LXXXVIII, 1364.

smaller ones are cast in one piece. The larger ones have a core on to which is keyed the toothed shell. The diameter of the pitch circle is made smaller than the diameter of the roll in view of the necessity of this being turned down when worn. The length of the neck is 1 1/3 diam. of the pitch-circle; the pitch with the smallest rolls is $1 \times \pi$, with the largest $2 \times \pi$, and with plate-rolls even $2.5 \times \pi$.

Coupling-box.—This is a cast-iron or soft-steel hollow cylinder which usually has four prongs, as shown in Fig. 517. On one side it receives the coupling spindle, on the other the wobbler of the roll. A coupling spindle is made 3 1/2 to 4 times as long as its diameter, 1 1/2 to 2 1/2 in. play being left between the ends to be connected; the coupling-box is 1/2 in. longer than twice the length of the wobbler, its thinnest part is 1/4 to 1/3 the diameter of the wobbler; there is 1/2 in. play between the wobbler and the box.

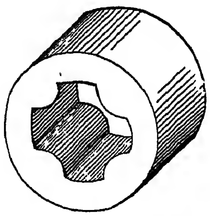


Fig. 517.—Coupling box.

Handling of Metal.[1]—The fore-plate, rest-plate, guide and guard of a mill have been referred to in connection with Figs. 490 to 494; additional reasons for requiring guards have been given on p. 667.

Tongs are used if a piece is light enough for one man to lift; if heavier than 50 lb., hand-levers (*m*, Fig. 490) come into play, as with these a man can handle weights of 300 to 400 lb. Should the weight be greater than 600 to 900 lb., two men each with a lever can work together. Instead of this, the suspension rod or chain of the lever can be raised and lowered by power and thus the difficulty overcome in a more satisfactory way as the lever has only to be directed by hand. With still heavier pieces or in large mills, lifting tables[2] with live rollers, Figs. 502 to 507, are generally used for handling. Sometimes tables, instead of being lifted front and back at the same time, are supported each near the middle by a shaft journaled at the ends, when front and back are alternately raised and lowered. In reversing rolls, where there is no lifting, stationary tables, Figs. 495 to 501, are common; dead rollers are used when the piece is to be moved by hand. One form of manipulator for handling and lifting ingots on a table has been represented in Figs. 502 to 504; a skid for transferring a plate from the roughing to the finishing-rolls is shown in Figs. 495 to 497; transfer tables accomplish this mechanically.

Rolling-mill Engines.—The strain put on a rolling-mill engine is severe on account of the irregularity of the load which becomes suddenly a maximum when the piece enters the roll and a minimum after a pass. With non-reversing engines this inequality is equalized by a heavy flywheel. The Porter-Allen engine with its high piston-speed, large weight of reciprocating parts, suitable cut-off of boiler-steam and shut-off of exhaust-steam, and sensitive governor direct-connected, has become the leading engine for non-reversing rolling-mills. With reversing mills compound engines with piston-valves are used, as they never come to rest at a dead-point.

[1] Stauber, *Stahl u. Eisen*, 1907, XXVII, 965; 1908, XXIX, 1009, 1089.

[2] *Stahl u. Eisen*, 1909, XXIX, 649.

289. Reduction of Metal by Rolls.[1]—In discussing reduction, it is convenient to consider separately the gripping and drawing-in, the lengthening and flattening, and the reduction of sectional area.

GRIPPING AND DRAWING-IN.—If, in Fig. 518, a bar be placed against a pair of moving rolls, a radial pressure P will be exerted upon it. This resolves itself into two components, the vertical $a=P\times\cos\phi$ which causes the reduction of area and the friction between roll and bar, and the horizontal $b=P\times\sin\phi$ which tends to prevent the bar from entering the rolls. As long as the friction resulting from a is greater than b, the bar will pass through the rolls, and will be rolled if it is plastic, or broken if brittle. Now a and b are functions of ϕ; with $\phi=0$ we have $b=0$, *i.e.*, the bar will readily pass through the rolls as long as its thickness is not greater than the distance between the rolls. The angle ϕ, *i.e.*, the pressure required, increases with a thicker bar, or with rolls of a smaller diameter, as in either case b becomes larger. But the arc AB increases with the diameter of the roll, and as the friction between bar and roll is proportional to the surface, it follows that with a greater diameter of the roll there is an increase in the power to draw in the bar. This shows that rolls of large diameter nip more readily than those of small diameter; also that thin bars are more readily nipped than thick bars.. Practical experience has shown that under ordinary conditions a bar will not be nipped by a smooth roll, if its thickness exceeds one-tenth the diameter of the roll on entering and one-twentieth after passing the roll, or if the angle ϕ is smaller than 30 deg.

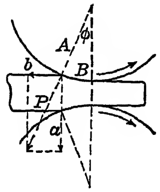


FIG. 518. Work of rolls.

. LENGTHENING AND FLATTENING.[2]—If a cube of a brittle substance be subjected in the direction of one axis to a gradually increasing pressure until it breaks, it will theoretically be found to have split into six pyramids, all of which have their apices in the center of the cube. If the brittle substance has the form of a cylinder (say a stearin candle), there will be formed two cones (Fig. 519), joined at the apices and with the bases coinciding with the pressure planes, while the material outside of the cones will have shelled off. If the brittle substance is a rectangular block instead of a cylinder, there will be formed two truncated pyramids joined at the smaller bases. If the body subjected to pressure is a plastic metal, the pressure cones or pyramids will also form as with brittle .material; but instead of shelling off, the metal will flow from the apices to the bases of the cones (cups) or pyramids. The formation of these can be seen in pulling apart an iron rod in a testing machine, when one surface of fracture will have a convex, the other a concave form only as shown in Fig. 520; the part that flowed from one cone has adhered to the other.

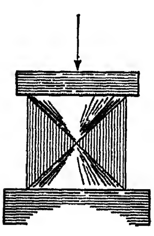


FIG. 519. Crushing of brittle cylinder.

If a metal is subjected to pressure between rolls, two small cones form in the line in which the pressure is being exerted, while the rest of the metal flows

[1] Falk, *Stahl u. Eisen*, 1910, XXX, 1986; 1912, XXXII, 816, 832.

[2] Sheld, *Stahl u. Eisen*, 1910, XXX, 415; criticism, *op. cit.*, 1910, XXX, 766, 1419.

out in the other two directions so that it becomes lengthened and flattened. The lengthening of the metal in rolling is represented in Fig. 521. Let H and h be the thickness of the bar before entering and after leaving the rolls. In passing from H to h, the bar is compressed by the arcs AB and CD of the rolls. The radial pressure of the rolls causes the formation of pressure cones ABE and CDE. If the rolls move through the arc $FB=GD$, the bar of the section $BFEGD$ will be drawn through the rolls and leave them at BD. As a rectangle of the length BF and the height h is smaller than the surface enclosed by $BFEGD$, its length must become larger than BF, or the length of a bar, after being rolled, is greater than the arc of the roll that did the work. It can be proved that the difference in velocity of a bar entering and leaving rolls is the greater the smaller the diameter of the rolls, or, rolls of small diameter increase the length of a bar more quickly than those of large diameter. Therefore in rolling small rods (which cool very quickly) the diameter of the rolls will be made as small as can be without danger of breaking.

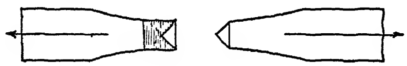


FIG. 520.—Square test piece.

The flattening of the metal is shown in Fig. 522. Suppose a flat bar of width B and thickness H is rolled down to thickness H_1. With a pressure-cone angle α, the metal outside of the cone occupying the area will have flowed out and will occupy the area $defgh$. The rectangle $degh$, representing the increase in width, is equal $\triangle\, abc - \triangle\, efg$; hence the increase in width $de = \frac{defg}{H_1} = \frac{abce-fg}{H_1}$. This proves that the flattening of a bar is dependent only upon its thickness before and after a pass, and is independent of its width. If the amount of flattening of a bar has been found experimentally, the pressure-cone angle α can be calculated; or, given the angle, the width can be determined. Supposing $H=1$ in., $H_1=0.75$ in., and $\alpha=45°$, then $de=\frac{1\times 0.25-0.75\times 0.1875}{0.75}=0.15$ in. on either side. Practical experiment shows that a bar flattens the more the lower its temperature; hence the pressure-angle must decrease with the temperature.

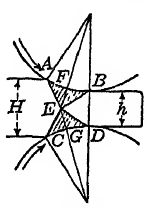


FIG. 521.—Lengthening of metal by rolls.

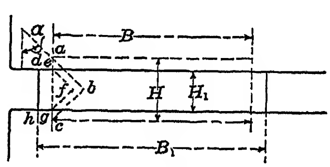


FIG. 522.—Flattening of metal by rolls.

In rolling a plate between smooth rolls or a rod in an open pass, the edges become uneven and ragged and have to be trimmed later on. If in a closed pass there is not sufficient room to allow for the flattening out of the piece, considerable lateral pressure will be exerted by it on the collar (flange) unless special precautions are taken to round off the form of the groove.

REDUCTION OF SECTIONAL AREA.—The reduction of the sectional area of a metal in rolling may not exceed a certain amount without tearing of the metal. The greater the reduction the greater the saving in labor (because fewer passes are needed), and to some extent in power (because the metal is hotter and hence softer). The permissible reduction depends on the temperature and resistance of the piece, the power of the rolls, and the ratio of size of rolls and of piece. It thus varies greatly, and is a matter of practical experience.

Wrought iron or low-carbon steel, being usually hotter and always softer than high-carbon steel, can stand a greater reduction; it will also be more drawn out and less flattened. Rolls of large diameter can exert a greater pressure and nip better than those of small diameter; but the larger the diameter and the slower the speed the more will they flatten the piece and the less will they lengthen it. Other things (temperature, hardness, section of piece) being equal, the resistance to rolling will be smallest if the reduction is distributed evenly over a number of passes. If H is the section of the original piece; $H_1 = xH$, the section after the first pass; $H_2 = x^2H$, after the second pass; $H_n = x^nH = h$, after the nth pass; then $x = \sqrt[n]{\frac{h}{H}}$ is the reduction factor. A simple example by Ångström[1] shows its application. Supposing that a plate 120 mm. in thickness is to be reduced to 6 mm. in 18 passes with a constant reduction factor, then $x = \sqrt[18]{\frac{6}{120}} = 0.84668$. In Fig. 523 the upper curve represents graphically this reduction. Thus $120 \times 0.84668 = 101.6$ mm. is the size to which it is reduced by the first pass, $101.6 \times 0.84668 = 86.0$ mm. that by the second and so on. On account of the increasing hardness of the piece in rolling, the lower curve (by Gresser[2]) with an increasing reduction factor is more satisfactory than the upper one with a uniform factor.

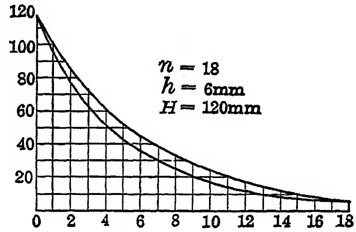


FIG. 523.—Reduction of metal in rolling.

With grooved rolls[3] the matter becomes more complicated, as the grooves turned in a roll will weaken it more at the center than at the ends. Therefore the larger grooves should be put in the ends and the smaller ones near the center. If the curve of reduction has been drawn, it will have to be distributed over the number of pairs of rolls that are to be used, and in each pair the reduction in the middle will have to be smaller than at the ends. This will give the reduction curve the form of a serpentine line.

The reduction in area by a pass ranges from 3 to 50 per cent.; 30 per cent. is a large figure; 20 per cent. is common with steel rails at the start, falls off to 15 and 12 per cent., and is very small toward the finishing passes.

[1] *Oest. Jahrb.*, 1891, xxxix, 373.
[2] Blass, *Stahl u. Eisen*, 1882, II, 189.
[3] Tafel, *Stahl u. Eisen*, 1909, xxix, 649.
Tuppe, *ibid.*, 1679.

290. Forging with Hammers in General.[1]—In this operation a metal is shaped by the mechanical effect of a blow or, as stated in §284, by applying pressure quickly and releasing it before the metal has yielded to it. In rolling, the pressure is applied slowly and released only after the metal has yielded to it. Hence, shaping a metal by means of the intermittent action of a hammer will progress more slowly than by the continuous action of rolls, especially as the forging effect may extend only a small distance beyond the upper and lower surfaces of a piece, because otherwise it would tear it. On the other hand, a piece can be made to assume a variety of forms in cross- and longitudinal sections not practicable in rolling. Rolls, therefore, will be used to turn out cheaply and quickly a large production of uniform section; hammers to forge at a higher cost a smaller quantity of metal which may vary greatly in section. However, hammering can furnish a better product, as the finishing temperature and the force of the blow to be given are under control of the operator. Each method has its own field.

The mechanical effect of the blow of a hammer at the moment of impact is $\frac{Mv^2}{2}$, in which M = mass (*i.e.*, W, weight, divided by g, acceleration of graivty), and v the velocity at the time of impact ($v = 2gh$, h = height of fall). According to the formula a heavy hammer falling through a short distance should have the same effect upon a piece as a light hammer falling through a long distance as long as the $\frac{Mv^2}{2}$ of the two is the same. The fact is that the forging effect of the light hammer extends only a short distance beyond the surface in comparison with that of the heavy hammer.

In the following example two hammers I and II of the same cross-section, having different weights and falling different heights, are supposed to act upon two pieces of metal of the same composition and form. It is assumed that the resistance to deformation by both hammers is the same, say 200,000 lb.

TABLE 205.—EFFECT OF LIGHT AND HEAVY HAMMERS

Hammer	I	II
W, Weight of hammer, pounds.............. ..	10,000	5,000
Falling distance, feet........................	1	2
Work done, foot-pounds.....................	10,000	10,000
F, Resistance of piece to deformation, assumed to be constant, pounds.	200,000	200,000
v, Velocity at impact $= \sqrt{2gh}$, feet	$\sqrt{2\times 32.2\times 1}=8.03$	$\sqrt{2\times 32.2\times 2}=11.35$
A, Retardation of velocity of hammer until it comes to a rest $=\frac{F}{M\ (mass)}$ feet per sec.	$\frac{2000000\times 32.2}{10000}=644$	$\frac{2000000\times 32.2}{5000}=1288$
The same expressed in time, $t = \frac{v}{a}$, sec....	$\frac{8.03}{644}=0.01247$	$\frac{11.35}{1288}=0.00881$

[1] Fischer, Hammers and Presses at the Exposition of Düsseldorf, *Zt. Ver. deutsch. Ing*, 1902, XLVI, 1384.

The table shows under the above assumption, that the resistance is the same in both cases; the lighter hammer (II) falling a longer distance comes to rest more quickly than does the heavier hammer (I) falling a shorter distance, As the resistance of the metal to flowing depends to a certain extent upon time; the time required by the hammer to come to a rest will govern its forging effects the heavier hammer will cause a greater deformation than the lighter. Therefore, with large forgings heavy hammers will be used and light ones with small forgings. With the latter the energy of the blow may be increased by admitting steam above the piston on the downward stroke.

Of the total energy exerted by the head of a hammer, part goes into the piece to be forged, part into the anvil; the larger the weight of the hammer-head in proportion to the weight of the anvil, the greater will be the forging effect upon the piece.

291. **Forge-hammers.**—These may be broadly classed as lever-hammers and direct-acting or stamp-hammers.

LEVER-HAMMERS.—In lever-hammers, the head, attached to one end of a horizontal shaft, is raised vertically through a small arc by a rotary cam-shaft, and then allowed to drop on to the piece on the anvil. According to the manner of raising, these hammers may be divided into tilt- and helve-hammers. In a tilt (or trip-) hammer the hammer-stock is pivoted as a lever of the first order; the head is fixed to the end of the larger arm, and the cam acts upon the shorter one. In a helve-hammer,[1] the hammer-stock is pivoted as a lever of the second order, that is, the head and the point of attack for the cam are on the same side of the fulcrum. With a nose- (frontal-, tenant-) helve, the head is placed between the fulcrum, at one extremity, and the lifting-cam which acts at the other; with the belly-helve, the positions of head and cam are reversed.

Lever-hammers have become about obsolete on account of the fact that the faces of anvil- and hammer-head can be parallel only in a single position, and that the height of the lift and with it the size of piece are confined to narrow limits. Tilt-hammers were used mainly for drawing down iron and steel into rods; they are still employed for necking light forgings. Tenant-helves were commonly used for shingling or making blooms from puddle-balls; they have been largely replaced by squeezers. Belly-helves served for general forge-work, but they have become obsolete.

DIRECT-ACTING OR STAMP-HAMMERS.—These have different forms, such as drop-hammers, board drop-hammers, power-spring hammers, pneumatic hammers, etc. They are passed over, as they are used mainly in small and light die-forgings.[2]

The leading form of metallurgical hammer is the steam-hammer. This is essentially a power-hammer moving vertically between guides and delivering a blow on a piece placed up on an anvil. It consists of four parts, a supporting

[1] Egleston, *Tr. A. I. M. E.*, 1880, VIII, 515, Plates VI, VII, VIII.

[2] Woodworth, J. V., "Drop Forging, Die Sinking and Machine Forming of Steel," Henley Publ. Co., New York, 1911.

Plant of A. O. Smith Co., Milwaukee, Wisc., *Iron Tr. Rev.*, 1912, LI, 915.

frame-work, a vertical steam-cylinder with valve-system carried by the frame, a hammer-block (or tup) which forms the lower part of the piston-rod passing downward through the cylinder-head, and the anvil mounted on a suitable foundation.

The form of the frame, or stand, and the material of which it is made depend upon the weight of the hammer. With small hammers weighing say less than 0.5 ton, a single-frame stand of cast-iron, which frequently carries the anvil (Fig. 524) also is common. When the hammer weighs from 1 to 30 tons, a cast-iron double stand is used; part of it carries the guides (Naysmith) between which the hammer-block ascends and descends (Fig. 525); with heavier hammers the doublestand is often built of structural steel.

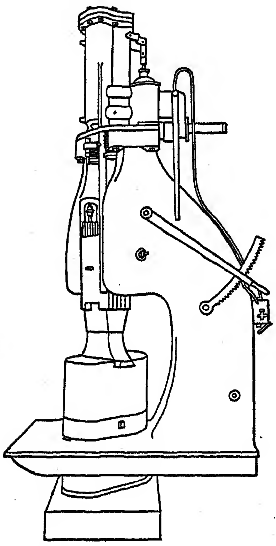


FIG. 524.—Single-frame Naysmith steam hammer.

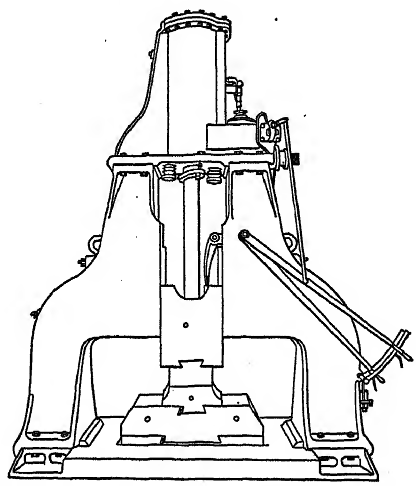


FIG. 525.—Double-frame Naysmith forging hammer.

The anvil, Fig. 525, consists of two parts, the anvil-block, a heavy iron casting of the form of a truncated cone or pyramid, and the anvil-die, of steel or chilled iron, fitted into the block by a dove-tail joint and wedges. The weight of the anvil-block is large in comparison with that of the hammer-block, from 10 times with general forge-work to 30 times with heavy work. The foundation must be correspondingly heavy.[1] It is customary to place between the concrete foundation and the anvil-block, or the bed-plate carrying the block, heavy timbers laid crosswise, as these distribute the force of a blow evenly upon the concrete and furnish, at the same time, an elastic cushion which weakens the jar.

The hammer-block, Fig. 525, consists of two parts, the cast-iron tup, which gives the required weight, and the steel or wrought-iron hammer-head; the two are fitted with dove-tail groove and wedges. The shape of the forged piece

[1] Anon., *Iron Trade Rev.*, 1909, XLIV, 377.

(square, round or octagonal) will depend, in part upon the mode of operating, in part upon the form given to the surface of the anvil-die and the hammer-head.

The inverted cylinder differs from the ordinary form in the valve-system, in that the motion of the valves is controlled by hand, Figs. 524, 525 and 526. According to the manner of admitting steam, hammers are classed as single- or direct-, and double-acting. In a single-acting hammer steam is admitted only on the lower side of the piston to raise the hammer-head; it is then automatically shut off and the exhaust-port opened when the hammer falls by gravity. In the double-acting hammer steam is admitted also on the upper side of the piston, whereby the rapidity of the fall and force of the blow are increased.

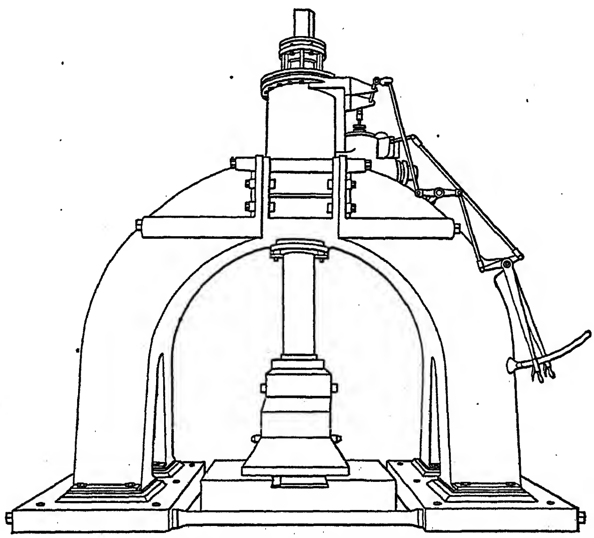


FIG. 526.—Double-frame Morrison forging hammer.

In order to deliver a gentle blow the exhaust-steam below the piston may be throttled so as to act as a cushion. Most modern two-stand hammers are of the Naysmith, or guided-ram, type, Figs. 524 and 525, *i.e.*, the falling weight is concentrated in the hammer-block which travels on parallel guides attached to the stands below the steam-cylinder, and the piston-rod is of relatively small diameter. In the other, Morrison or guided-rod, type, Fig. 526, the falling weight has the form of a large cylindrical bar of which the piston, situated near the middle of its length, is an integral part. The bar extends above the piston and passes through the upper cylinder-head, being guided by this as well as by the lower head. This type of hammer is used for work that is high or irregular in shape, as it furnishes much more room than does a hammer of the guided-ram type.

Nearly all steam-hammers are double-acting; the valve-motion is usually so fitted that the hammer can be worked as single- or double-acting, that the stroke can be adjusted to the thickness of the piece, and the exhaust controlled to give a light cushion or a heavy dead-stroke blow. The latter method

is applied, *e.g.*, in drop-forging[1] with die-faced shapes of the exact form of the finished piece. Fig. 524 represents a single-frame guided-ram hammer. The leading data are given in Table 206.

TABLE 206.—SINGLE-FRAME GUIDED-RAM HAMMERS

Size, pounds	Diam., inches	Stroke, inches	Face of ram, inches	Usual die face, inches	C. of die to frame inches	Steam, inches	Exhaust, inches	Weight, pounds
200	5	16	5 × 7½	4 × 7	12½	1½	2	4,400
250	5	16	5½× 7½	4½× 7	12½	1½	2	5,000
400	6	19	6 × 9	5 × 8½	15	1½	2	7,700
600	7	22	7 ×11	5 ×11	17½	2	2½	11,000
800	8	25	8 ×13	6 ×12½	20	2	2½	14,000
1,100	10	28	8½×13½	6 ×13	22½	2½	3	19,000
1,250	11	30	9 ×13½	7 ×13	23½	2½	3	23,000
1,500	12	33	9 ×16	7 ×14½	25	2½	3	28,000
2,000	13	36	10 ×17	8 ×16	27½	2½	3	35,000
2,500	14	39	10½×17½	8 ×17	30	3	4	42,000
3,000	15	42	10½×19	8½×18	33	3	4	49,000
3,300	15	42	9½×16	8 ×16	38	3	4	55,000
3,500	16	45	11 ×19	8½×18	33	3½	4½	56,000
4,000	17	42	12 ×20	9 ×20	33	3½	4½	65,000
4,500	18	42	13 ×21½	10 ×21	48	4	5	75,000
5,000	20	48	14 ×22½	11 ×22	48	4	5	85,000

A special form is the tilting-hammer used for drawing down tool-steel into flats, squares, octagons and rounds. It is made exceedingly heavy and delivers a large number of uniformly powerful blows in quick succession, *e.g.*, 400 per min., by having the valve-gear so arranged that there is practically no variation in the length of stroke.

Fig. 525 is a typical form of double-frame guided-ram forging-hammer. The important dimensions and weights are given in Table 207.

The illustration stands also for a drop-hammer designed for die-forging, swaging and cutting off. The narrow double frame is held together by springs under the nuts to prevent breakage due to continued concussions. It is adapted to give single or successive dead-blows, or to run automatically.

The 20-ton steam-hammer of the Latrobe Steel Works,[2] and the 35-ton hammer of the Howard Axle Works[3] belong to this type.

Fig. 526 represents a double-frame forging-hammer of the guided-rod, or Morrison, type.

[1] *Iron Age*, Anon., 1908, LXXXII, 308.
[2] *Iron Age*, 1890, XLV, 495.
[3] *Op. cit.*, June 30, 1901, p. 10.

TABLE 207.—DOUBLE-FRAME GUIDED-RAM HAMMERS

Size, pounds	Diam., inches	Stroke, inches	Greatest die surface of ram, inches	Usual die face, inches	Distance between frame, feet	Steam, inches	Exhaust, inches	Weight without anvil, pounds
600	7	22	7 ×11	5 ×11	3½	2	2½	7,000
800	8	25	8 ×13	6 ×12½	3¾	2	2½	8,000
1,000	10	28	8½×13½	6 ×13	4	2½	3	9,000
1,250	11	30	9 ×13½	7 ×13	4½	2½	3	10,000
1,500	12	33	9 ×16	7 ×14½	5	2½	3	13,000
2,000	13	36	10 ×17	8 ×16	5½	2½	3	15,000
2,500	14	39	10½×17½	8 ×17	6	3	4	19,000
3,000	15	42	10½×18½	8½×18	6½	3	4	24,000
3,500	16	45	11 ×20	8½×19	6½	3½	4½	28,000
4,000	17	48	12 ×21	9 ×20	7	3½	4½	30,000
4,500	18	48	13 ×22½	10 ×22	7½	4	5	33,000
5,000	20	54	14 ×24½	11 ×24	8	4	5	35,000
6,000	20	54	15 ×25½	12 ×25	8½	4	5	45,000
6,000	22	54–60	15 ×25½	12 ×25	8½	4	5	48,000
7,000	23	60–66	17 ×26½	13 ×26	9	5	7	52,000
8,000	24	60–66	17 ×28½	14 ×28	9½	5	7	57,000
10,000	25	66–72	18 ×31	15 ×30	10	5	7	70,000
12,000	26	72	18 ×32	15 ×31	10½	5	7	85,000
14,000	28	72	18 ×34	15 ×33	11½	6	8	105,000
16,000	30	84	19 ×36	16 ×35	12½	6	9	120,000
20,000	34	96	20 ×38	16 ×38	14½	7	12	140,000

292. Mode of Operating.[1]—The usual steps in the operation of shaping a piece with a hammer are the heating, holding and forging. The heating is carried on in reverberatory furnaces direct or gas-fired. The latter are frequently provided with Siemens regenerative chambers. Care is to be observed in firing to have the flame more reducing than oxidizing so as to diminish scaling. Muffle-furnaces are the exception.

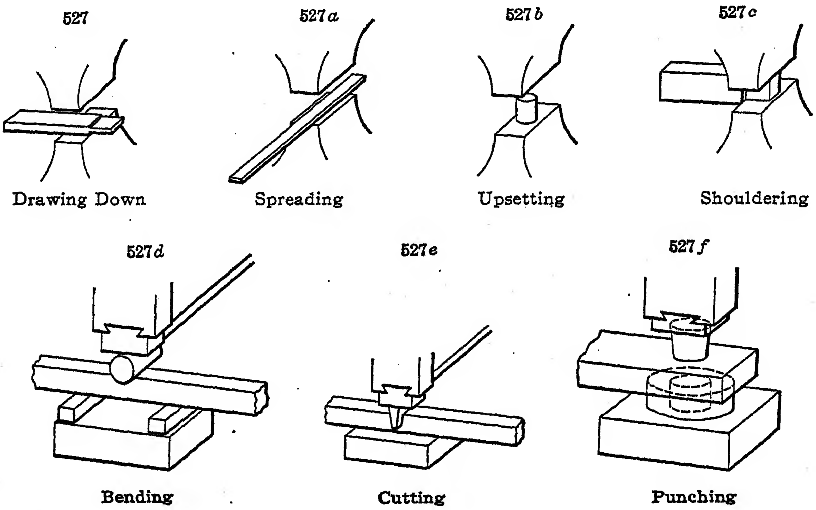
Pieces sufficiently small for one man to lift are manipulated with tongs fitted to them. For heavier pieces, lifting-tongs (dogs) suspended from an overhead runner carry the weight, while the piece is guided by hand and turned on the anvil by means of suspended hand-levers (*m* in Figs. 490 and 491). For very heavy pieces, the porter-bar[2] is convenient; a long rod provided at one end with jaws in which an ingot is secured by bolts, at the other with handles for manipulation, and at the middle with a sheave around which passes an endless chain for suspension of the load from a jib-crane or an overhead trolley. With the heaviest pieces, additional mechanical apparatus become necessary.

[1] Bacon, J. L., "Elementary Forging Practice," Wiley, New York, 1904.
Horner, "Die-forging," *Engineering*, 1901, Vols. LXXI and LXXII; 1902, Vol. LXXIII.
[2] Harbord-Hall, "Steel," Lippincott, Philadelphia, 1911, Vol. II, p. 844.

In forging a piece that is to be much drawn out, it is often necessary to begin work near the middle and complete it before attacking the ends, as in this case only the ends of the elongated piece will have to be re-heated, and this can be done without any difficulty. If the work is begun at one end, later on the piece may be too long to permit heating the center in a given furnace.

A piece receives several transitional forms, while it is being forged, before it can be given its finished shape. These forms are produced by certain manipulations,[1] a few of which may be given.

DRAWING-DOWN AND SPREADING.—If the hammer-head strikes a piece of metal a pressure-cone will form at the point of impact, and the metal will flow from the apices to the bases (see §289). If the metal is to be drawn out, the



527 Drawing Down; 527a Spreading; 527b Upsetting; 527c Shouldering

527d Bending; 527e Cutting; 527f Punching

oblong faces of the hammer-head and working-anvil ought to be narrow, and the piece to be held at right angles to the faces or in the direction in which it is to be drawn, Fig. 527. The unevenness of the surface is then smoothed by spreading as seen in Fig. 527a.

UPSETTING OR JUMPING.—Fig. 527*b* illustrates the means of reducing the length of a piece by endwise blows, and thickening or spreading laterally.

SHOULDERING (Fig. 527*c*).—The cross-section of a piece is reduced at one place by striking and turning over after every blow so as to form a shoulder; the metal of the neck flows back into the piece.

BENDING (Fig. 527*d*).—The piece is placed on two supports, and the fullering tool placed on the point where the bend is to start.

CUTTING (Fig. 527*e*).—A cutter (or hot-chisel), a tool triangular in cross-section and having a blunt edge, is forced through the piece.

PUNCHING (Fig. 527*f*).—A collar, higher than the thickness of the piece and larger in diameter than the punch, is placed under the piece and the punch. A

[1] Beckert, Th., "Leitfaden zur Eisenhüttenkunde," Springer, Berlin, 1909, Vol. III, p. 167.

short truncated cone longer than the thickness of the piece is forced through it with the big end down.

293. Forging with Hydraulic Presses.[1]—The presses used for shaping metal are of a great variety and are worked in different ways as, *e.g.*, by means of levers, screws, eccentrics, toggles, etc.; here only presses worked by water-pressure will be briefly considered.

Large ingots are now almost always forged with hydraulic presses instead of with hammers, as the latter have several disadvantages which make them ill-suited for the work. Some of these are: (1) A large part of the energy exerted by a blow is dissipated into the anvil-block and surrounding parts, while with a press, in which the hydraulic cylinder and base are rigidly connected, all the energy goes into the forging. (2) The effect of the sudden blow of the hammer extends only a short distance beyond the upper and lower surfaces of a piece; that of the enduring squeeze of the press goes deeper, hence the piece is worked more thoroughly and more quickly by the press; further, since the press can take a cooler ingot, it can work more accurately to dimensions and, with steel, nearer the critical range. (3) The plant is more expensive. While a press costs more than a hammer of equivalent power, the anvil-block of the latter with its immense foundations, and the larger steam-plant required, make the pressing-plant in the end the cheaper.

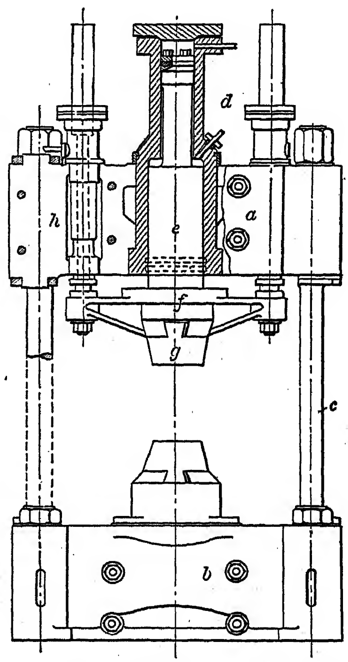


FIG. 528.—Baare hydraulic press.

The essential structural parts of a press, Fig. 528, are a heavy cross-head *a* and base-plate *b* joined by four circular machined steel columns *c*. The cross-head contains the hydraulic press-cylinder *d* with its ram or plunger *e* which carries by means of the tool-holder *f* the upper die *g* and the subsidiary cylinders *h* (hydraulic or steam) for lifting the ram when the water in the press has been released. The base-plate carries the lower die *i* and the four steel columns *c* which take up the tensile strain and serve as guides for the tool-holder as it rises and falls. Presses are classed as being low-pressure and high-pressure; the former exert a maximum pressure of about 1,000 lb. per sq. in. and serve to raise weights; the latter reach about 8,000 lb. per sq. in. and do the hot-

[1] Gautier, *Bull. Soc. Ind. Min.*, Ste. Étienne, 1889, III, 839.
Daelen, *Stahl u. Eisen*, 1892, XII, 57, 155; 1898, XVIII, 314.
Twedell, *Engl. Inst. Civ. Eng.*, 1894, CXVII, 1; *Stahl u. Eisen*, 1894, XIV, 900.
Fischer, *Zt. Ver. deutsch. Ing.*, 1902, XLVI, 1384.
Macka, *Oest. Zt. Berg. Hüttenw.*, 1911, LIX, 541, 553, 570, 627, 644, 658; 1912, LX, 689, 701, 719, 729.
Hofmann, F. J., "Die Hydraulischen Schmiedepressen," Springer, Berlin, 1912.

pressing and cold-pressing of metal. Forging presses have a range in size from 600 to 14,000 tons. With high-pressure work the pressure-cylinder must be of steel, as cast-iron allows water to percolate when a pressure of about 3,000 lb. per sq. in. has been reached; special attention must be given to the form of valve and to the kind of packing. There are three classes of presses:

1. Presses with Water-pressure from an Accumulator.—Water is pumped into an accumulator, loaded with a dead weight, and drawn off when required, thus causing the pressure-piston to descend regularly and continuously. The Baare forging press,[1] Fig. 528, may serve as an example. There

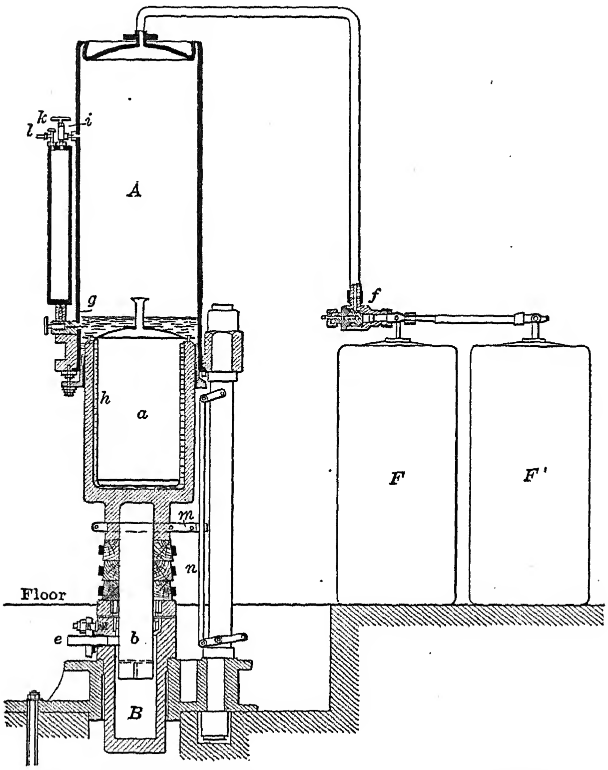


Fig. 529.—Prott and Seelhoff pneumatic accumulator.

are two pairs of pumping-engines serving two accumulators. One with water under a pressure of 8,000 lb. per sq. in. serves the press-cylinder. It has a Prött-Seelhoff pneumatic accumulator[2] to avoid the shocks that would be caused by the momentum of the falling weight with an hydraulic accumulator when the escape-valve is suddenly closed. It is represented in Fig. 529, in which *A* is the air-cylinder with hollow cast-steel plunger *a*, *B* the water-cylinder

[1] *Tr. A. I. M. E.*, 1892–93, XXI, 336.

[2] Prött -Seelhoff, *Stahl u. Eisen*, 1891, XI, 132.
Daelen, *Tr. A. I. M. E.*, 1892–93, XXI, 331.

with plunger *b* having 1/10 the area of *a*; hence, if the pressure in *A* be 50 atm., and water from the pump is forced through *e* into *B* so as to raise *a* and *b*, the pressure in *B* will be 500 atm. When water is forced into *B*, the air in *A* will be compressed; when water is taken out of *B*, the air will expand and drive the plunger downward, but without shock. *F* and *F'* are auxiliary air-tanks which may be cut off or connected by valves, as at *f*, to increase the air-space and thus diminish proportionally the range of pressure in *A*. Air is forced into *A* from a compressor connected at *l*. In order to prevent any leakage of air through the plunger *a*, a sheet-iron cylindrical vessel *h*, connected with the air-cylinder *A*, is placed in it, and the annular space filled with oil to above the oil-gauge *g*, through which the oil is introduced. The pump is engaged or disengaged by means of *m* and *n*. The second accumulator, with water under 700 lb. pressure, is loaded in the usual way; it serves to lift the cross-head. The ram in the press-cylinder is not of uniform diameter, the upper part having an area 1/3 that of the lower. By admitting pressure-water of nearly 4 tons per sq. in. to the larger cylinder, or to the smaller, or to both at the same time, three working pressures of 4,000, 2,700 and 1,300 tons respectively, in the proportions of 3 : 2 : 1 can be obtained. The valve-motion[1] is managed by a single lever. At the first movement the water from the press-cylinder is released and the press-head raised; at the second, all valves are closed and the head remains stationary; at the third, low-pressure water is admitted to bring the head down to the piece; at the fourth, high-pressure water is admitted to work the piece. As many as 30 strokes per min. can be had when light finishing-work is being done.

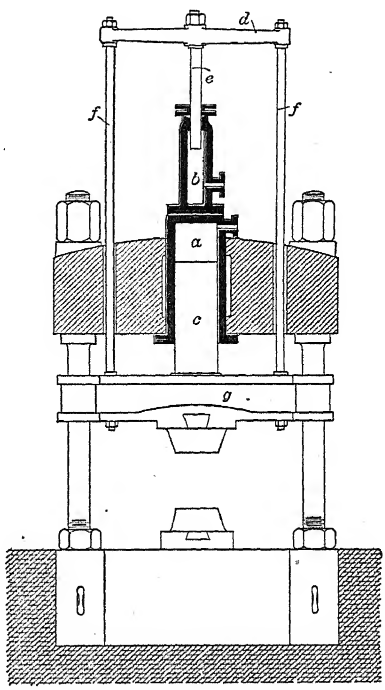


FIG. 530.—Kamp & Co. hydraulic press.

2. PRESSES WITH WATER-PRESSURE DIRECT FROM PUMP.—A simple form of this press is that of Kamp & Co. sketched in Fig. 530. The larger press-cylinder *a* and the smaller single lift-cylinder *b*, each with its water-connection, are placed one on top of the other. The power of the press-plunger *c* is increased or diminished by having the pressure-pump make a larger or smaller number of strokes; the cross-head *d* of the lift-plunger *e* is joined by connecting-rods *f* to the tool-holder *g*. In order to obtain a uniform delivery of pressure-water, the piston-rod of the steam-cylinder operates two or three water-plungers by means of a crank and a two- or three-throw crank-shaft. The Allen press[2]

[1] Daelen, *Tr. A. I. M. E.*, 1892–93, XXI, 239.
[2] *Tr. A. I. M. E.*, 1892–93, XXI, 327; *J. I. and St. I.*, 1891, II, 62.

works on a similar principle. The 14,000-ton Whitworth press[1] of Bethlehem Pa., for the liquid compression of steel belongs to this class of machines as does the Vickers[2] 8,000-ton press, and the Haniel-Lueg press made by the Mesta Machine Co., Pittsburgh, Pa.[3]

3. Presses with Hydraulic Steam Intensifier.—In this class of machines the pressure is obtained directly from a steam-piston and the prolongation of its rod which forms the plunger of a hydraulic cylinder. The press of Breuer, Schumacher and Co.,[4] shown in Fig. 531, is a machine with a single-acting steam intensifier. The prolongation of the piston-rod in the vertical steam-cylinder, *a* forms the plunger in the hydraulic intensifying cylinder *b*; the pressure-water is conducted through pipe *c* into the press-cylinder enclosed in the cross-head *d*; the areas of the cylinders are so proportioned that steam of 60 to 80 lb. per sq. in. in *a* will produce a pressure of 2 to 3 tons per sq. in. on the plunger in *b*; *e* is the ram, *f* the tool-holder, *g* the die, *h* the base-plate, *i* the steam lifting-cylinders connected by rods with the tool-holder *f*, *k* and *l* pipes connected with an overhead water-tank. Water is admitted to *b* and *d* through valves which open downward, but are closed again when *b* and *d* are under pressure; the admission and cut-off of steam are regulated at the back (not shown) through a lever raised and lowered by hand. In starting the operation, the lever is depressed and steam admitted to cylinder *a*. This raises the piston and with it the plunger which forces water from *b* into *d* and causes the working ram to move downward. The lever is now raised; this cuts off the steam supply and exhausts the steam in *a*; the result is that the piston *a* and with it the plunger *b* fall by gravity, and release the pressure in pipe *c*. In order to raise the ram *e*, steam is admitted into the lift-cylinders *i*, when the rising ram *d* forces the water above it back into cylinder *b*. The ram sinks again, when steam under cylinder *i* is exhausted; the vacuum created above it is filled by water flowing in through supply-pipe *k*; steam

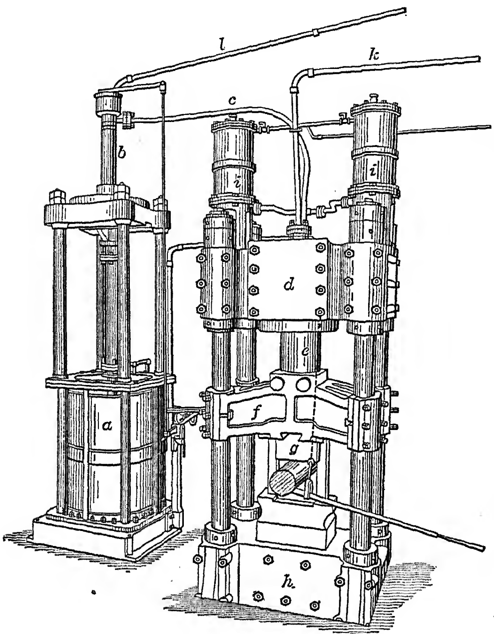


Fig. 531.—Hydraulic press of Breuer, Schumacher & Co.

[1] Howe, "Steel," p. 155; *Tr. A. I. M. E.*, 1907, xxviii, 94.
[2] *Engineering*, 1897, lxiv, 557.
[3] *Iron Age*, 1911, lxxxvii, 48, 1471; *Iron Trade Rev.*, 1911, xlviii, 147.
[4] *Stahl u. Eisen*, 1892, xii, 155; 1897, xvii, 257; 1899, xix, 606; *Iron Age*, 1909, lxxxiii, 1188.

is now again admitted under *a*. The press used as a shear is described by Daelen.[1]

294. Wire-drawing.[2]—This is the operation of reducing the cross-section and increasing the length of a wire by drawing it through a succession of conical tapering holes in a die-plate, each hole a little smaller than the preceding one. One end of the wire is sharpened, passed through the hole and connected with power which pulls through the rest of the wire. The flow of metal in drawing is opposite in direction to that in rolling, as, instead of the top and bottom of a bar being forced onward by the friction between the two surfaces and the rolls, the frictional resistance between the entire bar and the die holds back the surface and allows the center to pass on; thus the curves in Fig. 486 for rolling would be reversed for wire-drawing.

The power that may be applied in drawing must of course always be smaller than the tensile strength of the metal; its range is from 40 to 80 per cent. of the latter. As the tensile strength of a metal decreases with a rise of temperature, metals are always drawn cold. The resistance a wire offers to drawing is governed by the hardness of the metal, the intended reduction, the form of the hole in the die, and the speed of drawing. As the hardness of a metal is increased by working, wire has to be annealed at intervals when it is drawn, and in order to prevent oxidation by the air the wire is heated in a closed vessel which is sometimes filled with illuminating gas. The reduction permissible by a pass is small; the reduction-factor is with soft iron 0.90, steel 0.95, brass and copper 0.925, silver 0.85; *i.e.*, if the diameter before rolling is 1, it will be reduced by the first pass with soft iron to 0.90. The reduction may be greater with smaller sizes than with larger ones. The hole in the die-plate, as shown in Fig. 532, has the form of two truncated cones joined at their bases; the cone through which the wire enters will be pointed. In order to reduce friction, the wire is lubricated with tallow or oil for coarse, with soapy water for medium, and with starch-water for fine sizes. The speed of drawing ranges from 100 to 300 linear feet per min. The die-plate, which is 1/4 to 1 in. thick, is made of white cast-iron for large sizes and of high-carbon steel (2.3 per cent. C.) for medium and fine sizes, and has a number of holes. Holes in cast-iron dies when worn are reamed out for the next larger sizes, in steel plates they are narrowed by hammering and then enlarged to the required size by driving in a punch. The finest sizes of wire (0.001 in.) are drawn through bored rubies or diamonds set in soft metal, the setting being held in a steel disc. Wire-gauges are given on p. 535.

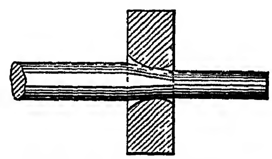


FIG. 532.—Die-plate.

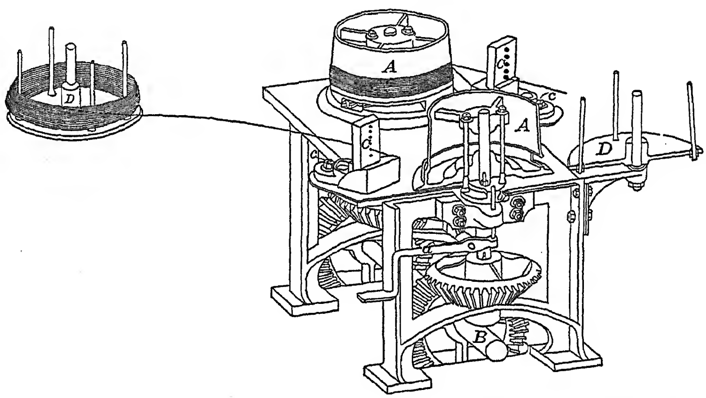
The apparatus for drawing is intermittent or continuous. With the former, the wire is drawn through one die at a time; with the latter, used only for fine sizes, the wire passes through a succession of dies the number depend-

[1] *Tr. A. I. M. E.*, 1892–93, XXI, 324.

[2] Smith, J. B., "Wire, Its Manufacture and Uses," Wiley & Sons, New York, 1891.

ing upon the amount of reduction the wire can stand before it has to be annealed.

Fig. 533 represents two intermittent wire-drawing machines on a single bench. A coil of rod from the rolling-mill, suitably prepared by pickling to remove scale and washing to remove acid, is placed over the reel, *D*. One end having been sharpened is passed through the draw-plate, *C*, which is pivoted to the bench by the bolt, *c*. It is then grasped by nippers, pulled through sufficiently to allow fastening to the cast-iron drum or block, *A*. This is rotated through connecting-pin, *b*, by bevel-gearing driven from the main-shaft, *B*, and draws the wire through the plate and coils it up. The rotation of the drum is stopped (when the rod is to be fastened or when one pass has been made) by placing the foot on the pedal, *a*, which raises the drum sufficiently to disconnect



A The power-driven drum or "block" which draws the wire through the draw-plate, and on which the drawn wire coils.
B The shaft which drives. *A*
C The die or draw-plate through which the wire is drawn.
D The reel on which the wire to be drawn is coiled.
The wire passes from *D* through *C* to *A*.

Fig. 533.—Intermittent wire-drawing machine.

it from *b*. An improved form is shown in the Carrol Friction Block.[1] Draw-benches do not coil up the wire.

A continuous wire-drawing machine (Rankin & Ludington) is shown in Fig. 534. It has four dies, each followed by a revolving flanged wheel, and one finishing-die next to the main drum which is connected to a vertical shaft rotated by bevel-gearing. The wire coming from a reel is passed through the first die, the one farther from the drum, then wound twice around the first flanged wheel which gives it the grip necessary to draw it through the die; it is then passed through the second die and wound again twice around the second wheel and so on. As the length of the wire is increased with the reduction of the cross-section, the second wheel must be rotated faster than the first, the third faster than

[1] *Iron Age*, 1908, LXXXII, 296. Examples: Standard Machinery Co., Providence, R. I., *Iron Age*, 1909, LXXXIV, 1390; Humphrey & Sons, Joliet, Ill. *Op. cit.*, 1910, LXXXV, 973.

the second, and the fourth faster than the third. Allowance has to be made in the increasing r.p.m. for a small amount of slip.

Fig. 535 represents an Eight-block Tandem Wire-drawing Machine of the Waterbury Machine Co., Waterbury, Conn. It draws at one time from eight posts eight brass or copper wires continuously through 13 dies over 24 rolls, 4.5 in. diam., starting with No. 16 and finishing with No. 29 B. & S. gauge.

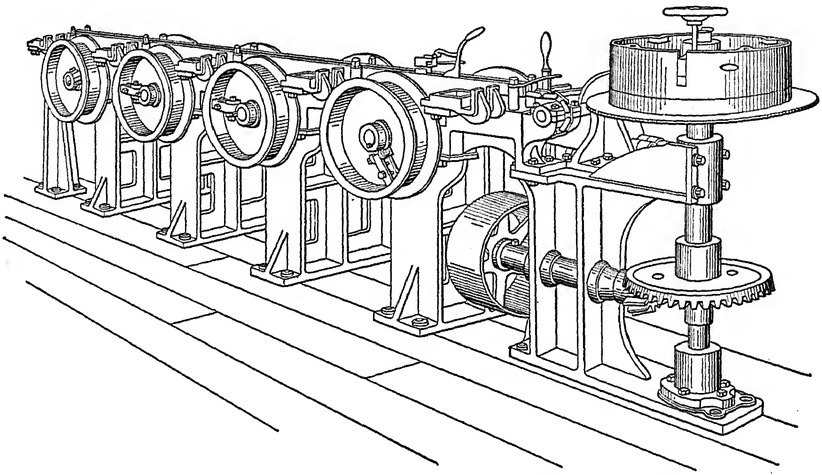


FIG. 534.—Rankin and Ludington continuous wire-drawing machine.

C. LIQUIDS

295. Moving Liquids in General.—The liquids that have to be moved in metallurgical plants vary in character. They are either clear water, or water charged with sand and slime, or charged with salt, acid or alkali. The object of moving the liquid may be to raise it to a higher level, to agitate it in order to keep solids in suspension or cause them to go into solution, to force it through resistances such as a filtering medium. Reciprocating, rotary and centrifugal

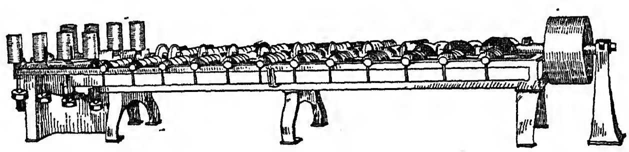


FIG. 535.—Eight-block tandem wire-drawing machine, Waterbury Machine Co., Waterbury, Conn.

pumps, injectors, compressed-air apparatus, and siphons are used for this purpose. The material of which they are constructed, beside having to be of sufficient strength, must be suited to the character of the liquid. Thus cast-iron is the common material for a pump; wrought iron and steel serve for injectors and some compressed-air apparatus. In many cases iron lined with hard-lead, tin, etc., has done well with corroding solutions; tobin bronze will resist a

weak acid, especially dil. H_2SO_4; glass, stone-ware, and earthen-ware have been employed with more or less success in pump-construction.

296. Reciprocating Pumps.[1]—A reciprocating pump may be a piston- or plunger-pump. In a piston-pump, Fig. 536, the water is moved through a cylinder by the action of a solid piston-head fitting tightly into it; in a plunger-pump, Fig. 540, the water is displaced by a solid plunger traveling loosely in a cylinder, but fitting tightly in a ring or stuffing-box in some part of the cylinder. Piston-pumps are used mainly when suction plays an unimportant part. At sea-level, the theoretical figure for maximum height of suction is 34 ft.; on account of losses due to leakage, friction, weight of valves, dissolved air and tension of aqueous vapor, this figure is reduced to from 22 to 26 ft.

Plunger-pumps are usually used with large quantities of water, which may be muddy, and with high lifts, overcoming pressures of say 300 lb. per sq. in. A plunger-pump has less friction and a more accessible packing than a piston-pump. The latter takes up less room for a given capacity. A modification of the regular piston-pump suitable for low lifts is the bucket-pump, which is a single-acting vertical piston-pump with one or more valves opening upward in the piston-head. Combinations, as of bucket- and piston-, bucket- and plunger-, piston- and plunger- (differential) pumps can be mentioned by name only. A piston is always packed inside, a plunger may be inside-packed, which is the case with clear water, or outside-packed, mainly with muddy water. The suction-pipe frequently has a strainer to exclude foreign material. The inlet- and outlet-valves of a cylinder or barrel have various forms; the leading types are clack-, wing-, spindle-, disc-, and ball-valves. The last are suited especially for muddy water. A reciprocating pump is essentially a slow-speed machine, on account of the severe shocks that a frequent reversal of the motion would cause with the incompressible water, and on account of the time required by the valves to fall into place and to seat themselves quietly. The piston-speed is about 100 ft. per min. It is desirable that there shall be a uniform pressure in the delivery-pipe of a pump, and this requires a uniform speed. A single-acting pump with flywheel does not attain the uniform speed of a double-acting pump.

Duplex pumps give a more uniform discharge than single pumps; they do not require as large an air-chamber; they need less space and have less weight than two single pumps of the same combined capacity. Piston-pumps are usually double-acting; plunger-pumps are as often single- as double-acting; the inherent irregularities are overcome by having a number of plungers deliver into a single main.

The performance of a pump is termed its duty. This is usually measured by pounds of fuel consumed, and is given in terms of ft. lb. of work per 100 lb. coal. The duty of a good pumping engine is about 150,000,000 ft. lb., which is equivalent to a utilization of 90 per cent. of the i.h.p. in the steam-cylinder; the remainder is absorbed mainly by the friction of the water in its passage through the pumps. A horse-power of water-work would be developed on about 1.35 lb. of coal.

[1] Barr, W. M., "Pumping Machinery," Lippincott, Philadelphia, 1908.

EXAMPLES. 1. THE CAMERON PISTON-PUMP.—Fig. 536 represents[1] a vertical longitudinal section through one-half of the steam-cylinder of a single direct-acting Cameron pump; the other half is shown in elevation with the bonnet

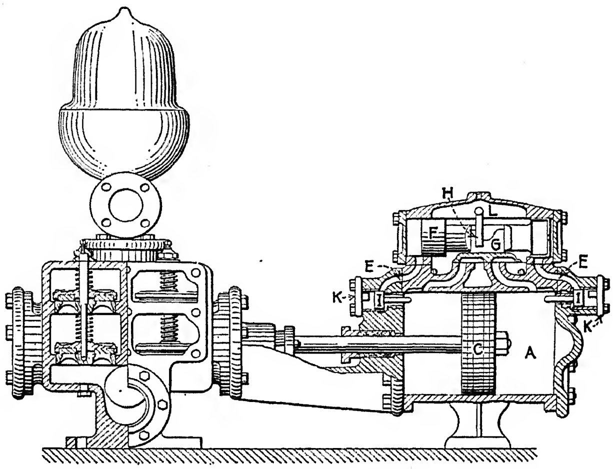


FIG. 536.—Cameron piston pump.

FIG. 536.—*A*, Steam-cylinder; *C*, piston; *D*, steam-chest; *F*, steam-chest plunger, the right of which is shown in section; *G*, slide-valve; *H*, lever with handle outside for reversing, *F*, by hand when necessary; *I*, reversing valves; *K*, bonnets on valve-chamber; *E*, exhaust-ports leading from the ends of steam-chest direct to main exhaust and closed by reversing-valves, *I*; the body-piece marked "Cameron" connects the steam- and water-cylinders.

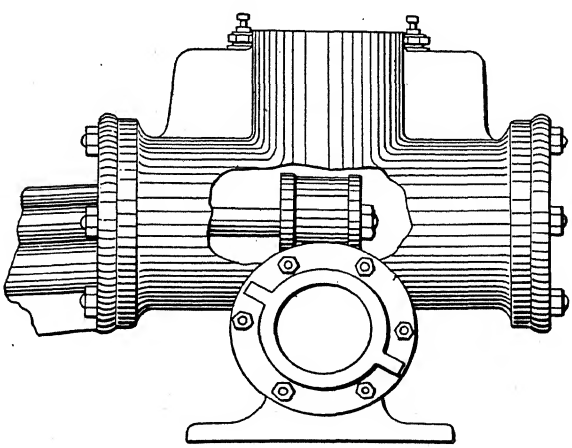


FIG. 537.—Cameron piston pump.

removed. The water-cylinder lying next to the water-valve chest is shown in Fig. 537. The pump has a suction-opening on each side, and a single discharge-opening beneath the air-chamber. There are two pairs of valves, one valve-

[1] A. S. Cameron Steam Pump Works, New York, N. Y.

stem being common to a pair. The valves are of the disc pattern, brass shells filled with vulcanized rubber held down by springs.

In operating, steam admitted to chest, *L*, fills the hollow ends of plunger, *F*, down to reversing valves *I*. In Fig. 536, plunger *F* and slide-valve *G* are in position to the right, hence steam will enter in cylinder, *A*, to the right of piston, *C*, moving it to the left. When it reaches reversing valve *I*, it pushes this to the left and at the same time opens port, *E*, through which the steam at the left of the plunger is exhausted; now the steam at the right of *F*, pushes this to the left and with it slide-valve, *G*, and thereby reverses the motion of the piston.

Table 208 represents some of the leading sizes with their capacities.

TABLE 208.—CAMERON PISTON-PUMP

Size number	Diameter of steam-cylinder, inches	Diameter of water-cylinder, inches	Stroke of piston, inches	Capacity per stroke, gallons	Capacity at ordinary speed per minute, gallons	Steam-pipe	Exhaust-pipe	Suction-pipe	Discharge-pipe	Floor-space, inches	Weight
0	3⅓	2	4	0.054	8	⅜	½	1¼	1	32×9	136
1	4	2	6	0.081	12	⅜	½	1¼	1	40×10	210
2	5	2½	6	0.12	18	½	¾	1½	1¼	40×11	254
3	6	3	7	0.21	28	¾	1	2	1½	47×13	418
3a	6	3½	7	0.29	38	¾	1	2½	2	47×15	435
4	7	3½	7	0.29	38	¾	1	2½	2	47×15	459
4a	7	4	7	0.39	50	¾	1	2½	2	51×16	457
5	7	3½	12	0.5	50	1	1½	3	2½	58×17	820
5b	7	5	13	1.10	100	1	1½	4	3	63×20	1,117
6	8	4	12	0.65	65	1	1½	3	2½	58×18	864
6a	8	5	13	1.10	100	1	1½	4	3	63×20	1,160
7	10	5	13	1.10	100	1¼	2	4	3	64×21	1,345
8	10	6	13	1.59	150	1¼	2	4	3½	64×21	1,411
9	2	7	13	2 16	200	1½	2½	5	4	66×24	1,928
10a	14	8	13	2.83	261	2	3	5	5	73×26	2,548
10	14	9	18	4.96	330	2	3	6	5	81×30	3,126
11	16	10½	18	6.75	450	2½	4	8	6	90×37	4,827
12	18	12	20	9.80	587	3	4	10	8	103×41	6,360

2. THE WORTHINGTON PISTON-PUMP.[1]—Fig. 538 shows a vertical longitudinal section of one-half of a Worthington duplex direct-acting piston-pump; it is duplex, as the two direct-acting pumps, placed side by side, Fig. 539, are so combined that the steam-piston of one operates through a rocker and lever; the slide-valve of the other has finished its own stroke and waits for its valve to be acted upon by the other pump before it can resume its motion. This pause allows the water-valves to seat quietly; further, there is no dead-point, as one

[1] Henry R. Worthington, New York, N. Y.

or the other of the steam-valves is always open to receive steam when this is admitted. The packed water-piston, *G*, works in a brass-lined cylinder, *H*. The valves, straight-lift and spring-loaded, are situated above the water-piston

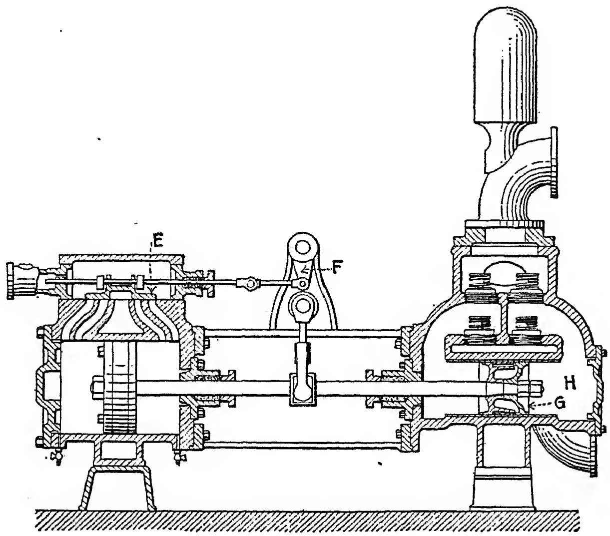


FIG. 538.—Worthington piston pump, sectional view.

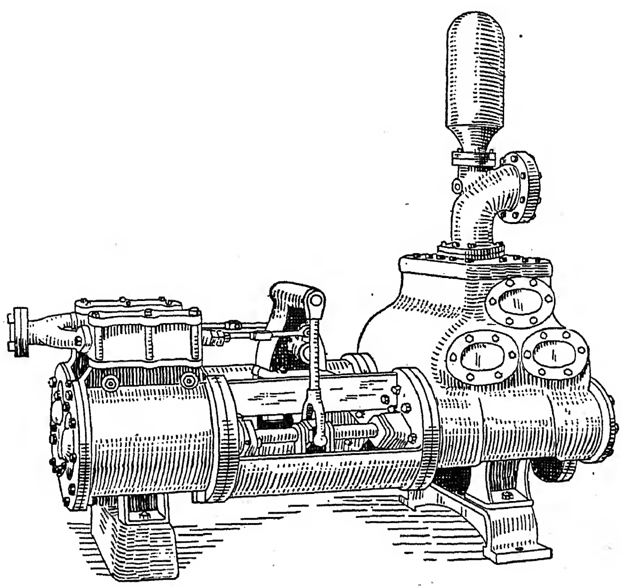


FIG. 539.—Worthington piston pump, perspective view.

which is at all times submerged. Table 209 gives the leading sizes for general service under a pressure of 150 lb. per sq. in.

3. SINGLE DIRECT-ACTING PUMP.—With single direct-acting pumps, the

steam-end is sometimes compounded by the addition of a low-pressure cylinder tandem to that of the high-pressure; duplex pumps are correspondingly cross-compounded.

TABLE 209.—WORTHINGTON PISTON-PUMPS

Diameter of steam-cylinders	Diameter of water-pistons	Length of stroke	Gallons per revolution	Max. revolutions per minute	Max. gallons per minute	Sizes of pipes for short lengths to be increased as length increases				Approximate space occupied, feet and inches	
						Steam-pipe	Exhaust-pipe	Suction-pipe	Delivery pipe	Length	Width
2	$1\frac{1}{4}$	$2\frac{3}{4}$	0 056	80	4.4	$\frac{3}{8}$	$\frac{1}{2}$	1	$\frac{3}{4}$	1 $9\frac{1}{4}$	0 7
3	2	3	0.115	80	12 4	$\frac{3}{8}$	$\frac{1}{2}$	$1\frac{1}{4}$	1	2 0	0 9
$3\frac{1}{2}$	$2\frac{1}{4}$	4	0 275	75	23	$\frac{3}{8}$	$\frac{3}{4}$	$1\frac{1}{2}$	$1\frac{1}{4}$	1 $9\frac{1}{2}$	0 9
$4\frac{1}{2}$	$2\frac{3}{4}$	4	0 39	75	29	$\frac{1}{2}$	$\frac{3}{4}$	2	$1\frac{1}{2}$	2 9	1 1
$5\frac{1}{4}$	$3\frac{1}{2}$	5	0 81	70	56	$\frac{3}{4}$	$1\frac{1}{2}$	$2\frac{1}{2}$	$1\frac{1}{2}$	3 5	1 4
6	4	6	1.25	65	81	1	$1\frac{1}{4}$	3	2	3 7	1 5
$7\frac{1}{2}$	5	6	1.96	65	127	$1\frac{1}{2}$	2	4	3	3 9	1 10
$7\frac{1}{2}$	$4\frac{1}{2}$	10	2.61	54	141	$1\frac{1}{2}$	2	4	3	4 11	1 10
9	$5\frac{1}{4}$	10	3.55	54	192	2	$2\frac{1}{2}$	4	3	5 0	1 11
10	6	10	4.7	54	254	2	$2\frac{1}{2}$	5	4	5 1	2 2
12	6	10	4 7	54	254	$2\frac{1}{2}$	3	6	5	6 2	3 1
12	7	10	6 4	54	346	$2\frac{1}{2}$	3	6	6	6 2	3 0
14	7	10	6.4	54	346	$2\frac{1}{2}$	3	6	6	6 2	3 1
12	$7\frac{1}{2}$	10	7.39	54	399	$2\frac{1}{2}$	3	6	6	6 2	3 0
14	$7\frac{1}{2}$	10	7 39	54	399	$2\frac{1}{2}$	3	6	6	6 2	3 1
16	8	10	8 44	54	455	$2\frac{1}{2}$	3	6	6	6 7	3 10
12	$8\frac{1}{2}$	10	9.56	54	516	$2\frac{1}{2}$	3	6	6	6 7	3 0
14	$8\frac{1}{2}$	10	9.56	54	516	$2\frac{1}{2}$	3	6	6	6 7	3 1
16	$8\frac{1}{2}$	10	9.56	54	516	$2\frac{1}{2}$	3	6	6	6 7	3 10
14	$9\frac{1}{4}$	10	11.37	54	614	$2\frac{1}{2}$	3	8	7	6 7	3 1
16	$9\frac{1}{4}$	10	11.37	54	614	$2\frac{1}{2}$	3	8	7	6 8	3 10

4. PLUNGER-PUMPS.—In Figs. 540 to 543 are given the water-ends of several types of direct-acting plunger-pumps.

Fig. 540 represents a simple form of a single-acting solid plunger[1] that is outside-packed. The suction- and delivery-valves are situated one above the other, and above the discharge is, as usual, the air-chamber to furnish the air-cushion necessary for counteracting the shocks due to the reversals of the moving water.

Fig. 541[2] shows a pump-barrel with a double-acting single solid plunger that is inside-packed.

Table 210 gives size and capacities.

[1] Carr, *op. cit.*, p. 28, Fig. 18.

[2] Epping-Carpenter Co., Pittsburgh, Pa.

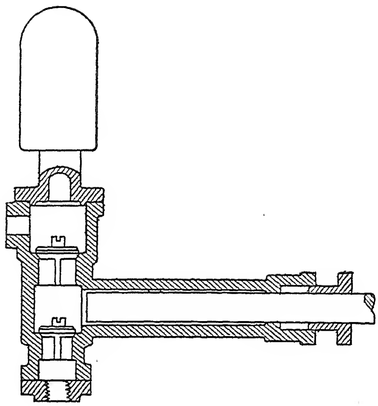


FIG. 540.—Single-acting solid plunger pump.

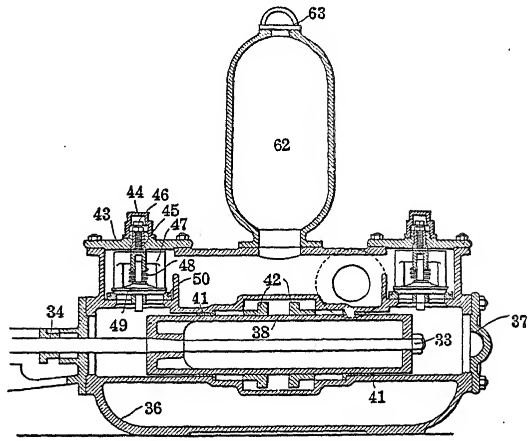


FIG. 541.—Epping-Carpenter double-acting single solid plunger pump.

FIG. 541.—33, Nut; 34, plunger-rod gland; 36, water-cylinder; 38, plunger; 41, packing; 45, valve-cover cap-bushing; 46, set-screw with nut; 47, valve-cage; 48, valve-spring; 50, valve-seat; 62, air-chamber; 63, air-chamber cap.

TABLE 210.—EPPING-CARPENTER DOUBLE-ACTING SINGLE SOLID PLUNGER-PUMP

Diameter of steam-cylinders	Diameter of plunger	Length of stroke	Piston speed in feet per minute of one piston	Gallons delivered per minute by both plungers at stated piston speed	Steam-pipe	Exhaust-pipe	Discharge-pipe	Suction-pipe
7½	4½	10	65 to 100	107 to 165	1½	2	4	5
7½	5	10	65 to 100	133 to 204	1½	2	4	5
9	4½	10	65 to 100	107 to 165	2	2½	4	5
9	5	10	65 to 100	133 to 204	2	2½	4	5
10	5	10	65 to 100	133 to 204	2	2½	4	5
10	6	10	65 to 100	191 to 295	2	2½	4	5
10	5	12	70 to 125	143 to 256	2	2½	4	5
10	6	12	70 to 125	206 to 368	2	2½	4	5
12	6	12	70 to 125	206 to 368	2½	3	4	5
12	7	12	70 to 125	280 to 500	2½	3	5	6
14	7	12	70 to 125	280 to 500	2½	3	5	6
16	7	12	70 to 125	280 to 500	3	4	5	6
18	7	12	70 to 125	280 to 500	3	4	5	6
14	8	12	70 to 125	368 to 655	2½	3	6	7
16	8	12	70 to 125	368 to 655	3	4	6	7
18	8	12	70 to 125	368 to 655	3	4	6	7
20	8	12	70 to 125	368 to 655	4	5	6	7
14	8½	12	70 to 125	414 to 740	2½	3	6	7
16	8½	12	70 to 125	414 to 740	3	4	6	7
18	8½	12	70 to 125	414 to 740	3	4	6	7
20	8½	12	70 to 125	414 to 740	4	5	6	7
14	10	12	70 to 125	570 to 1,018	2½	3	7	8
16	10	12	70 to 125	570 to 1,018	3	4	7	8
18	10	12	70 to 125	570 to 1,018	3	4	7	8
20	10	12	70 to 125	570 to 1,018	4	5	7	8
16	12	12	70 to 125	827 to 1,470	3	4	8	10
18	12	12	70 to 125	827 to 1,470	3	4	8	10
20	12	12	70 to 125	827 to 1,470	4	5	8	10

Fig. 542 illustrates a water-cylinder[1] steam pump divided by a partition into two compartments. In each is a single-acting solid plunger outside-packed. The plungers are connected by cross-heads and side-rods which serve to transmit the power.

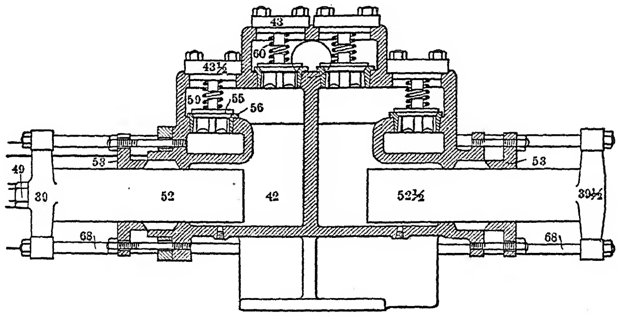


Fig. 542.—Buffalo two-compartment single-acting solid plunger pump.

Fig. 542.—39 and 39½, Cross-heads; 43, pump cylinder-head; 43½, suction-chamber head; 49, plunger-nut; 52, front (or main) plunger; 52½, rear plunger; 53, plunger-gland; 55, valve; 56, valve-seat; 59, suction-valve spring; 60, discharge-valve spring; 68, side or stretcher rods.

Table 211 gives sizes and capacities.

Table 211.—Buffalo Two-compartment Single-acting Solid Plunger-pump

Diameter steam-cylinders	Diameter water-plungers	Length of stroke	Number of gallons per revolution (4 single strokes)	Normal capacity in gallons per minute	Pipe sizes				Approximate outside dimensions, inches
					Steam	Exhaust	Suction	Discharge	
4½	1½	4	0.122	6	½	¾	1½	1	18× 48
4½	2	4	0.213	11	½	¾	1½	1	18× 48
5¼	3	6	0.734	36	¾	1¼	3	2	20× 70
6	3	6	0.734	36	1	1½	3	2	20× 70
6	3½	6	1.00	50	1	1½	3	2	20× 70
7½	4	8	1.74	82	1½	2	4	3	26× 83
7½	4½	8	2.20	104	1½	2	4	3	26× 83
7½	4½	10	2.75	123	1½	2	4	3	26× 87
10	4½	10	2.75	123	2	2½	5	4	26×100
10	5	10	3.40	153	2	2½	5	4	26×100
10	6	10	4.89	220	2	2½	5	4	26×106
12	6	12	5.88	235	2½	3	6	5	40×130
12	7	12	8.00	320	2½	3	6	5	40×130

Fig. 543 represents in one diagram the piston pattern (lower section of water-cylinder)[2] and the plunger and ring pattern (upper section). The plunger in the plunger- and ring-pump is made hollow and of such thickness as to be equal in weight to the displaced water. It thus exerts no pressure on the ring in which it works and can be run without requiring any packing as long as the cast-iron

[1] Buffalo Steam Pump Co., Buffalo, N. Y.

[2] Buffalo Steam Pump Co., Buffalo, N. Y.

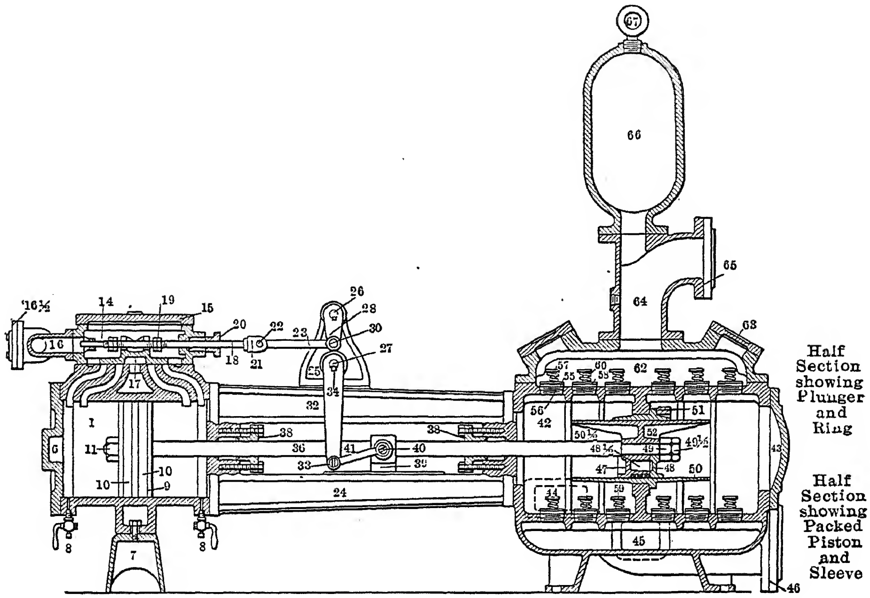


FIG. 543.—Buffalo duplex pump.

1. Steam-cylinder (1 and 2) pump body in small duplex-pumps.
6. Steam-cylinder head.
7. Steam-cylinder foot.
8. Drain cock.
9. Steam piston.
10. Steam piston-ring.
11. Steam piston-nut.
14. Steam chest.
15. Steam-chest cover.
16. Steam pipe.
$16\frac{1}{2}$. Steam-pipe screw flange.
17. Slide valve.
18. Valve rod.
19. Valve-rod nut.
20. Valve-rod gland.
21. Valve-rod head.
22. Valve-rod head pin.
23. Long valve-rod link.
24. Cradle.
25. Cross stand.
26. Upper rock-shaft.
27. Lower rock-shaft.
28. Long crank.
30. Crank pin.
32. Short valve motion lever.
33. Lever pin.
34. Rock-shaft key.
36. Piston-rod.
38. Piston-rod gland.
39. Crosshead (front of main).
40. Crosshead pin.
41. Crosshead link.
42. Pump-cylinder.
43. Pump-cylinder head.
44. Pump-cylinder hand-hole plate.
45. Suction chamber hand-hole plate.
46. Suction screw flange.
48. Pump-piston follower.
$48\frac{1}{2}$. Piston packing.
49. Piston or plunger nut.
$49\frac{1}{2}$. Piston or plunger check-nut.
50. Piston sleeve.
$50\frac{1}{2}$. Plunger sleeve.
51. Binder ring.
52. Front plunger (main).
55. Valve.
56. Valve-seat.
57. Valve-stem.
58. Valve-washer.
60. Discharge-valve spring.
62. Force chamber.
63. Force chamber hand-hole plate.
64. Delivery tee.
65. Delivery screw flange.
66. Air-chamber.
67. Air-chamber eye-bolt.

plunger and brass ring are accurately turned and bored. The pump requires clear water and cannot work against any high pressure (maximum 75 lb.).

TABLE 212.—BUFFALO PLUNGER- AND RING-PUMPS

Diameter steam cylinders	Diameter water-plungers	Length of stroke	Number of gallons per revolution (4 single strokes)	Normal capacity in gallons per minute	Pipe sizes				Approximate outside dimensions, inches	Prices	
					Steam	Exhaust	Suction	Discharge		Regular fitted	Brass fitted
7½	6	10	4.9	220	1½	2	5	4	24× 61	$400	....
7½	7	10	6.6	300	1½	2	6	5	24× 65	500	....
7½	8	12	10.4	418	1½	2	7	6	30× 81	550	....
7½	8½	12	11.8	472	1½	2	8	6	30× 81	560	....
9	8	12	10.4	418	2	2½	7	6	30× 81	570	....
9	8½	12	11.8	472	2	2½	8	6	30× 81	580	....
10	8½	12	11.8	472	2	2½	8	6	30× 81	590	....
10	9	12	13.2	538	2	2½	8	7	30× 81	650	....
10	10	12	16.3	653	2	2½	10	7	36× 81	675	....
10	12	12	23.5	940	2	2½	10	8	40× 81	850	....
12	12	12	23.5	940	2½	3	10	8	44× 90	900	....
12	14	12	32.0	1,280	2½	3	14	10	52× 94	1,100	....
14	14	12	32.0	1,280	2½	4	14	10	52× 96	1,175	....
18	18	18	79.0	2,667	3	4	18	14	60×120	2,200	....

Brass fittings on all plunger and ring pattern pumps include bronze piston-rods, water-plungers, ring sleeves, valve-seats, bolts, springs, plates and stuffing-box glands, also brass valves if necessary.

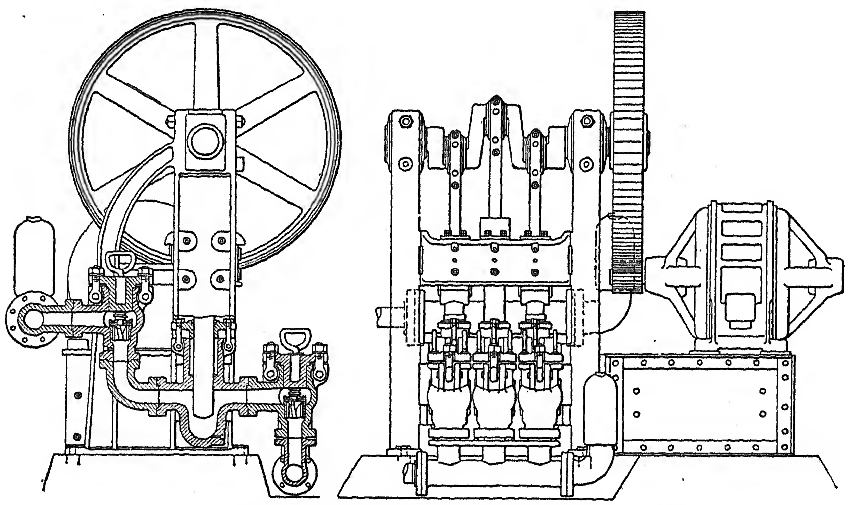


FIG. 544.—Allentown triplex motor-driven plunger pump.

5. BELT- AND MOTOR-DRIVEN PUMPS.—With direct-acting steam-pumps there is a great loss of steam at the end of the stroke, as the steam-cylinder cannot have a cut-off allowing the steam to work expansively. Thus a steam-pump uses 100+ lb. steam per h.p. per hr. Pumps are therefore often belt-driven and more recently motor-driven, the belt or motor receiving its power

from an engine that works with high efficiency, requiring, *e.g.*, 13 lb. steam per h.p. per hr. Fig. 544 represents a vertical section and front elevation of a triplex motor-driven plunger-pump;[1] replacing the motor by a pulley would transform it to a belt-driven pump. Table 213 gives dimensions, capacity and other details.

TABLE 213.—ALLENTOWN TRIPLEX MOTOR-DRIVEN PLUNGER-PUMP

Maximum capacity in U. S. gallons per minute	Pump speed in revolutions per minute	Size of pump		Capacity per revolution	Size of suction and discharge	Motor
		Plunger, inches	Stroke, inches			Motor speeds 850 rev. per minute — Size horse-power
100	68	4	9	1 47	4	13
140	61	5	9	2.30	5	20
175	63	5½	9	2 78	5	25
200	61	6	9	3.30	6	25
250	56	7	9	4.50	6	30
350	54	8	10	6.53	7	40
400	49	9	10	8.26	8	50
500	50	9	12	9.91	8	65
600	49	10	12	12.24	9	75
700	47	11	12	14.81	10	90
100	68	4	9	1.47	4	20
140	61	5	9	2.30	5	25
175	63	5½	9	2.78	5	30
200	61	6	9	3.30	6	35
250	56	7	9	4.50	6	45
350	54	8	10	6 53	7	65
400	49	9	10	8 26	8	70
500	50	9	12	9.91	8	90
100	68	4	9	1.47	4	25
140	61	5	9	2.30	5	35
175	63	5½	9	2.78	5	45
200	61	6	9	3.30	6	50
250	57	6	12	4.40	6	65
350	57	7	12	5.98	7	90
400	51	8	12	7.83	8	100
100	68	4	9	1.47.	4	35
140	61	5	9	2.30	5	50
175	63	5½	9	2.78	5	60
200	61	6	9	3 30	6	75
250	57	6	12	4.40	6	90

[1] Allentown Rolling Mills, Allentown, Pa.

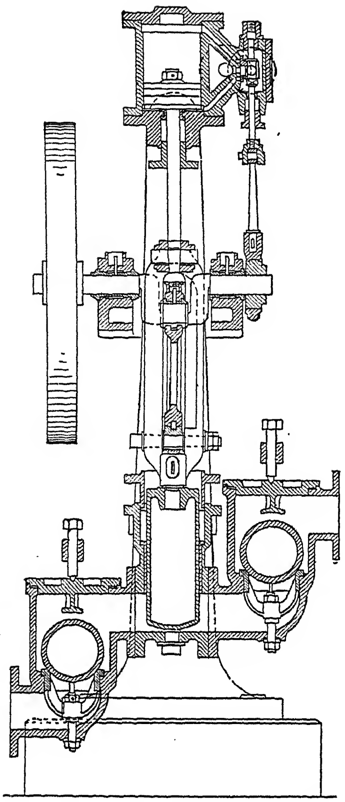


FIG. 545.—Klein-Schanzlin-Becker diaphragm pump.

6. DIAPHRAGM PUMPS.—This is a special form of plunger-pump suitable for clear, muddy and gritty corrosive liquids. An upright form[1] is shown in Fig. 545. It has a single-acting plunger working in a chamber filled with clean water and separated from the corrosive liquid by a rubber diaphragm. The ball-valves are of hard rubber or of metal covered with rubber. The pumps are usually constructed to work against a pressure of 45 lb. per sq. in. They are often made duplex, also horizontal, and for pressures greater than 45 lb. The plunger is driven by a crank and pulley. Further details are given in Table 214. In the pump made by Dehne,[2] which is similar to that shown in Fig. 545, the plunger is supposed to be direct-connected with the steam-cylinder.

TABLE 214.—DEHNE DIAPHRAGM-PUMPS

No.	Diameter plunger, mm.	Diameter steam-cylinder, mm.	Length of stroke, mm.	Capacity per hour, liters	Revolutions per minute	Diameter suction- and discharge-pipes, mm.	Weight, kg.
3	50	80–100	130	2,000	80	30	600
4	65	100–125	160	3,600	70	40	850
5	80	125–150	200	5,800	60	50	1,250
6	100	150–200	250	9,400	50	60	1,850
7	125	175–250	300	14,000	40	80	2,850
8	150	200–300	350	18,000	30	100	4,150

[1] Klein, Schanzlin and Becker, Frankenthal (Rheinpfalz), Germany.
[2] Machinenfabrik, A. L. G. Dehne, Halle, a. S., Germany.

297. Rotary Pumps.—A rotary pump,[1] as shown in Fig. 546, consists essentially of a pair of spur gears running in a closed casing. The points of the teeth, which are in contact with the case, pick up the liquid on the suction side, hold it in a series of pockets which they form, and discharge it on the delivery side. The shafts are driven by exterior spur gears on both sides of the pump-case in order that the interior gears shall not transmit any power. The pumps deliver a large volume of water against a high pressure (from 100 to 200 lb. per sq. in.), but do it at the expense of considerable power which is wasted in friction, in the necessary leakage at the closing of a chamber, and in the shocks produced by the teeth suddenly picking up water, changing its direction, imparting to it a high velocity, and discharging it into the delivery chamber. The pumps are used under conditions where power is of minor importance or where solutions have to be pumped only at intervals. In either case the liquid ought to be clear. The pumps will lift water 8 to 12 ft. Their mechanical efficiency is 50 per cent. Tables 215 and 215*a* give details, capacity and power.

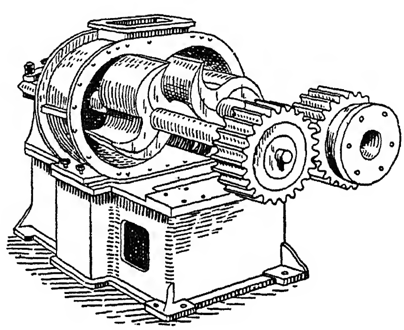


FIG. 546.—Gould rotary pump.

298. Centrifugal Pumps.—The leading parts of a centrifugal pump[2], Figs. 546 and 547, are a fan (impeller) having a number of backward-curved vanes (blades) which rotates in a closed casing. The blades *u, v, w, x, y, z,* are attached to one end of the shaft, *s*, which is belt- or motor-driven at the other end; in rare cases only is it direct-connected to a steam-engine. Water is sucked in at the center, *I*, of the casing, whirled to the periphery at an accelerating speed, and made to travel with a decreasing velocity along the snail-shaped channel, *ABC*, to the discharge, *O*. These pumps are used for moving large volumes of water, clean, muddy or sandy, against a head of 30 to 40 ft. They work steadily and evenly, and are therefore especially suited to be driven by an electric motor. They require little foundation, but must have a careful alignment. The pressure exerted by the pump depends upon the diameter and speed of the vanes. If the normal speed of 10 to 12 ft. per sec. is reduced as little as 10 per cent. and the head remains constant, water will cease to flow; if the head is reduced and the speed kept constant, the amount of water discharged will be greatly increased; if the delivery-head exceeds the pressure of the centrifugal force of the water, the water in the pump will be churned around the vanes and become heated, but will not be discharged. The pump therefore is not suited for a varying duty, but must be chosen or built to satisfy given uniform conditions.

[1] Goulds Mfg. Co., Seneca Falls, N. Y.

[2] Illustrations of Steam Engines by Instructors of Steam Engineering, Mass. Institute of Technology, Boston, Mass., 1908.

TABLE 215.—GOULD ROTARY PUMPS: DETAILS

No.	Capacity one revolution	Speed and capacity per minute, varying with kind of work and pressure[1]	Suction	Discharges	Tight and loose pulleys
1	0.25 gal.	100 to 250 rev., 25 to 60 gal.	3 in. pipe	1½ in. pipe, 1½ in. hose	12×3½ in.
2	0.50 gal.	100 to 200 rev., 50 to 100 gal.	4 in. pipe	2 in. pipe, 2 in. hose	16×4 in.
3	1.00 gal.	100 to 175 rev., 100 to 175 gal.	5 in. pipe	2½ in. pipe, 2½ in. hose	18×5 in.
4	1.67 gal.	100 to 150 rev., 165 to 250 gal.	6 in. pipe	4 in. pipe, 2½ in. hose	24×6 in.
5	2.50 gal.	75 to 125 rev., 185 to 310 gal.	7 in. pipe	5 in. pipe, 2½ in. hose	30×8 in.
6	4.50 gal.	60 to 100 rev., 270 to 450 gal.	8 in. pipe	6 in. pipe, 2½ in. hose	36×10 in.

[1] Speeds given are a fair rate for continuous running; they can be doubled for occasional service.

TABLE 215a.—GOULD ROTARY PUMPS: CAPACITIES AND POWER

Size	Revolutions per minute	Gallons per minute	75 lbs. pressure, horse-power	100 lbs. pressure, horse-power
No. 1	350	87.5	8	10
	400	100	9	12
	450	112.5	10	13
	500	125	11	15
No. 2	250	125	11	15
	300	150	13	18
	350	175	16	20
	400	200	18	24
No. 3	250	250	18	24
	300	300	23	30
	350	350	26	34
	400	400	30	40
No. 4	200	325	23	31
	250	405	30	39
	300	500	35	47
	350	580	45	60
No. 5	150	375	27	36
	200	500	36	48
	250	625	45	60
	300	750	54	72
No. 6	100	450	33	43
	150	675	49	65
	200	900	65	86
	250	1,125	81	108

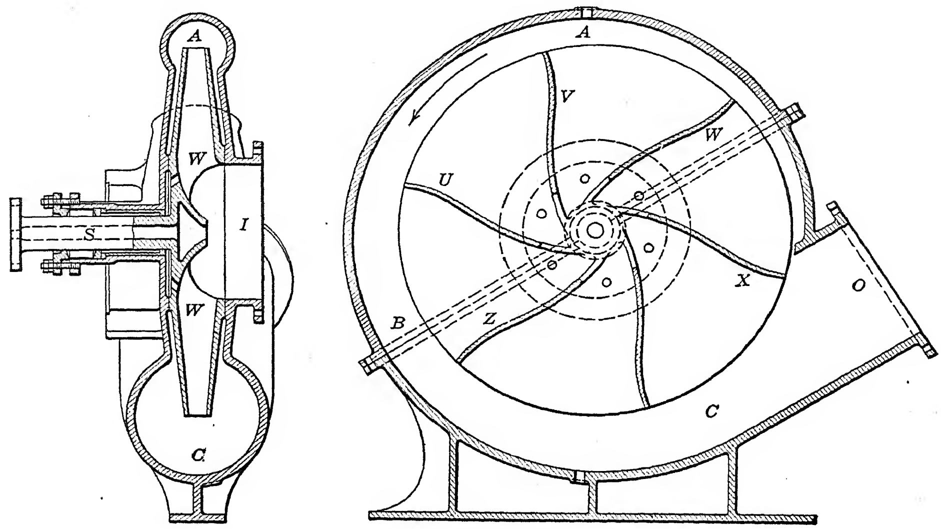


FIG. 547.—Centrifugal pump.

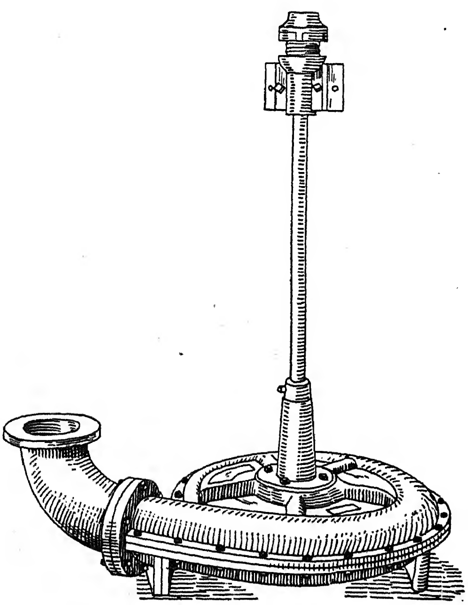


FIG. 548.—Morris submerged vertical centrifugal pump.

The efficiency depends upon the form of the vane and the speed. Experiments with different forms of curves have given the following efficiencies: straight and inclined vanes 32 per cent.; radial 45 per cent., circular 61 per cent. and involute 68 per cent.; with the involute curve and varying heights, efficiencies were obtained ranging from 70 per cent. with a 20-ft. lift to 40 per cent. with a 60-ft. lift, and showed a maximum with a lift of 20 ft. With clear water the efficiency is greater than with gritty water, as in the latter case the speed will have to be reduced in order to prolong the life of the internal parts of the pump. In order to work, the pump must first be primed, *i.e.*, filled with water. In a submerged pump this is, of course, not necessary; in an empty pump, water is run in from the top or drawn up from the bottom. In the former

case, the suction-pipe must have a foot-valve to keep the water from running out. In the latter the discharge-pipe must be closed by a gate-valve in order to permit producing a vacuum by means of an ejector or air-pump placed on top of the casing. The drawing, Figs. 546 and 547, represents a left-hand (fans revolve right to left) single-suction horizontal (shaft horizontal) pump. Fig. 548[1] gives a perspective of a Morris submerged vertical pump, and Fig. 549 one of a suction vertical pump.[1]

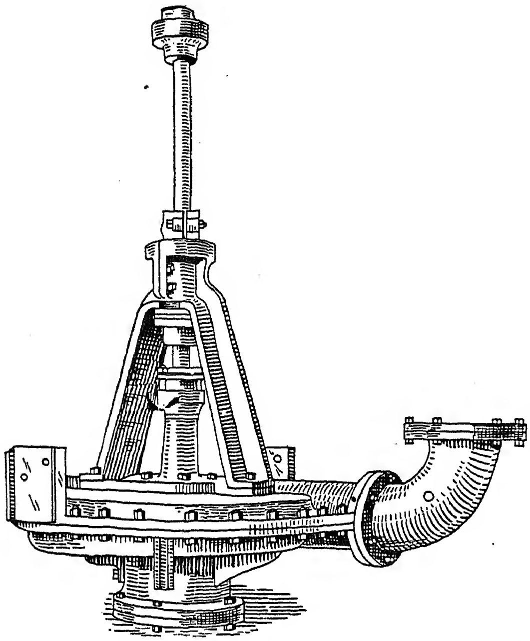


FIG. 549.—Morris suction vertical centrifugal pump.

Fig. 550 represents a double-suction pump,[2] *i.e.*, the suction-pipe is divided, so that the water enters at both sides of the center. This arrangement has the advantage that there is no end-thrust as is the case when the water enters on one side only, but the disadvantage that it works well only with clean water, while the single-suction machine will pass almost anything that enters the suction-pipe. Thus, a single-suction pump driven at high speed will raise water with 50 to 55 per cent. fine or 25 to 35 per cent. coarse sand through a straight vertical pipe 30 ft. in height. The De Laval centrifugal pump with double suction, direct-connected to a steam-turbine or an electric motor making 2,000 r.p.m. raises clear water to heights comparable with those reached by plunger-pumps and attains an efficiency of about 70 per cent.

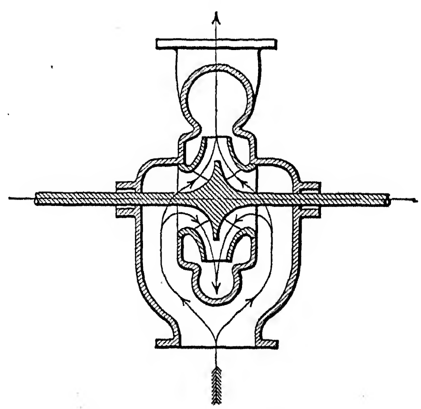


FIG. 550.—De Laval double-suction centrifugal pump.

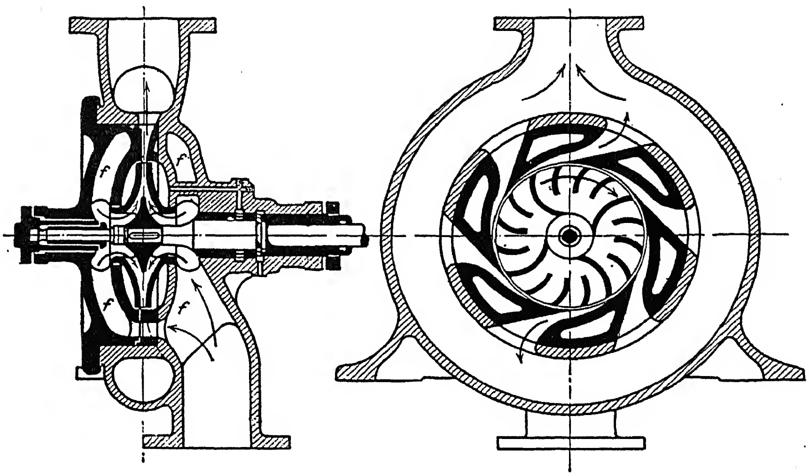
299. Turbine Pump.[3]—This is the commonly accepted term for a liquid-lifting apparatus which consists of a number of centrifugal pumps placed on a single shaft and connected in series in such a way that the discharged liquid

[1] Morris Machine Works, Baldwinsville, N. Y.

[2] De Laval Steam Turbine Co., Trenton, New Jersey.

[3] Van Nedden, *Eng. Mag.*, 1909–10, XXXVIII, 546, 692, 883; 1910, XXXIX, 55, 214.

from one impeller forms the feed of the next following. The liquid being acted upon successively by each of the impellers is delivered at the outlet with a pressure similar to that of a reciprocating pump. Figs. 551 and 552 give two sections of a Sulzer[1] single-wheel (1-stage) high-pressure centrifugal pump. The improvement over the ordinary form consists in the addition of a stationary guide-wheel with diffusion vanes through which the liquid is made to pass after



FIGS. 551 and 552.—Sulzer single-stage high-pressure centrifugal pump.

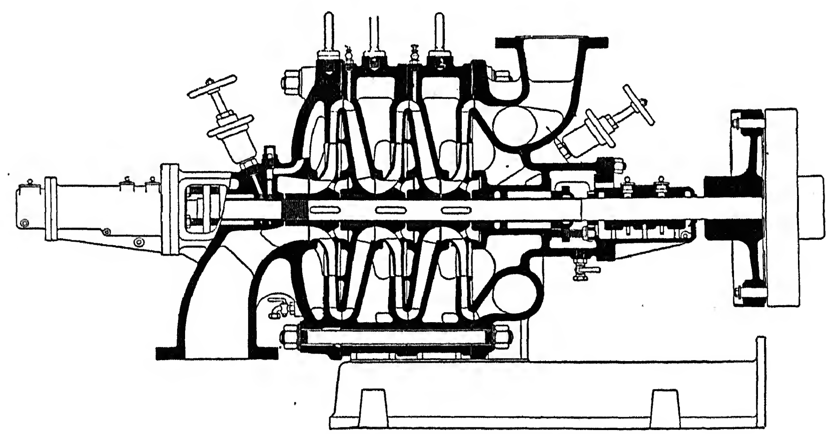


FIG. 553.—Worthington two-stage turbine pump.

leaving the impeller. The velocity of the liquid which is at its maximum at the periphery of the impeller is gradually reduced so as to connect the velocity-head into static pressure. The symmetric entrance of the liquid into the vanes of the impeller is assisted by having in the cover as well as in the main casting concentric suction-spaces which are connected by openings in the guide-wheels.

[1] Illustrations of Steam Engines by Instructors of Steam Engineering, Mass. Institute of Technology, Boston, Mass. 1908.

Fig. 553 gives a longitudinal section of a two-stage Worthington[1] turbine pump. The water leaving the suction-opening is directed by vanes, enters the first impeller through an annular opening at the center and is discharged at the periphery; it now travels through the diffusion vanes and passes through an intermediate casing into the suction-opening of the second impeller; it flows through this in the same manner as with the first and leaves through the delivery with the accumulated pressure. Turbine pumps are made with as many as eight stages; they are built vertical and horizontal. Tables 216 and 217 give the capacities of a single-wheel pump of the two classes *C* and *A* run at 1,450 to 1,470 r.p.m.; class *C* is intended for a head not exceeding 400 ft.; class *A* for a maximum head of 1,100 ft. If a pump is required, *e.g.*, to deliver 295

TABLE 216.—WORTHINGTON CLASS C TURBINE PUMPS FOR A MAXIMUM HEAD OF 400 FEET

No. of pump	Maximum capacity, gallons per minute	At 1,450–1,470 revolutions per minute	
		Height of delivery, feet	Capacity, gallons
1 C	96	$16\frac{1}{2}$	25
2 C	158	$21\frac{1}{2}$	42
3 C	237	33	84
4 C	395	54	185
5 C	622	82	355
6 C	1,000	144	725
7 C	1,480	210	1,320
8 C	2,220	302	2,220
9 C	3,380	400	3,380
10 C	4,480	....	....
11 C	6,330	....	....

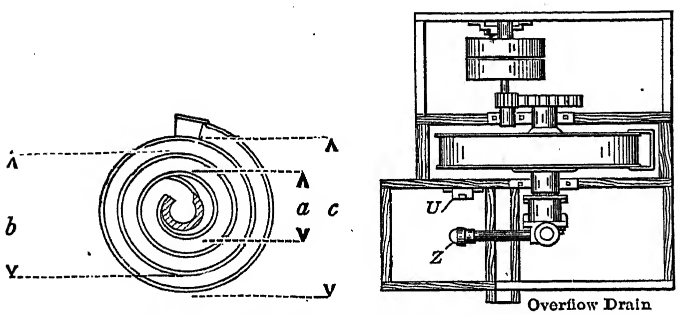
TABLE 217.—WORTHINGTON CLASS A TURBINE PUMPS FOR A MAXIMUM HEAD OF 1,100 FEET

No. of pump	Maximum capacity, gallons per minute	At 1,450–1,470 revolutions per minute	
		Height of delivery, feet	Capacity, gallons
3 A	235	36	90
4 A	395	$65\frac{1}{2}$	195
5 A	635	$98\frac{1}{2}$	390
6 A	1,000	168	790
7 A	1,480	244	1,425
8 A	2,220	348	2,110
9 A	3,380	492	2,610

[1] Henry R. Worthington, New York, N. Y.

gal. per min. against a head of 328 ft. making 1,450 r.p.m., the correct selection will be a four-stage 5*C* pump, as 4×82 ft. = 328 ft. and 355 gal. For a pump to deliver 750 gal. against a head of 672 ft. at 1,450 r.p.m., a four-stage 6*A* pump will be chosen, as 4×168 ft. = 672 ft., and 790 gal. gives the desirable excess.

300. Frenier Spiral Sand Pump.[1]—As shown in Figs. 554 and 555, this is a narrow, hollow, partly-closed cylinder with a spiral passage, which rotating at rate of 20 r.p.m. around its horizontal axis in a box charged with the liquid,



FIGS. 554 and 555.—Frenier spiral sand pump.

scoops this up at the circumference and, passing it along through the spiral passage, discharges it at the center into a lift-pipe. The lift, depending upon the diameter of the wheel, is equal to the sum of the water-columns *a*, *b*, *c*, and does not exceed 24 ft.; for higher lifts, pumps have to be run in tandem. The pump requires less power than a centrifugal pump and suffers less from wear, but it occupies a much larger space.

TABLE 218.—DETAILS OF FRENIER SPIRAL SAND PUMP

Size of wheel, width by diameter, inches	Capacity, maximum per minute (gallons)	Lift, maximum, feet	Weight, pounds
6×44	50	14	
6×48	53	18	1,000–1,200
6×54	54	24	
8×44	60	14	
8×48	70	18	1,200–1,400
8×54	75	24	
10×44	83	14	
10×48	86	18	1,400–1,600
10×54	90	24	

[1] Frenier, J. H., and Son, Rutland, Vt., *Eng. Min. J.*, 1900 LXXXVII, 769.

301. Steam Water-lifters.—The apparatus to be considered are the injector, the ejector and the pulsometer. The efficiency is low, less than 1 per cent., but on account of convenience, this apparatus is applied under conditions in which the heat of the condensing steam is utilized.

1. THE INJECTOR or INSPIRATOR is an instrument for forcing water into a boiler by means of a jet of steam obtained from the boiler to be fed. It is used, however, also for raising liquids. Its operation is based upon the two facts, that water is drawn up to the point of delivery by a vacuum formed by the condensation of steam, and that some of the heat energy of the steam is transformed into kinetic energy thus making the jet of water travel through the water-orifice with a velocity greater than that with which the water would issue from the tank or boiler that is being supplied. Thus steam of 100 lb. pressure can feed water into a boiler which stands under 150 lb. pressure.

Fig. 556 represents a simple form of single-tube injector in which a needle-valve closes the contracted steam-nozzle; this reaches somewhat into the com-

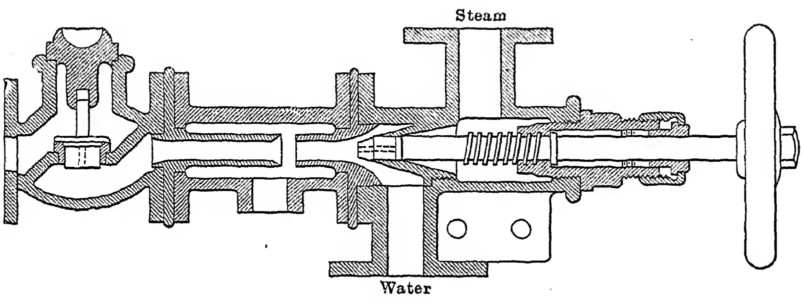


FIG. 556.—Single tube injector.

bining or mixing nozzle, and the continuation of the latter, the forcer, ends in a chamber which is closed by a valve leading into the boiler. In starting the injector, the overflow-valve is opened and some steam turned on gradually to remove condensed water from the chamber and to exhaust the air in the supply-pipe. As soon as a solid stream of water appears at the overflow, the overflow-valve is closed and the steam-valve opened fully, when water will be forced into the boiler. With an aspirating injector the water to be pumped should not be hotter than 30° C., as the colder the water the more rapid the condensation of the steam; with water of 50 to 60° C. the injector will cease to feed. An injector will discharge from 9 to 12 pounds of water per pound of steam supplied. The greater the lift and the higher the discharge-pressure the smaller will be the weight of water and the higher the temperature of discharge. An injector requires from 1,200 to 1,700 lb. of steam per horse-power of water work per hour.

2. The EJECTOR (jet-pump, steam-elevator, steam-siphon) is an apparatus not uncommon in vitriolization plants for lifting warm liquids during short periods of time by means of steam. Its principle is similar to that of the injector, only the construction is more simple on account of the low pressure that

the discharge has to overcome. The working part of the apparatus is an ejecting-pipe inside of which is a nozzle connected with a live-steam pipe represented in Fig. 557. Fig. 558 gives the leading parts of the Koerting "steam-siphon."[1] With 60 lb. steam and an elevation of 20 ft. a 1/2-in. siphon will handle 150 gal. acid of 50° Bé. per hr.; a 1-in. 600 gal.; a 2-in. 2,400 gal.; and a 4-in. 9,600 gal.

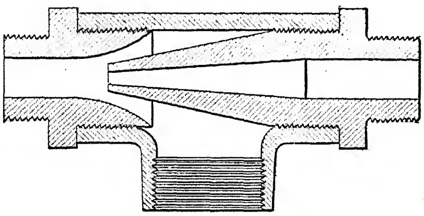


FIG. 557.—Ejector nozzle.

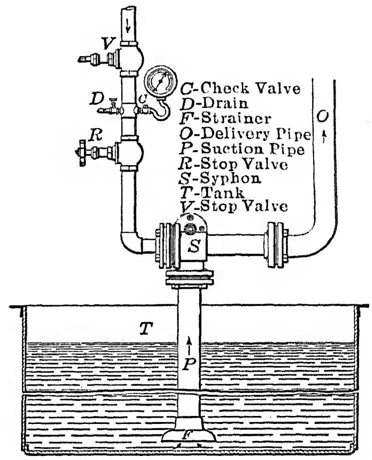


FIG. 558.—Koerting steam-siphon.

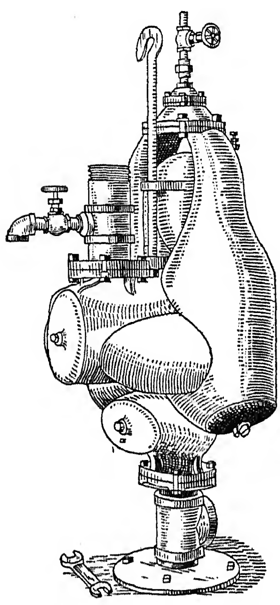


FIG. 559.—Pulsometer.

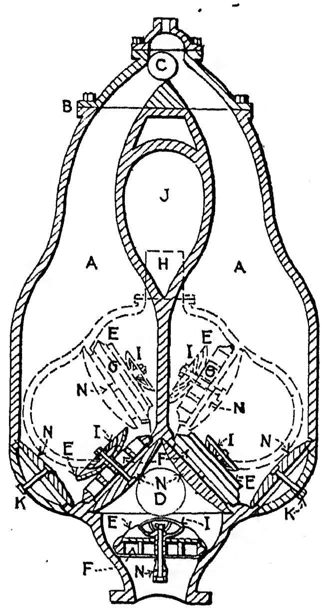


FIG. 560.—Pulsometer.

FIGS. 559 and 560.—*AA*, Bottle-shaped working chambers; *B*, neck-piece; *C*, valve-ball; *D*, suction chamber; *EE*, suction-valves; *FF*, seats of suction-valves; *GG*, seats of discharge-valves; *H*, delivery-pipe; *I*, valve-guards; *J*, vacuum chamber; *K*, hand-holes; *N*, clamps and bolts.

3. THE PULSOMETER.[2]—Fig. 559 gives a perspective view and Fig. 560 a

[1] Schutte & Koerting Co., Philadelphia, Pa.

[2] The Pulsometer Steam Pump Co., New York, N. Y.

vertical section through the apparatus. This consists of two bottle-shaped chambers, *A*, cast side by side, with tapering necks bent toward one another; their upper parts, formed by a separate casting—the neck-piece *B*—terminate in a common steam-chamber. In this chamber the ball, *C*, can oscillate from one side to the other; the lower parts of the chamber are connected through suction-valves, *E*, with suction-chamber, *D*, and with a second set of valves, *E* (dotted lines), through the discharge-chamber with the delivery-pipe, *H*. The vacuum-chamber *J* situated back of the working-chambers, *A*, at the lower ends, is connected with the suction through pipe *D*. Small air-check valves (not shown) in the necks of chambers, *A*, and in chamber, *J*, allow some air to enter in, *A*, above the water to form an air-piston and prevent entering steam from being condensed, and in *J* to take up the water-ram due to the alternate filling of chambers, *A*. In starting, the apparatus has to be primed, *i.e.*, filled with water. When steam is admitted at the top, it passes into whichever chamber is not covered by ball *C*, presses on the surface of the water, forces this down and out through the discharge valves into the rising main, *H*. As soon as the sinking water-level reaches the discharge-valves, steam blows through and is condensed so that a vacuum is rapidly formed which pulls over the ball-valve, *C*, when steam being shut off, the vacuum is completed and water rushes in through the suction-pipe and fills the chamber. The same process now takes place in the second chamber. The alternate filling and discharging of the two chambers keeps up a continuous flow of water.

TABLE 219.—SIZES AND CAPACITIES OF PULSOMETER

Size of pipes, inches				Capacity in gallons per minute at different elevations with boiler-power and steam-pressure usually provided. (Approximate)				Dimensions and weights		
Number	Steam	Suction	Discharge	25 Feet	50 Feet	75 Feet	Horse-power required	Height, inches	Floor space, inches	Weight, pounds
2	1/4	1 1/2	1 1/2	20	17	13	4	25	14×13	95
3	3/8	2	2	60	50	38	5	27	17×14	140
4	1/2	2 1/2	2 1/2	100	80	65	6	33	19×19	295
5	1/2	3	3	180	160	115	9	38	21×22	430
6	3/4	3 1/2	3 1/2	300	265	200	12	43	23×24	570
7	3/4	4	4	425	375	275	15	49	25×26	745
8	1	5	5	700	625	450	25	61	32×33	1,375
9	1 1/2	7	6	1,000	900	650	35	72	38×36	2,100
10	2	8	8	2,000	1,800	1,400	70	88	52×45	3,800

Tests made at the Massachusetts Institute of Technology[1] have shown that the steam consumption per h.p.-hr. reaches 200 lb.

302. Compressed-air Water-lifters.—There are two kinds of apparatus in which compressed air serves to raise liquids. They are represented by the *montejus* and the *air-lift* pump.

[1] *Technology Quarterly*, 1895, VIII, 257.

1. The Montejus is constructed on the principle of the wash-bottle of the chemical laboratory. Fig. 561 represents the acid-egg employed for elevating sulphuric acid. It is a cast-iron vessel (lined or not) of about 40 cu. ft. capacity with delivery-pipe, *A*, filling-pipe, *B*, and compressed-air pipe, *C*. In some acid-eggs all the connections are concentrated in the cover. In Fig. 562 is shown a simple form of montejus which acts intermittently; the leading parts are: *a*,

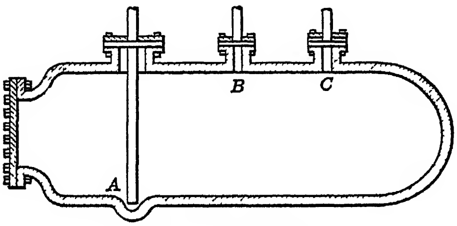


Fig. 561.—Acid egg.

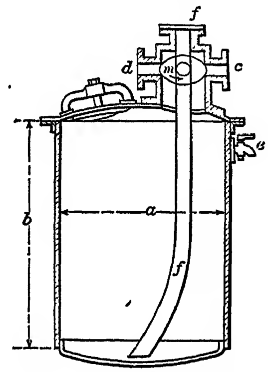


Fig. 562.—Montejus.

cylindrical vessel of steel or copper, with hand-hole and test-cock, *e*, provided with a neck having the solution inlet-pipe, *m*; solution delivery-pipe, *f*, air inlet-pipe, *d*, and air outlet-pipe, *c*. Table 220[1] gives details of the different sizes.

Table 220.—Details of Montejus (Dehne)

Number	Marked	1	2	3	4	5	6	7	8
Capacity, liters, per minute.									
Diameter, mm.	*a*	500	750	1,000	1,500	2,000	3,000	4,000	5,000
Height, mm.	*b*	900	900	950	1,100	1,200	1,400	1,570	1,600
Inlet-pipe for liquor, mm.	*m*	900	1,200	1,410	1,580	1,770	1,940	2,060	2,500
Outlet-pipe for liquor, mm.	*f*	50	50	50	65	65	65	80	80
Inlet-pipe for air, mm.	*d*	50	50	50	65	65	65	80	80
Outlet-pipe for air, mm.	*c*	25	25	30	30	40	40	50	50
Test-cock, mm	*e*	20	20	20	20	25	25	25	25
Weight, kg. mm.	....	540	600	700	860	1,000	1,350	1,750	2,500

In order to do away with the attendance required for opening and closing the valves while operating, the montejus has been made self-acting. Fig. 563 shows the Schutte and Koerting (Philadelphia) automatic montejus. The liquid, entering at *A*, under a certain head, raises the check ball-valve and flowing into the tank causes the upper float, *C*, to rise; this is connected by a vertical rod with the exhaust-valve, *F*, and the compressed-air inlet, *E*, and rising closes the former and at the same time opens the latter. The liquid is now forced up the delivery-pipe, *G*, by the compressed air. When the level of the liquid in the tank has fallen to a certain point, valves *F* and *E* are automatically reversed, and the liquid flows in again through pipe *A*. The apparatus works with pressures ranging from 30 to 70 lb., and is filled and discharged from 30 to 60 times

[1] A. L. Dehne Mfg. Co., Halle a. S., Germany.

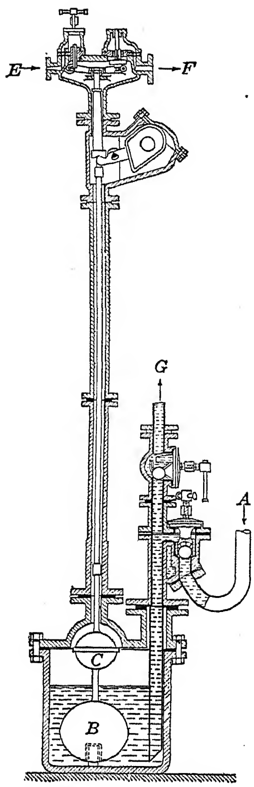


Fig. 563.—Schutte-Koerting automatic montejus.

per min.; it is therefore made much smaller than the ordinary montejus. Thus an intermittent acid-egg of 500-gal. capacity filled once every 30 min. will handle in 24 hr. 24,000 gal. An automatic apparatus of 20 gal. capacity will handle with 30 operations 14,400 gal., and with 60 operations 20,800 gal., *i.e.*, it can easily replace the larger vessel. Other automatic montejus are those of Laurent,[1] Meade,[2] Schuetze.[3]

2. The Air-lift Pump.—The idea of raising a column of water by having compressed air pass through it is old.[4] It was revived or re-invented by Pohlé in 1888[5] and proposed for raising water from wells and shafts. In metallurgical plants the Pohlé air-lift pump has been used for raising electrolyte in copper refining plants,[6] hyposulphite solutions in silver-leaching plants,[7] and slimes and sands in cyanide-leaching works;[8] in the first two instances the air has little or no harmful effect; in the last it is even beneficial.

Fig. 564 represents a diagrammatical sketch of the Pohlé air-lift pump.[9] A delivery-pipe, open at both ends, is submerged in a well to a depth greater than the height to which the solution is to be raised; a compressed-air pipe of relatively small diameter with return-bend enters the delivery-pipe through the bottom or, as in the illustration, through a foot-piece. The compressed air upon entering the larger

[1] *Eng. Min. J.*, 1905, LXXX, 543.

[2] *Met. Chem. Eng.*, 1910, VIII, 545.

[3] *Eng. Min. J.*, 1910, XC, 330.

[4] Stetefeldt, *Eng. Min. J.*, 1890, XLIX, 443.

[5] Brown-Behr, *Tr. Tech. Soc. Pac. Coast*, 1890, VII, 1; *Iron Age*, 1890, XLVI, 254; *Min. Sc.*, 1890, LX, 160; Summary by Rix, *J. Assoc. Eng. Soc.*, 1900, XXV, 173; *Comp. Air*, 1900, V, 1113; Rix, *Min. Sc. Press*, 1910, CI, 505; Green, *Eng. Min. J.*, 1909, LXXXVIII, 251; Anderson, *op. cit.*, 1910, LXXXIX, 256; Lorenz, *Comp. Air*, 1909, XIV, 5361; Westinghouse Air Brake Co., *Eng. Min. J.*, 1909, LXXXVII, 646; Davis-Weidner, *Eng. Mag.*, 1912, XLIII, 260; Richards, *Comp. Air*, 1912, XVII, 6413; *Min. Eng. World*, 1912, XXXVI, 1253; Harris, E. G., "Compressed Air," McGraw-Hill Book Co., New York, 1910, p. 49; Peele, R., "Compressed-air Plant for Mines," Wiley & Son, New York, 1910, p. 443; Davis-Weidner, Univ. Wisc, Bull, 450, Eng. Series, Vol. VI, No. 7; *Met. Chem. Eng.*, 1913, XI, 129.

[6] Hofman, *Tr. A. I. M. E.*, 1904, XXXIV, 311.

[7] Stetefeldt, *Eng. Min. J.*, 1889, XLVIII, 566.

[8] Archibald, *Tr. Austral. Inst. Min. Eng.*, 1902, VIII, 102.
Henderson, *Engineer* (London), 1908, CV, 26; *Comp. Air*, 1908, XIII, 4780.
Anon., *Eng. Min. J.*, 1911, XCI, 706.
Allen, *Min. Mag.*, 1912, VI, 285; *Met. Chem. Eng.*, 1912, X, 377.

[9] Peele, R., *op. cit.*; see also *Eng. Min. J.*, 1909, LXXXVII, 646.

pipe forms air-bubbles which in the shape of piston-like layers,[1] alternating with the water, rise rapidly, lift the water and discharge it at the top, provided that the cross-section of the delivery-pipe is not too large in comparison with that of the air-pipe. If this is the case, the air is admitted through a number of openings in order that it may become disseminated through the water, in which case the rising of the water is due mainly to the difference in specific gravity between the communicating columns of water and intermixed air and water. Finally, the air under pressure acts to a slight extent on the injector-principle. In Fig. 564, h_1 is the level of water when at rest; h_2, level of water when pump is in operation; H, height of column of mixed air and water; $L=H-h_2$ = net height of lift. The pressure of the air, P, must be at least equal to that of the water-head h_2, and this is equal to 0.434 h_2 per ft. With air-bubbles acting as pistons the pressure of H is theoretically equal to h_2, but on account of the friction in the delivery-pipe, h_2 must be larger. The net lift L ranges from 0.5 to 0.65 h_2. Taking the latter figure, $h_2=\frac{L}{0.65}$; substituting for h_2 its value in P gives $\frac{P}{0.434}=\frac{L}{0.65}$, or $P=0.434\frac{L}{0.65}=0.67L$. Thus for $L=30$ ft., $P=0.67\times30=20.1$ lb., and $h_2=\frac{30}{0.65}=46$ ft. The efficiency of the pump[2] decreases as the height of the lift and the air-pressure increase. With small lifts efficiencies of 30 to 35 per cent. are common; they may rise to 40 and 45 per cent. According to Rix, assuming a velocity of the liquid in the pipe of 4 to 8 ft. per sec., it takes 2 to 3 cu. ft. of air to raise 1 cu. ft. water for a height of 15 to 50 ft., and 3 to 4 cu. ft. for heights 50 to 100 ft.; with very low heads 1 cu. ft. of air will lift 1 cu. ft. water 20 ft.

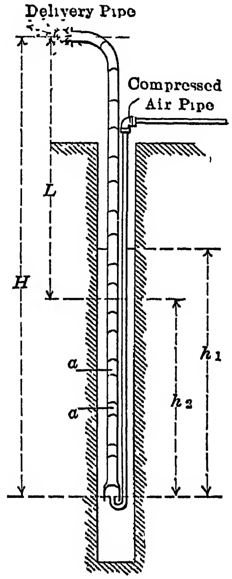


FIG. 564.—Pohlé air-lift pump.

Compound or multiple-stage air-lift pumps[3] hardly come into play in metallurgical plants.

303. **Siphons.**—The simple bent tube used in the laboratory to transfer liquids from one vessel to another has assumed various forms in large-scale work. Its field of operation has been extended by the Steitz siphon to moving melted lead, aluminum, and fusible alloys.

An acid-siphon is shown in Fig. 565. The two limbs, T and L, are connected by the bent tube, M, which carries the bulb, c, with stopper, c, and the branch,

[1] This theory is disputed by Richards, *loc. cit.*

[2] Merriam, *Eng. News*, 1894, XXXII, 27.
Editor, *op. cit.*, 1897, XXXVII, 140.
Johnson, *op. cit.*, 1897, XXXVII, 250.
Friedrich, *Tr. Ohio Soc. Mech., Elect. and Steam Eng.*, 1906.

[3] Hiscox, G. D., "Compressed Air," Henley & Co., New York, 1901, p. 711.

ab. Limb, *L*, and bulb, *c*, are filled with liquid, the feed-opening is closed with stopper, *e*, and limb, *T*, is immersed in the liquid to be transferred. Upon opening cock, *d*, the liquid in *L* begins to flow and empties bulb, *c*. At the same time the air in *T*, passes off through *ab*, into bulb, *c*, and is followed by liquid, and thus the siphon filled. Another form is the Hohmann siphon for carboys.[1]

In concentration pans placed side by side the "ever-set siphon," Fig. 566, is in common use.[2] The limbs are of equal length, and the end of each limb is turned up. It is called "ever-set," because it can be transferred from one pair

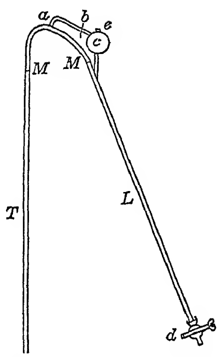


Fig. 565.—Acid siphon.

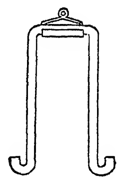


Fig. 566.—Ever-set siphon.

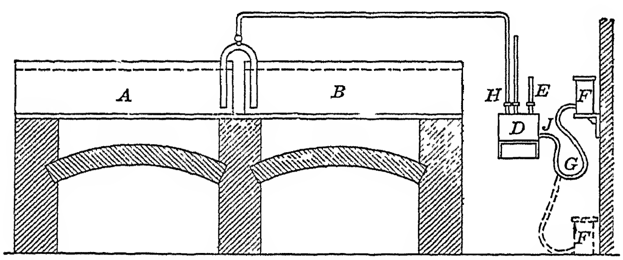


Fig. 567.—Starting siphons from one central place.

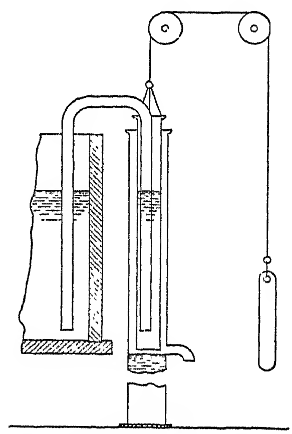


Fig. 568.—Intermittent siphon.

of vats to another without becoming emptied as long as it is kept level. The turned up end of the lifting limb of a siphon permits drawing off liquor from the top of a precipitate without disturbing the latter.

Another way of setting a siphon in pans is shown in Fig. 567. Here the air is drawn out through pipe, *H*, by means of the empty fixed box, *D*, and the movable filled jar, *F*, the two being joined by tubing, *G*. In starting the siphon, cock, *E*, is opened, jar, *F*, is placed on the shelf which is situated above the top of *D*, and filled with enough water to displace the air in *D*; now *E* is closed, *H* opened and *F* lowered. The water running from *D* into *F* draws the air from the siphon. The box, *D*, can have several taps like *H* to serve a corresponding number of siphons.

An intermittent siphon is illustrated in Fig. 568. The discharging limb is

[1] *Stahl u. Eisen*, 1909, XXIX, 250.
[2] *Eng. Min. J.*, 1913, XCV, 1099.

enclosed by a cylindrical vessel which is balanced, and can be raised and lowered in a stationary cylinder filled to the level of its discharge-pipe. In order to start the flow, the balanced vessel is lowered until the surface of its liquid is below

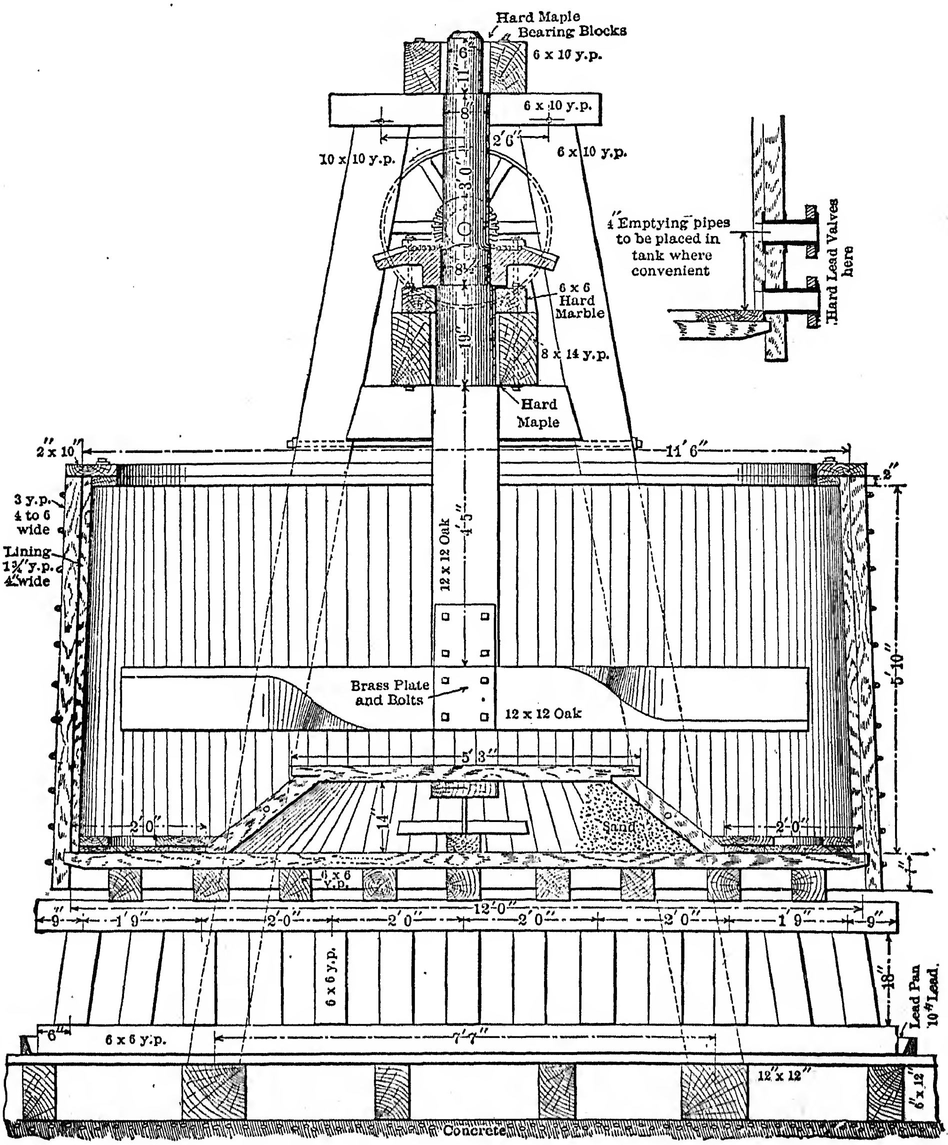


FIG. 569.—O. Hofmann stir-tank.

that in the tank; the siphon acts at once, and fills the balanced vessel which overflows into the stationary cylinder. The liquid then passes off through the discharge-pipe. Raising the balanced vessel until the level of its liquid is above

that in the tank stops the flow; the liquids in vessel and tank then find their common level.

304. Agitators and Mixers.[1]—Agitating in metallurgical plants means the mixing of solid and liquid, or of liquid with gas (aëration, condensation); it thus forms a part of mixing in general which in addition has to deal with the mixing of solid with solid, liquid with liquid, and gas with gas.

The mixing of solid and liquid is usually accomplished by mechanical or compressed-air stirrers, sometimes by withdrawing solution or pulp at the bottom and introducing it again at the top of a tank by means of a centrifugal or an air-lift pump.

Mechanical stirrers in stationary vats resemble one another, as the essential parts are a cylindrical tank and a vertical rotating shaft to which are attached stirring arms or blades. They do satisfactory work, but are generally of small capacities and expensive to operate.

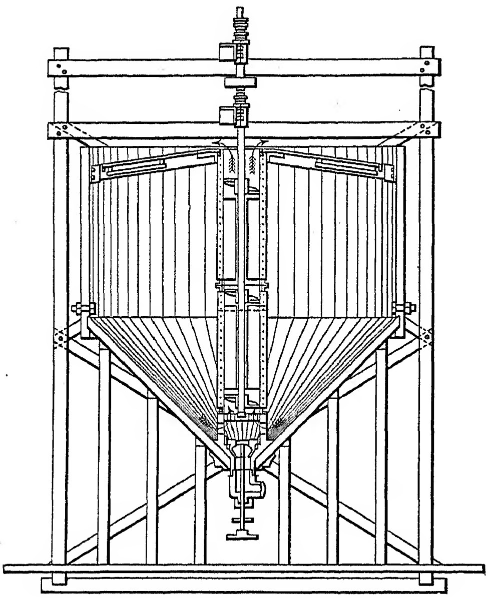


FIG. 570.—Hendryx agitator.

The "stir-tank" constructed by O. Hofmann for the treatment of roasted matte[2] with dilute H_2SO_4, shown in vertical section and elevation in Fig. 569, forms a good example. A wooden tank, 12 ft. in diam. and 6 ft. deep, with bottom and sides lined for protection against wear, has a suspended wooden shaft with radial arms. In the center of the bottom, and fastened to it, is a conical projection of wooden staves filled with sand. It not only prevents matte from accumulating at the center, but forces it toward the periphery where it is subjected to the swift rotating motion of the solvent. Another stir-tank is the A-Z Agitator.[3]

THE HENDRYX AGITATOR[4] works on a different but well established principle. It consists of a circular tank with conical bottom as shown in Fig. 570. In the

[1] Nagel, *Electrochem. Met. Ind.*, 1908, VI, 416.
Warwick, *Min. World*, 1911, XXXIV, 243.
Fischer, H., "Mischen, Rühren, Kneten," Spamer, Leipsic, 1911.
Megraw, *Eng. Min. J.*, 1912, XCIV, 360.

[2] Details: Hofman, "Metallurgy of Copper."

[3] Masson-Edward, *Met. Chem. Eng.*, 1910, VIII, 585.

[4] Bishop, *West. Chem. and Met.*, 1907, III, 187; *Proc. Colo. Sc. Soc.*, 1908, IX, 99.
Wilson, *Mines and Minerals*, 1908, XXIX, 60.
Hendryx, *Min. Sc.*, 1910, LXII, 472; 1912, LXVI, 365; *Met. Chem. Eng.*, 1910, VIII, 601; 1913, XI. 112.

center is a pipe, supported by braces from the side of the vat, which extends from near the bottom to a sloping roof above; in it revolves a shaft carrying a driving-pulley and a number of screw-propellers; these draw the pulp into the pipe and discharge it on to the sloping roof where it comes in contact with the air. The apparatus is built in three sizes, 8 ft. in diam. with 6 tons of ore to a charge, 12 ft. with 18 tons, and 17 ft. with 35 tons. A 17-ft. agitator with a 2-ft. pipe, when charged with 70 tons of solution and 35 tons of ore, requires 8 h.p., and the pulp is turned over every 6 min.

By suspending suction filters in the tank Hendryx[1] combines agitation and filtration as do Blaisdell (§307) and Akins-Rothwell.[2] In recent years whenever possible, mechanical agitators have been replaced by air-agitators. The older apparatus, still in extensive use, consists essentially of a shallow tank, along the bottom of which are placed zig-zag or radial pipes with borings on the upper sides. The compressed air rises through the holes in a finely-divided state, stirs the solution and aërates it at the same time. Such vats require[3]

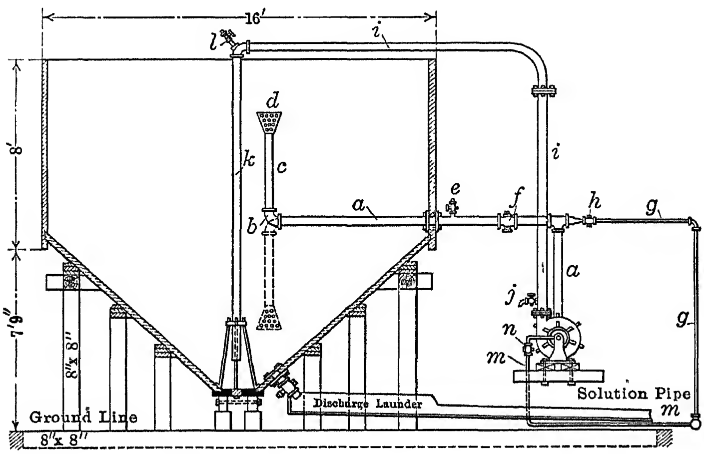


FIG. 571.—Palmarejo agitation vat.

about 200 cu. ft. of air per sq. ft per hr. for ordinary, and 400 cu. ft. for violent agitation. A modern example is furnished in the treatment of anode-mud in an electrolytic copper refinery.[4]

In recent years, in connection with cyaniding, air-agitation of pulp containing mixed slimes and sands, or slimes alone, has become of considerable importance, as it accomplishes with the desired agitation the necessary aëration of the solvent.

THE PALMAREJO AGITATION-VAT[5] is represented in Fig. 571. The cylindrical vat is of 3-in redwood, its conical bottom slants 45° and is closed by a cast-

[1] *Eng. Mag.*, 1908, XXXVI, 485; *Proc. Colo. Sc. Soc.*, 1908, IX, 99.

[2] Colorado Iron Works Co., Denver, Colo.

[3] Davis, G. E., "Handbook of Chemical Engineering," Manchester, 1904, II, 153.

[4] Hofman, "Metallurgy of Copper."

[5] Oxnam, *Tr. A. I. M. E.*, 1906, XXXVI, 269.

iron plate; *a* is a 4-in. suction-pipe with movable elbow, *b*, and pipe, *c*, provided with strainer, *d*, of 1/8-in. iron perforated by 1-in. holes; *e* is an air-cock; *f*, service-cock to close *a* for repairs at the pump; *g*, 2-in. pipe-line, with valve *h*, connected with an upper solution-tank from which, after closing *f*, clear solution is drawn to wash the solid matter from the pump; *i*, 4-in. discharge-pipe; *j*, bibb-nosed pet-cock for taking samples; *k*, drop-pipe, extending to within 15 in. from bottom, supported by clamp and 4 legs made of 3/4-in. bolts; *l*, air-cock; *m*, small pipe-line, with valve, *n*, connected with *g* to supply clear solution (5–6 tons per charge) to bearings of pump. The 4-in. centrifugal pump, lined with manganese steel, is run at 900 r.p.m.; the 3 1/2-in. discharge-cock is placed 10 in. above the bottom.

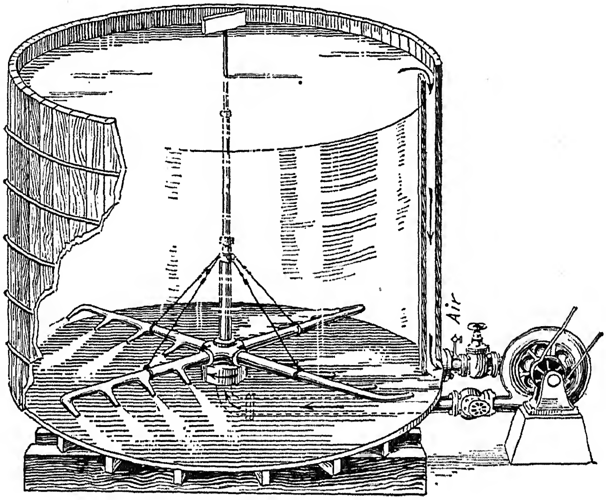


FIG. 572.—Trent agitator.

THE TRENT AGITATOR is an apparatus used for agitating slimes in cyanide plants. Fig. 572 shows a circular tank having in the center, on the bottom, a grit-proof step-bearing connected below with an inlet-pipe, and carrying a reaction wheel with the tangential nozzles pointing downward. The pulp overflowing at the side of the tank is delivered to the suction of a centrifugal pump and forced through the discharge connected with the inlet-pipe of the tank. The pulp forced outward and downward through the nozzles causes the shaft to revolve in an opposite direction; the jets sweep the bottom of the tank, and the contents are continuously agitated. The solution can be aërated by opening the air-cock on the suction-pipe of the pump. A tank 36 ft. in diam. and 22 ft. deep requires 7.5 h.p.

An agitator aiming to stir in the manner of Trent, but in a different way, is the Solis Compressed-air Slimes-agitator.[1]

[1] Lyle, *West. Chem. and Met.*, 1907, III, 213; *Min. Rep.*, 1907, LVI, 474.

Brown or Pachuca Agitating Tank.[1]—This tank which was used at first in New Zealand for slimes alone, is now being widely used for mixed sands and slimes. As seen in Fig. 573 it consists of a cylindrical steel vessel, 14×44 ft., with conical bottom having a slope of 60°. The cylindrical shell extends to the floor, is stiffened by vertical angle-irons and strengthened by triangular braces of angle-iron extending from center to sides. The lower end of the cone is closed by a cast-iron plate which has a 5-in. discharge-hole with quick-opening gate-valve; a doorway in the lower part of the shell gives access to the cone, and a man-hole in the latter allows inspection of the inside. In the center of the tank is a vertical cylindrical tube (air-lift), open at both ends and 1/10 of the diameter of the tank. It extends from about 18 in. from the bottom to about 24 in. from the top. One small pipe (1 1/2 in.) for compressed air passes down the center to the bottom of the tube and allows air to escape as long as its pressure exceeds that of the head of the pulp in the tank. The air lightens the column inside of the tube and causes it to overflow into the tank, in which the solution is kept 1 ft. or more below the top of the tube, while fresh pulp runs in at the bottom. A second 1-in. air-pipe passes downward outside of the central tube and serves to keep the pulp in suspension while the tank is being filled or emptied. Finally, there is an annular casting, near the bottom, surrounding the central tube. It is fed from a solution- or water-pipe, 2 in. in diam., and distributes pressure-water through eight flushing-pipes to wash down sands that settle on the conical bottom after agitation has stopped. The initial air-pressure must be as high as 50 lb., but when started, 25 lb. is sufficient. The amount of air required varies with the proportion of sand and slime. Usually 100 cu. ft. of air per min. will keep the pulp in suspension and circulation.

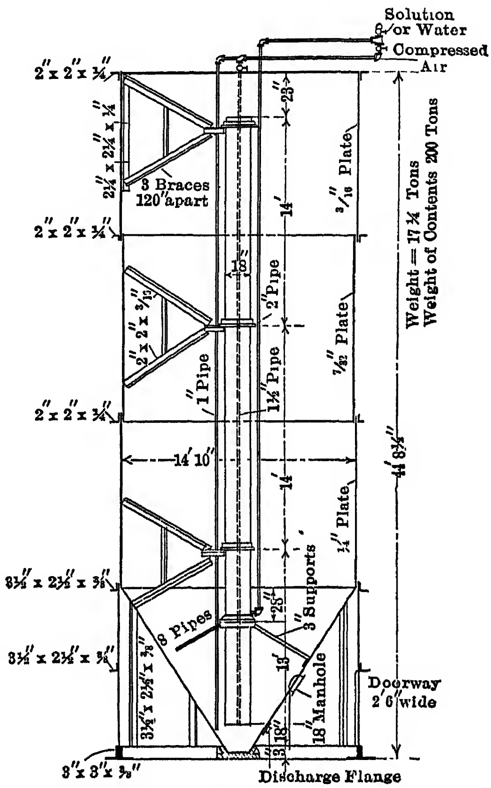


Fig. 573.—Pachuca agitating tank.

[1] *Comp. Air*, 1908, xiii, 4779; *Eng. Min. J.*, 1908, lxxxvi, 471 (Barbour), 652 (Rice), 901 (Lamb); *Min. Sc. Press*, 1908, lxxxvii, 425 (Brown), 1910, ci, 539, 844 (Yaeger), 1911, ciii, 80 (Grote), 409 (Swaren); *J. Chem. Met. Min. Soc. So. Afr.*, 1911, xi, 422 (Allen); *Tr. Mex. Inst. Min. Met.*, 1909–10, i, 213 (Kuryla); *Min. Sc.*, 1910, lxii, 56 (Kuryla); *Min. World*, 1911, xxxiv, 1079 (Grothe); *Tr. A. I. M. E.*, 1911, xlii, 595 (Adams), 807 (Lass).

A plant treating 100 tons of mixed sand and slime per day requires a 10-h.p. compressor. A tank, as shown in Fig. 573, weighs 17 1/4 tons when empty and 200 tons when filled; about 3 1/2 h.p. is required for agitating; a charge of gold ore in cyanide solution takes about 20-hr. treatment.

In regard to power required, the data in Table 221 are given by Lamb.[1]

TABLE 221.—POWER-DETAILS OF PACHUCA TANKS

Tank, diam. by height, feet	Ore	Charge, tons	Free air, cubic feet	Pressure, pounds per square inch	Horse-power	Pulp
7.5×37	Slime.....	15	5	22	0.5	Thin.
7.5×37	Concentrate	40	17	26	2.0	Thin.
10×40	Slime.......	35	9	22	0.75	Thin.
13×55	Slime.......	110	16	33	1.75	Thin.
10×40	Fine sand....	50	25	22	2 25	Thin.
7.5×37	Battery pulp	....	14	22	1.4	Thickened.
10×40	Battery pulp	....	22	23	2 3	Thickened.
13×55	Battery pulp	....	38	35	4 0	Thickened.

Brodie[2] has added circulation-pipes to the Pachuca tank which prevent any accumulation of heavy sands near the bottom and permit the treatment of coarser material than is usually the case.

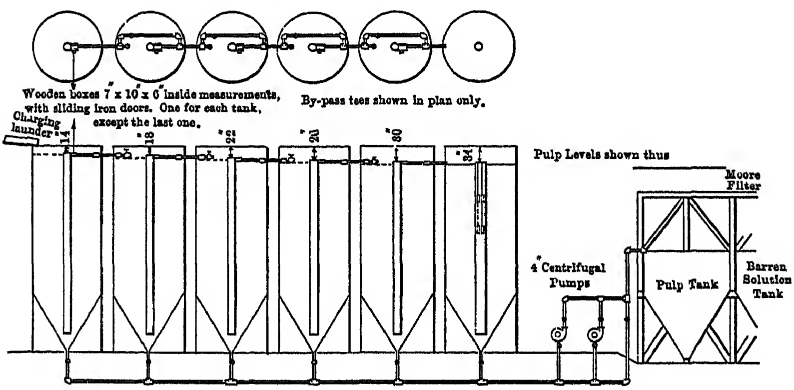


FIG. 574.—Pachuca tanks for continuous agitation.

CONTINUOUS PACHUCA TANK.—The ordinary Pachuca tank requires filling, agitating, and emptying with every charge. In order to make the process continuous, tanks have been arranged in series, with a 4-in. drop from tank to tank, as shown in Fig. 574.[3] Wooden boxes, 7 in. wide×10 in. long and 6 in. deep with

[1] *Eng. Min. J.*, 1908, LXXXVI, 901.
[2] *Eng. Min. J.*, 1909, LXXXVII, 695.
[3] Adams, *Tr. A. I. M. E.*, 1911, XLII, 595.

sliding doors, are fixed against the 15-in. central tubes with their tops flush with the top of the tubes; a 4-in. horizontal pipe passes from the bottom of each box to the next tank in series; by-passes fitted with valves join each pipe-connection with the rest in service. The bottoms of the tanks are connected with two centrifugal pumps. The last tank of the series discharges intermittently into the pulp-tank of a Moore filter (Fig. 610); for this purpose the central air-lift is provided with two pairs of sliding doors which permits agitating at different levels; pulp is drawn off at the required intervals through the bottom either by hydrostatic pressure or by 4-in. pumps.

THE PARRAL TANK.[1]—This is a flat-bottom tank, Figs. 575 to 577, 25 ft. in diam. and 42 ft. high, having four 12-in. vertical pipes, 12 in. from the bottom and 4 ft. from the side; compressed air is admitted to a pipe at the bottom

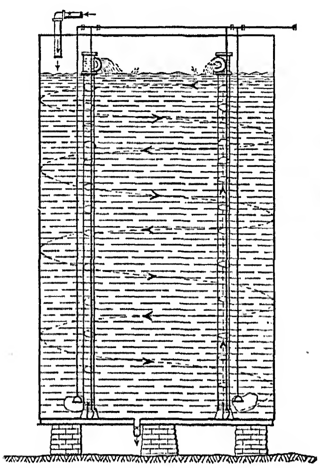


FIG. 575.—Vertical section.

FIG. 576.—Top view.

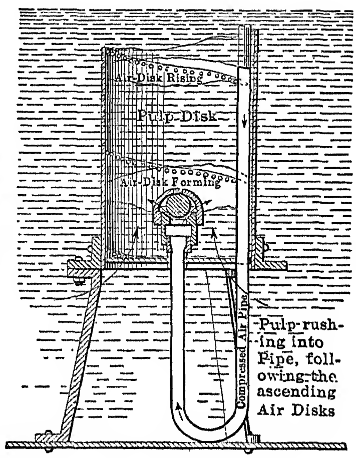


FIG. 577.—Pipe and valve.

FIGS. 575, 576 and 577.—Parral tank.

through a nozzle with ball-valve. The elbows at the top of the pipes have outlets so placed as to discharge the pulp in the lines of segment cords; the force of discharge in the same direction sets up a rotary flow which extends to the bottom of the tank. It is claimed that settling or coning of material does not occur owing to the rotary motion of the pulp; but, even if a cone forms, it is likely to remain constant.

THE ROTHWELL CONTINUOUS SYSTEM[2] aims at solution of metal by agitation, separation of treated pulp, precipitation of extracted metal and discharge of waste liquor. An earlier arrangement is that of Taylor-Jay.[3]

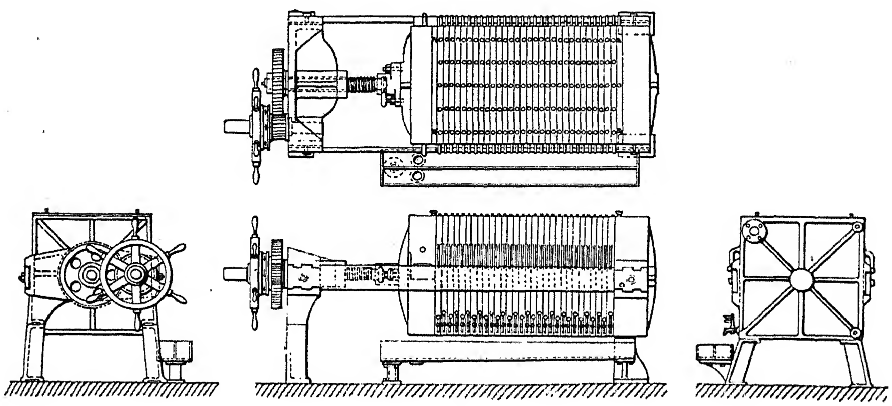
305. Filtering Apparatus in General.—The separation of solid matter from a liquid by the ordinary method of gravity-filtration is satisfactory as long as the

[1] MacDonald, *Tr. A. I. M. E.*, 1911, XLII, 819.
Megraw, *Eng. Min. J.*, 1912, XCIV, 361.

[2] *Met. Chem. Eng.*, 1911, IX, 373.

[3] *West. Chem. Met.*, 1909, V, 167.

ore is sufficiently coarse to allow the liquid to percolate at a desired rate through the open spaces between adjacent particles. With finely-divided suspended substances, such as slimes and many precipitates, which clog the pores of a filter, it becomes necessary to force the liquid through the filter. This must be strong and porous, but still sufficiently close to hold back the solid matter. Forced filtration is accomplished either by a *filter-press* in which the necessary pressure is put on the turbid liquid in a number of connected filtering chambers by means of a pump, an air-compressor, and overhead tank; or by a *pressure-filter* in which pressure is applied to a single tank holding a number of filter-leaves; or by a *suction-filter* in which a vacuum is created inside of the filter-leaves by means of a pump or some other suction-device.



FIGS. 578 to 581.—36-inch square filter-press.

306. Filter-presses.[1]—A filter-press consists essentially of a number (as many as 90) of upright connected chambers (usually rectangular, but sometimes circular or triangular) which are surrounded by a filtering medium, Figs. 578 to 581. A chamber is formed either by adjoining plates (12 to 42 in. sq.) having recessed faces (chamber-presses, Figs. 582 to 585), or by frames (1 to 3 in. thick) being placed between flat plates (frame-presses, Figs. 586 to 589). The plates, with or without frames, are clamped together, Figs. 578 to 581, between a movable and a fixed head by pressure exerted (from a screw, a bell-crank lever,

[1] Charleton, A. G., "Gold Mining and Milling in Western Australia," Spon & Chamberlain, New York, 1903.

Allen, R., "West Australian Metallurgical Practice," Kalgoorlie, 1906.

McNeill, *Tr. Inst. Min. Met.*, 1897–98, VI, 247.

Dixon, *Proc. Chem. Met. Soc. So. Afr.*, 1902–03, III, 13.

Prichard, *Eng. Min. J.*, 1904, LXXVII, 602.

Denny, *Proc. So. Afr. Assoc. Eng.*, 1905–06, XI, 295.

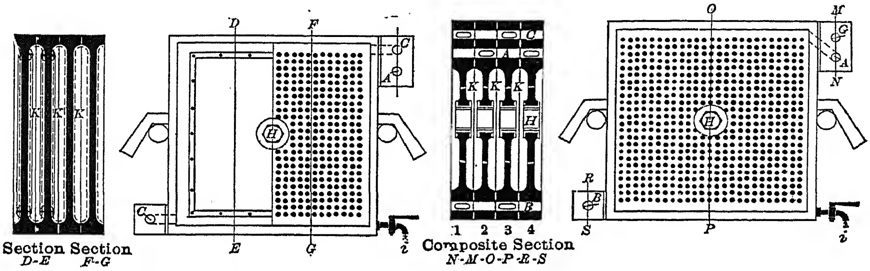
James, *Min. Sc. Press*, 1910, C, 46.

Richter, *Braunkohle*, 1910, VIII, 757.

Young, *Tr. A. I. M. E.*, 1911, XLII, 752.

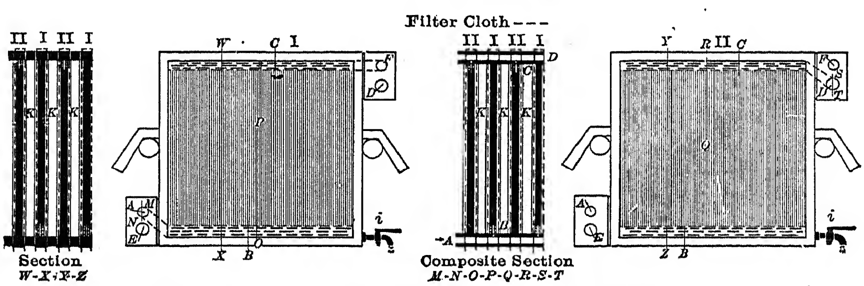
Warwick, *Min. Eng. World*, 1913, XXXVIII, 665.

or pressure-water) upon the movable head in the direction of the axis of the press. The fixed head serves as one of the supports of the machine. A special standard through which the pressure is applied is the other support. The two are connected by a pair of parallel heavy steel bars which take up the working pressure and carry the weights of the plates, the frames and the movable head, each of which has a supporting lug on either side. The chambers are connected with one another by holes, in the centers and margins of the



FIGS. 582 to 585.—Chamber-press with washing channels.

recessed plates, Figs. 582 to 585, or in the margins of the flush-plates and frames, Figs. 586 to 589, which form continuous channels for the liquids. The turbid liquid, forced by means of a gravity-head, a pump with air-vessel, or a *montejus* through the fixed head, travels through one set of channels into the chambers, and the filtrate from back of the filters through another; the solid matter remains in the chambers and forms a cake from less than 1 in. to 3 in. thick, which is removed at intervals after the press has been opened.



FIGS. 586 to 589.—Frame-press with washing channels.

The Merrill and Sweetland presses (see below) are exceptions to this general statement. Separate channels in alternate plates are provided at or near the corners for the admission and discharge of the wash-waters. In metallurgical plants the non-washing filter-presses of some chemical manufacturies are little used, excepting perhaps with presses that are used for clarifying solutions.[1] The plates and frames are made[2] of cast-iron or steel as long as the liquid is not

[1] Yates, J., *Chem. Met. Min. Soc. So. Afr.*, 1906–07, VII, 3.

[2] Hatscheck, *Electrochem. Met. Ind.*, 1905, III, 220.

corrosive; with acid liquors the iron is coated with tin, lead or rubber, or plates and frames are made of antimonial lead, gun-metal or wood.

In plates it is necessary to consider the quality and durability of the border and the panel, while in frames, of course, only the border needs attention. The borders of both are planed so that when forced against the intervening filter-cloths they will form a tight joint (Figs. 503, 582, 586, and 588). In cyaniding the planed borders are painted with P. & B. paint, as this counteracts leakage by securing a good contact. The panel of a plate, usually of the same material

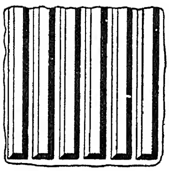


FIG. 590.—Ribbed panel.

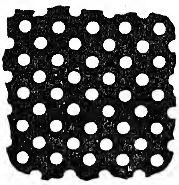


FIG. 591.—Perforated sheet for ribbed panel.

as the border, may have a few ribs covered by a perforated sheet, Figs. 590 and 591, or be corrugated, Figs. 587 and 589, or provided with short truncated pyramids, Fig. 592, or cones, Figs. 583 and 585; the last forms furnish the greatest, the first the smallest area of supported filter-cloth. Along the top and bottom of the panel, Figs. 587 and 589, are drainage-grooves for the filtrate.

The press of the Niles-Bement-Pond Co.,[1] Fig. 593, has a corrugated steel plate, *A*, held loosely in a cast iron frame, *F*; this arrangement permits having a deep groove and reduces the weight of the plate. In some presses used in the

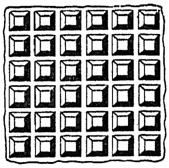


FIG. 592.—Panel with knobs.

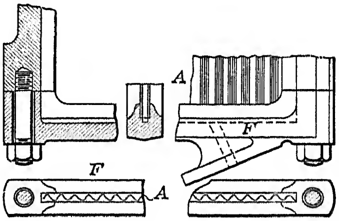


FIG. 593.—Cast-iron frame with corrugated steel panel.

manufacture of chemicals the plates are cast hollow for heating with steam or cooling with freezing-solution.

The common filtering medium is cotton duck; woolen material is used with dilute acids; asbestos cloth with concentrated acid, and paper backed by cotton duck for precious-metal precipitate. The method of attaching the filter-cloth varies with the form of the press and with the character of the liquid. With a chamber-press and a non-volatile liquid admitted through the central channel, the bag-shaped filter-cloth with central holes is slipped over the plate and

[1] *Eng. Min. J.*, 1904, LXXVIII, 994.

tightened at the center by means of two flanged unions, Fig. 584, having male and female screw-threads; the cloths must lie smoothly as they constitute the packing for the borders. With a volatile liquid, the filter-cloths may not extend outside of the press. A single cloth is laid over the panel, Fig. 583, from which project a number of screws, a frame with corresponding holes is placed over the cloth and fastened down with thumb-nuts (Fig. 582, section *D–E*); the central opening is made tight as shown in Fig. 584. In order to make the border gas-tight, one plate has a dove-tail groove carrying a rubber gasket (Fig 582, section *D–E*), while the other has a projecting rib. With a frame-press, the filter-bag, Figs. 586 and 588, is simply slipped over the plate and tightened by the borders. When necessary, the projections or borders through which the channels pass are made tight in both classes of presses by cloth-covered rings placed in recesses made around the holes.

The mode of operating differs with the character of the press. The following description may serve as an example for filtering and washing in a frame-press of the Dehne type.[1] Supposing the press to be closed, as shown in Figs. 578 and 580, the slime-pulp enters the channel, *E*, Figs. 587 and 589, and passes through ports in the frames (not shown) into the chambers, *K*, Figs. 586 and 588; the liquid filters through the cloth on either side, flows downward in the grooves back of the filters and runs along the bottom-channel, Figs. 587 and 589, to the outlet-cock, *i*, of both plates, *I* and *II*. When the chambers are filled with solid material, the channels, *E*, and cocks, *i*, are closed, and wash-water is forced through the wash-water inlet-channels, *A*, Fig. 587, which are connected by ports with plates, *I*, back of the filters; the water rises in the grooves as indicated by the arrows, Fig. 588, spreads over the filter, passes through it into the cake, traverses this, rises in the grooves of plate, *II*, leaves this through its port and enters the wash-water outlet, *D*. When wash-water is admitted in *A*, the air-channels, *F*, in plates, *I*, are opened, and then closed again when water appears in them. Instead of using channels, *D*, the water can be discharged through cocks, *i*, of plates, *II*, by closing *D* and opening *i*, while cock, *i*, in plate *I* remains closed. If the filtrate is not to flow through *i* into a trough, Figs. 578 to 581, but is to be discharged at a higher level, air-channel, *F*, in plate *I* and wash-water outlet channel in plate *II* can be used for this purpose.

With chamber-presses the procedure of filtering differs slightly from that with plate-presses, while the operation of washing is the same. In a chamber-press the turbid liquid enters chambers, *K*, through the central openings, *H*, Figs. 583 to 585; the solid matter remains behind and fills the chamber, while the liquid passes through the adjacent filter-cloths into channels as in plate-presses. A cake of ore-slime will retain from 30 to 50 per cent. water; the cake from a chamber-press usually contains more water than that from a plate-press.

The capacities of some chamber- and plate-presses are given in Tables 222 to 225.

[1] The Klein press, Schanzlin & Becker, *Metallurgie*, 1905, II, 226, is similar.

TABLE 222.—CHAMBER-PRESSES OF W. R. PERRIN & CO., CHICAGO, ILL.

Size of plate, inches	Number of plates in press	Thickness of cake, inches	Filtering area in square feet	Capacity in cubic feet	Working pressure	Weight of plate, pounds	Weight of press, pounds	Cost of plate	Cost of press
18	15	1	44	1.8	150 lbs.	60	1,800	$5.00	$205.00
18	20	1	58	2.4	to	60	2,100	5.00	230.00
18	25	1	72	3.0	square	60	2,400	5.00	255.00
18	30	1	86	3.6	inch	60	2,700	5.00	280.00
18	35	1	100	4.2		60	3,000	5.00	305.00
28	20	1½	154	7.8	150 lbs.	150	5,750	$10.00	$465.00
28	25	1½	191	9.7	to	150	6,500	10.00	515.00
28	30	1½	228	11.6	square	150	7,250	10.00	565.00
28	35	1½	265	13.5	inch	150	8,000	10.00	615.00

TABLE 223.—CHAMBER-PRESSES OF T. SHRIVER & CO., HARRISON, N. J.

Size of press, inches	Number of chambers	Nominal filtering area in square feet	Thickness of cake, inches	Cubic capacity		Working pressure per square inch, pounds
				1 inch cake	1½ inch cake	
12	6	8	1	0.27	....	150
12	12	16	1	0.55	....	150
12	18	24	1	0.82	....	150
12	24	32	1	1.09	....	150
18	12	37	1 or 1½	1.31	1.64	150
18	24	73	1 or 1½	2.62	3.28	150
18	36	110	1 or 1½	3.93	4.91	150
27	18	128	1 or 1½	4.80	6.00	150
27	24	170	1 or 1½	6.40	8.00	150
27	30	213	1 or 1½	8.00	10.00	150
27	36	255	1 or 1½	9.60	12.00	150
27	48	340	1 or 1½	12.80	16.00	150
36	36	461	1 or 1½	16.92	21.15	100
36	48	691	1 or 1½	23.76	29.70	100
36	60	768	1 or 1½	29.70	37.13	100
36	72	921	1 or 1½	35.64	44.55	100

TABLE 224.—FRAME-PRESSES OF W. R. PERRIN & CO., CHICAGO, ILL.

Size of plate, inches	Number of plates in press	Number of frames in press	Thickness of cake, inches	Filtering area in square feet	Capacity in cubic feet	Working pressure	Weight of plate and frame	Weight of press, pounds	Cost of plate and frame	Cost of press
24	30	31	1	190	7.9	150 lbs.	Plate	6,400	Plate	$ 661.00
24	35	36	1	221	9 2	to	95 lbs.	7,075	$ 8.00	731 00
24	40	41	1	252	10.5	square	Frame	7,750	Frame	801 00
						inch	40 lbs.		$ 6.00	
28	30	31	1	269	11.2	150 lbs.	Plate	11,300	Plate	$ 843.00
28	35	36	1	312	13.0	to	165 lbs.	12,550	$10.50	935.50
						square	Frame		Frame	
28	40	41	1	355	14.8	inch	85 lbs.	13,800	$ 8.00	1,028.00
40	30	31	3	573	71.7	100 lbs.	Plate	37,200	Plate	$2,100.00
40	40	41	3	758	94.8	to	390 lbs.	47,000	$30.00	2,580.00
						square	Frame		Frame	
40	50	51	3	943	117 9	inch	310 lbs.	44,200	$18.00	3,060.00

Presses are filled from an overhead tank when this is feasible; the pressure-head should not be less than 16 ft.[1] In most cases, however, pressure is applied from an air-compressor or a reciprocating pump. At first *montejus* were mainly employed. In cyanide-plants they are being more and more replaced by belt-driven 2- or 3-throw plunger-pumps, as the cost of compressed air, the loss of air with every filling of the montejus, the waste caused by air-agitation while filling the montejus, make its use expensive. Thus in Western Australia[2] filling with compressed air costs from 25 to 40 per cent. more than working with pumps.

In filter-pressing, it is necessary to work slowly, especially at the start, as otherwise the filtrate is liable to become turbid. The time required for filling and emptying a press varies mainly with the character of the solid matter and the kind and size of the press. McNeill[3] gives the following data for slime, from cyaniding gold ore in Western Australia, with a 20-chamber press holding about 1.25 tons of dry slime in cakes 28 in. sq. and 3 in. thick: filling 15, washing 19, emptying 16, total 50 min., or say 1 hr. Reducing this to the general statement of press-capacity in tons of slime per ton slime treated per day, gives 1.25 : 30 = 0.042 ton. Yates[4] gives for two Johnson presses, 48 in. sq. with 24 chambers and cakes 3 in. thick, each holding 5 tons of slime and treating together 120 tons of slime (= 104 tons of dry slime) 0.080 ton. An average of six Western

[1] Langguth, Filtration of Gold precipitated with SO_2 and H_2S, *Tr. A. I. M. E.*, 1892, XXI, 314.

[2] *Eng. Min. J.*, 1904, LXXVII, 602.

[3] *Trans. Inst. Min. Met.*, 1897–98, VI, 247.

[4] *Proc. Chem. Met. Soc. So. Afr.*, 1902–03, III, 36.

TABLE 225.—FRAME-PRESSES OF T. SHRIVER & CO., HARRISON, N. J.

Size of press, inches	Number of chambers	Nominal filtering area in square feet	Thickness of cake, inches	Cubic capacity 1 inch cake	Working pressure per square inch, pounds
12	6	10	1 to 3	0 32	150
12	12	20	1 to 3	0 64	150
12	18	30	1 to 3	0 96	150
18	12	47	1 to 3	1 95	150
18	18	70	1 to 3	2.92	150
18	24	94	1 to 3	3 89	150
18	30	117	1 to 3	4 87	150
24	24	169	1 to 6	6.96	150
24	30	211	1 to 6	8 70	150
24	36	253	1 to 6	10.44	150
24	42	295	1 to 6	12.18	150
30	24	266	1 to 6	11.08	100
30	30	333	1 to 6	13 88	100
30	36	399	1 to 6	16.63	100
30	48	532	1 to 6	22.17	100
32	30	375	1 to 6	15 63	100
32	36	450	1 to 6	18.75	100
32	48	580	1 to 6	25.00	100
32	60	750	1 to 6	31.25	100
36	36	585	1 to 6	24 36	100
36	42	683	1 to 6	28.43	100
36	48	780	1 to 6	32.49	100
36	60	975	1 to 6	40.61	100
42	36	800	1 to 6	33.34	75
42	42	933	1 to 6	38.88	75
42	48	1,067	1 to 6	44.44	75
42	60	1,334	1 to 6	55.55	75

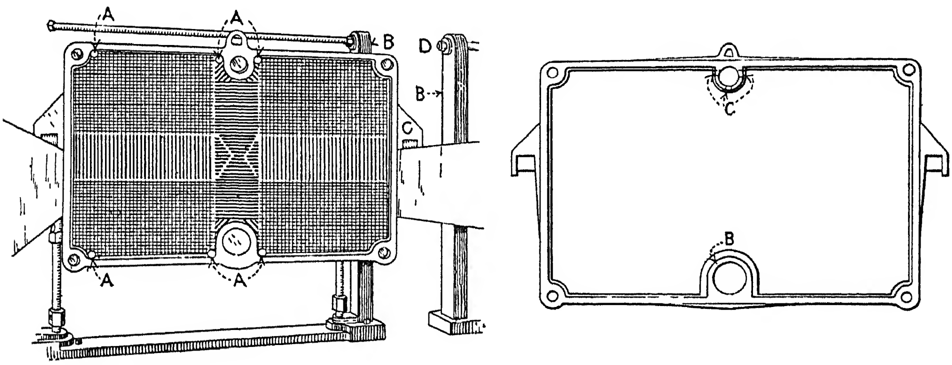
Australian mills[1] with 37 presses having a combined capacity of 122.4 tons of dry slime and treating in 24 hr. 833 tons, gives the figure of 0.149 ton.

The Merrill Press,[2] represented in Figs. 594 and 595, is of the frame type.

[1] Julian, H. F., and Smart, "Cyaniding Gold and Silver Ores," Lippincott, Philadelphia, 1907, p. 259.

[2] Merrill, *Tr. A. I. M. E.*, 1904, XXXIV, 585.
Bosqui, *Min. Sci. Press*, 1907, XCV, 21.
Ehle, *Mines and Minerals*, 1907, XXVII, 360.
Merrill, *Eng. Min. J.*, 1911, XCI, 721.
Clark-Sharwood, *Tr. Inst. Min. Met.*, 1912–13, XXII; *Min. Eng. World*, 1912, XXXVII, 1039, 1090, 1142, 1189; *Eng. Min. J.*, 1913, XCV, 161.

It has a very large capacity and is so constructed that the cake is sluiced out without the press being opened. There are 91 plates, Fig. 594, and 92 frames, Fig. 595, each 4 ft. high by 6 ft. long, and 4 in. thick. The original plates, shown in Fig. 594, had in the center two sets of grooves at right angles to one another, and in the rest of the panel pyramidal knobs with parallel base-lines. They showed lines of weakness and have been replaced by panels covered with staggered knobs. Of the 92 frames 6 have a top-feed and bottom-discharge; a few have two top-gates to permit examination of the chambers; the others are standard frames with no extra openings. The front and rear standards, 46 ft. apart, are connected by 15-in. channels which are strengthened in the center by adjustable posts, *B*, Fig. 594, and these are tied across the tops by rods, *D*, to prevent any sagging. The lugs, *C*, with which the plates and frames rest on the channels have roller-bearings. A plate, Fig. 594, has four 2 1/2-in.



FIGS. 594 and 595.—Merrill filter-press.

openings in the corners; the upper serve for the circulation of air, the lower for circulation of solution; there is also an upper central 4-in. opening for the entering pulp and a lower 6-in. central opening for the sluiced slime; lastly there are six small openings, *A*, which pass through the grooved panels; the two on the left are connected through cored ports with the left solution- and air-channels; in the two adjoining plates, the two side-holes are situated to the right and connected with the right channels, so that air and solution shall travel as indicated for solution alone in Fig. 588. The frame, Fig. 595, has openings similar to those of the plate. The upper central channel has three ports, *C*, leading into the chamber; the upper part of the lower central channel, *B*, is cored out for a width of 3 in. leaving on either side a rib 1/2 in. thick. Through its upper portion reaches a 3-in. sluicing-pipe provided with 92 nozzles 1 in. long and 5/32 in. diam. extending into the chamber; the rear end of the pipe rests in a seat of the movable head; the front end, connected with pressure-water, passes through a stuffing-box in the fixed head. The pipe is rotated mechanically by a rack and pinion through an arc of 120° that the water may reach every part of the

slime-cake in the chamber. The rotation is quick when the nozzles are vertical, slow when inclined.

One electrically driven rack and pinion suspended from an overhead track serves eight presses. Four parts water under a head of 50 lb. per sq. in. are required for one part of slime-cake. The sluiced slime enters the crescent-shaped space between the 3-in. sluice-pipe and the 6-in. delivery-channel *B*, and passes off through six openings in six bottom-discharge frames which are opened and closed together. The press is closed by means of two thrust-screws.

The press was developed at the Homestake mines. There the slime to be filtered has a consistency of 3 tons of water to 1 ton of dry slime. It arrives in a 10-in. pipe under a gravity-head of 35 lb. per sq. in., and enters the press through six 4-in. branch-pipes, the cocks of which are opened at the same time by one movement of a lever. The filtrate passes off through the lower openings and is collected in 5-in. pipes, while the chambers become filled with well compacted slime-cake. The cake is then leached with water containing 0.1 and 0.04 per cent. KCN, aërated and washed in 6 hr., and then sluiced. The lime required to neutralize the acidity of the pulp clogs the filters; it is removed at intervals by treatment in the press with HCl. The capacity of the press is 25 tons of dry slime. The time required for a complete treatment is 8 hr.: filling 1 hr., treating 6 hr., emptying 1 hr.; thus the capacity is 75 tons in 24 hr., and the press-capacity for each ton dry slime per day 25 : 75 = 0.33 ton, a great advance over the data given on page 725. The plant has (December, 1912) 28 presses. The cost of treating slime at the Homestake plant with cyanide is given in Table 226.

At the Esperanza mill, El Oro, Mexico,[1] two 90-frame presses are in operation. The 3-in. slime cakes are freed from KCN by central-washing. The wash-water, admitted through a central feed-channel, halves each cake as it flows both ways; 8 tons of water under a pressure of 65 lb. per sq. in. are necessary for washing and sluicing 1 ton of slime. The colloidal character of the slime made the usual method of washing unsatisfactory.

The Sweetland Filter-press,[2] Figs. 596 to 600, consists of a series of alternating upright frames, *A*, and plates, *B*, which are supported in the usual way by lugs, *n*, and clamped by jack-screws, *c*, acting upon two movable heads *d* and *e*. Both frames and plates are of cast-iron, the faces are machined and have on one side a rubber gasket, *f*, to form a water-tight joint; the lower parts are V-shaped in order to form pockets, *w*, which are to receive the sluiced slime; seven of the plates have outlets, *p*, for the slime. The upper part of the frame, *A*, is closed by two layers of cocoa matting, *q*, 30 in. sq., which are enveloped by a bag of 15-oz. duck filter-cloth. At the bottom of the mat is a flattened and perforated pipe which is connected by the union, *m*, to a nipple leading to the

[1] *Eng. Min. J.*, 1908, LXXXVI, 762.

[2] Sweetland, *Eng. Min. J.*, 1908, LXXXV, 359.

Wilson, *Mines and Minerals*, 1908, XXIX, 129.

Sweetland, *Min. World*, 1908, XXIX, 565; *Min. Sc. Press*, 1909, XCIX, 853; *Electrochem. Met. Ind.*, 1910, VIII, 52; *Met. Chem. Eng.*, 1912, X, 492.

channel, *k*. In the upper part of the frame, *B*, is placed the spray-pipe, *o*, to wash down the slime-cake into the hopper, *w*, whence it is discharged through the opening, *p*, into pipes, *u* and *v*. In the latest frames the mat has been replaced by a steel plate carrying on either side corrugated boards as seen in Fig. 599; the frame is then slipped into a filter-bag and the open end sewed up; the filter is prevented from bulging, when pressure is applied, by holding it down with vertical hardwood strips. The complete filter-leaf is shown in Fig. 600.

TABLE 226.—CYANIDE SLIME TREATMENT WITH MERRILL PRESS AT HOMESTAKE PLANT

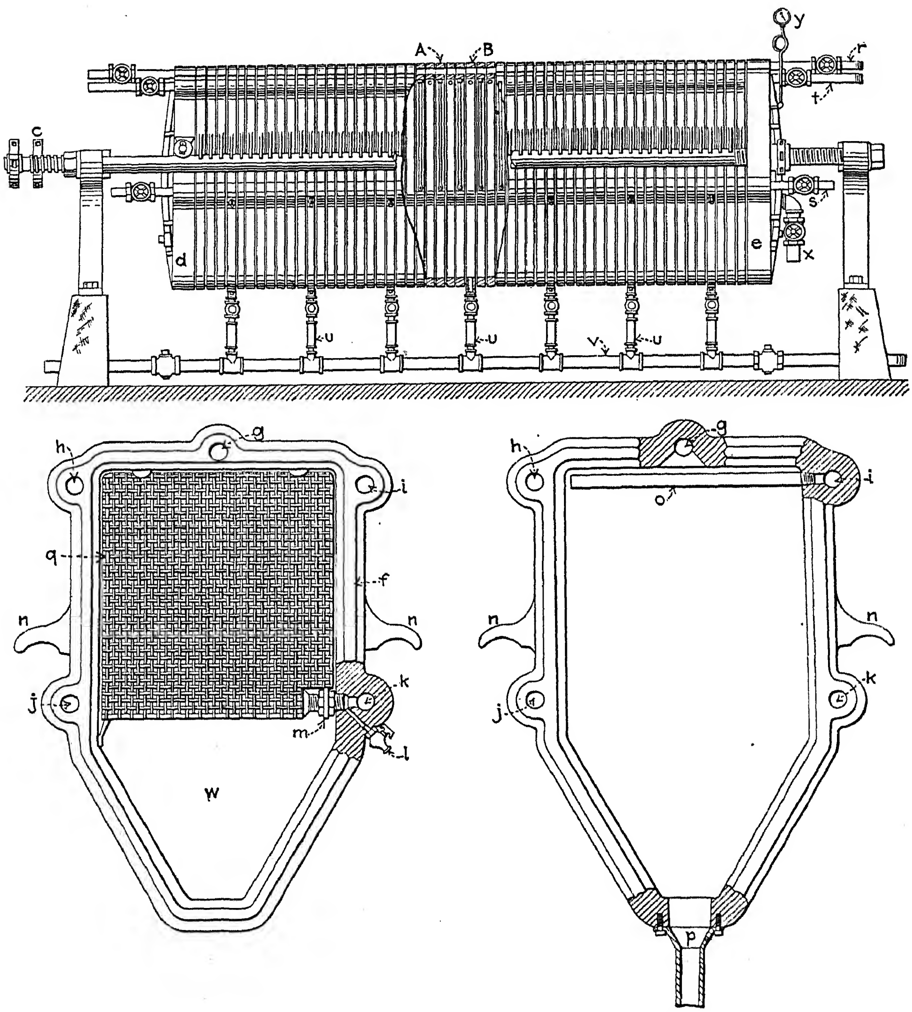
Operating Costs per Ton, March, 1908; Total Tons Treated, 49,946

Item	Labor	Electric power and lighting	Cost per ton			
			Chemicals		Other supplies	Total
			Item	Cost		
Thickening........	.00370	....	....	.....	.00006	.00376
Transportation............	....	....	....	....	.00030	.00030
Neutralization........	.00625	.00116	Lime 4.476 lb.	02236	.00006	.02985
Filling and discharging......	.01040	.00347	....	....	.00239	.01626
Dissolving and washing......	.02468	.01502	Cyanide .31 lb.	.06200	.00014	......
			HCl	.02324	.01 (Cloths)	.13508
Precipitation...........	.00281	...	Zinc .127 lb.	.00762	.00026 .00057 (Cloths)	.01126
Heating................	.00218	.....	....	......	.00537	.00755
Assay office............	.00348	....	....	......	.00518	.00866
Superintendence............	.00911	......	....	....	.00043	.00954
Miscellaneous...........	.01272	....	....	......	.00117	.01389
Fire protection...........	....	....	....	.....	00054	.00054
Refining, bullion, expressage and mint charges.	......	....	....	....	.00864	.00864
Total....................	.07533	.01965	....	.11524	.03511	.24533

General notes:

Hydrochloric acid @ \$4.30 per carboy—10 carboys per press.
Cyanide @ .20 per lb.
Lime @ .005 per lb.
Zinc @ .06 per lb.
Labor @ 3.00+ per 8-hour shift.
Power @ 7.50 per mechanical horse-power per month.
One suit of filter-cloths lasts one year—for 24 presses cloth consumption = 2 suits per month, = \$0.01 per ton of slime treated.

The mode of operating is as follows: the slime is agitated with strong KCN solution, drawn into a montejus and forced by compressed air through the press The air-pressure, at first 30 lb. per sq. in., rises to 60 lb., when the slime-cake ha become about 2 in. thick. The pulp-supply is now shut off, and a vacuum o about 5 in. is created inside the filters to drain off the solution. The stron



FIGS. 596, 597 and 598.—Sweetland filter-press.

solution is followed by a weak one, and this by wash-water. Now air, steam or water is admitted back of the filter-cloths to loosen the cakes which are then sluiced out by the spray-pipe, *o*, the slime retaining about 40 per cent. water. Supposing the slime to be of such a character that dry slime covering 1 sq. ft. filtering area to the thickness of 1 in. weighs 6 lb., a press of 500 sq. ft. filtering

area covered to a thickness of 1 1/2 in., will hold $500 \times 6 \times 1.5 = 4{,}500$ lb. dry slime. With a complete cycle of operations of 1 hr., the capacity for each ton of dry slime treated per day is 0.075 ton. The laws of the flow of liquids in filter cakes have been studied by Almy-Lewis.[1]

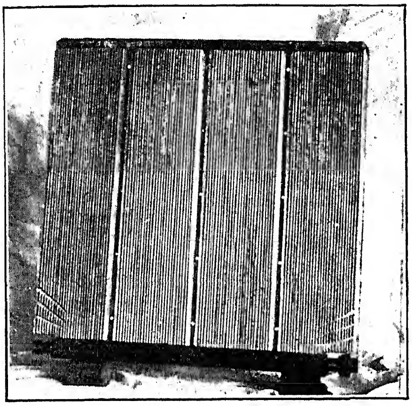


FIG. 599.—Sweetland filter-press.

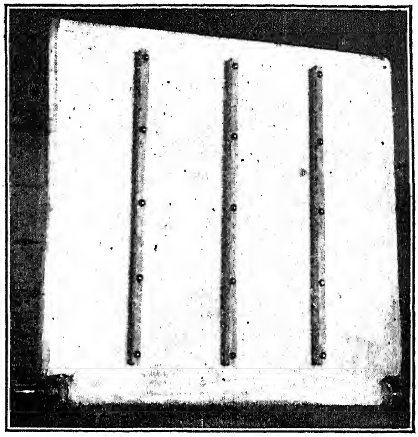


FIG. 600.—Sweetland filter-press.

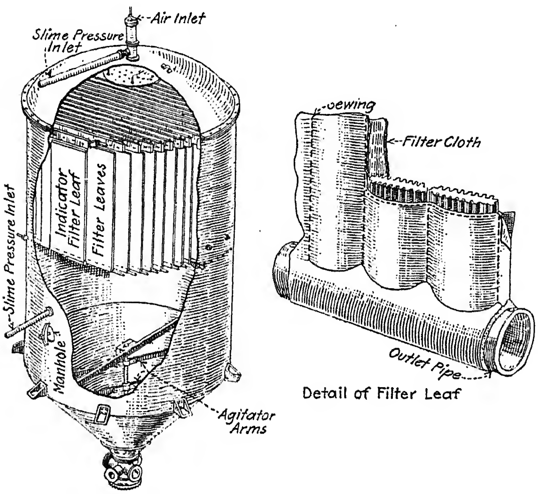
307. **Pressure-filters.**—Since the filtration of slime on a large scale has become such an important part of the cyanide-treatment of gold- and silver-ores, several filters have been constructed in which filtering is accomplished by forcing slime-pulp into a tank holding a number of filter-leaves, and the necessary cleansing by withdrawing the excess pulp and replacing it by dilute solutions or wash-water or both.

The Blaisdell pressure-filter[2] is shown in Fig. 601, and details of the filter-leaf are shown in Fig. 602. In the cylindrical closed tank are suspended 144 filter-leaves with 10 sq. ft. area; each leaf has a number of vertical wooden separating-strips which are ribbed and form drainage-channels; it also encloses at the lower end an outlet-pipe for the filtrate; these pipes are connected outside of the tank with a main delivery-pipe. Slime, having the consistency of 3.5 to 4.5 parts liquid to 1 dry slime, is agitated in solution tanks, discharged into a slime-tank, drawn from this with a centrifugal pump and forced into the filtering-tank with a pressure of from 25 to 50 lb. either through the lower or the upper inlets. The agitator-arms are rotated at the same time to prevent the slime from settling, while the filtrate passes off through the outlet-pipes. The outlet-pipe of the indicator-leaf is connected with a weighing-machine which rings a bell as soon as the cake has attained a pre-determined thickness,

[1] *J. Ind. Eng. Chem.*, 1912, iv, 528; *Eng. Min. J.*, 1912, XCIV, 161.

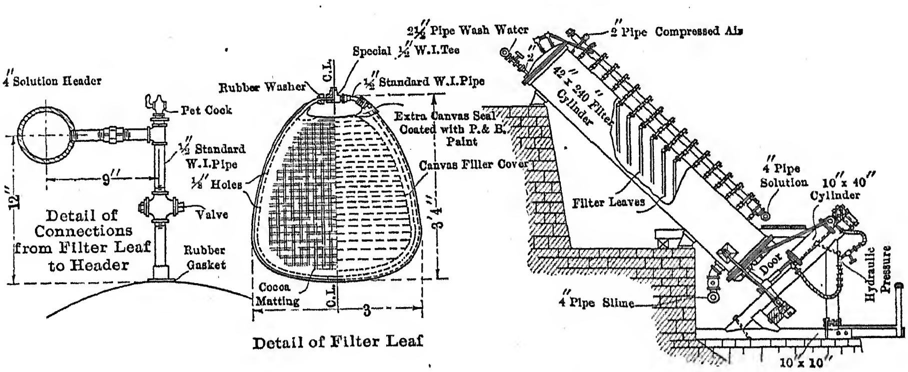
[2] Editor, *Eng. Min. J.*, 1907, LXXXIV, 446.
Editor, *Min. Sc. Press*, 1907, XCVIII, 188.
Wilson, *Mines and Minerals*, 1908, XXIX, 130.
Fulton, *Min. Ind.*, 1907, XVI, 540.

varying from 0.75 to 1.50 in. When the bell rings, the pump is stopped, and the slime-pulp in the tank returned to the slime-sump through one of the holes in the apex of the cone. At the same time the air-inlet valve opens to admit the necessary air. The main which joins the outlet-pipes is connected with a vacuum-pump to hold the slime-cake on the filters and to drain off the gold-solution. Weak KCN-solution is now pumped into the tank through one of the openings in the apex to replace the gold-solution in the cakes; the filtrate as well as the excess-solution in the tank go into a cyanide-wash sump. Clear water finally replaces in the same manner the weak KCN-solution. When the slime-cakes have been washed and the tank is still filled with water, pressure is turned into the outlet-pipes to dislodge the cakes from the leaves that they may be flushed out through the bottom. With ordinary slime and 35 lb. pressure, the cycle of operations requires 2 hr.



FIGS. 601 and 602.—Blaisdell pressure-filter.

The Burt pressure-filter[1] is shown in Figs. 603–605. The filter consists of a steel-plate cylinder, 5/16-in. thick, set at an angle of 45° which is closed



FIGS. 603 to 605.—Burt pressure-filter.

by cast-iron heads bolted to the ends; the head at the lower end has a door hinged horizontally which is opened and closed by means of a toggle-mechanism

[1] Forbes, *Eng. Min. J.*, 1908, LXXXVI, 458.
Rice, *op. cit.*, 1909, LXXXVII, 685.
Burt, *Min. Sc. Press*, 1907, XCV, 717.

operated by a hydraulic piston with a pressure of 100 lb. per sq. in.; a tight joint is obtained by a groove packed with rubber and a projecting ring in the rim of the door. From the top of the cylinder are suspended 18 pear-shaped filter-leaves, each with 135 sq. ft. effective filter-surface. The frame of a leaf is made up of two lengths of 1/2-in. wrought-iron pipe connected at the bottom by a 4-in. piece of 1/2-in. rod, and at the top by a swivel T-joint; on the inside are 1/8-in. holes from 2 to 3 in. apart for the passage of the filtrate; cocoa-matting fills the space enclosed by the pipe-frame; the whole is covered with a heavy canvas filter-cloth lapped over and sewed together by a machine. The T-joints of the leaves are connected by branches with a filtrate delivery-pipe; each leaf has a cock to permit cutting it out when necessary, and a pet-cock to test the filtrate.

The mode of operating is as follows: the cylinder is filled with slime-pulp (80 per cent. water) at the lower end from an overhead tank with a gravity-head of 40 to 60 lb. pressure per sq. in., or by means of a centrifugal pump. The filtrate passes off through the delivery-main to a solution-tank; after about 9 min. the excess-slime is drawn off into a holding-tank and its discharge assisted by air under a low pressure entering at the top, which at the same time clears the slime-cakes from solution. The latter operation takes about 8 min.; wash-water is then introduced and handled as was the slime-pulp. Lastly compressed air is admitted to the filter-leaves which dislodges the slime-cakes so that they drop and slide down and out through the discharge-door. A stream of water washes the cylinder in a few seconds. The excess slime in the holding-tank is now delivered into a centrifugal pump and the additional amount necessary to make up a full charge supplied from the main slime-tank. The whole operation from start to finish, without washing, takes 20 min.; with washing 35 to 40 min. The cakes are 2 1/2 to 3 in. thick and hold 30 per cent. water.

At El Oro, Mexico, a battery of five cylinders is operated as a unit. The filters have an effective filtering area of 1,170 sq. ft. and treat without washing from 600 to 650 tons of dry slime in 24 hr., which gives them a capacity of 0.53 ton per sq. ft. of filter-surface in 24 hr.; the cost of treatment is 1.5 cents per ton dry slime.

A Burt revolving filter[1] has recently been brought into the market.

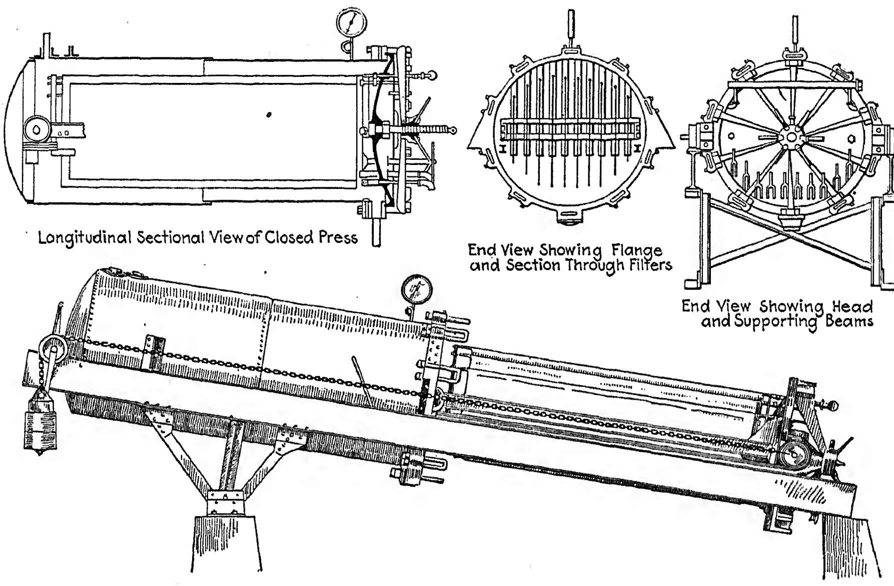
The Kelly pressure-filter,[2] Figs. 600 to 609, consists of an elevated, inclined, fixed, cylindrical tank of boiler-iron, 9 ft. 6 in. long by 4 ft. diam., which is closed permanently at the rear by boiler-iron, and temporarily at the front by a cast-iron head; the latter forms part of a truck which being balanced by two counter-weights is easily moved up and down a track in line with and in front of the cylinder. The mouth of the cylinder has a grooved flange with rubber gasket and the head a projecting ring. Along the inside walls of the tank is a track for the travel of a second truck which supports the back end of the filter

[1] *Mines and Minerals*, 1911, XXXII, 234.

[2] Kelly, *West. Chem. Met.*, 1907, III, 152.
Wilson, *Mines and Minerals*, 1908, XXIX, 132.
Lass, *Tr. A. I. M. E.*, 1911, XLII, 808.

carrier-frame while the front is connected with the closing head. Thus the frame can be pushed telescope-fashion in and out of the cylinder. The angle of the track and the balancing weights are so chosen that the frame loaded with slime will move downward, while it will move upward when empty. In order to obtain a tight joint, the head is pressed on to the flanged tank by a number of radial levers, the outer ends of which pass through lugs bolted to the flange, while the inner ends are pivoted to a movable collar which is clamped by a central hand-wheel.

The filter consists of 10 oblong frames of double-crimped wire-cloth enveloped by bags which are placed 4 in. apart and have a filtering surface of 410 sq. ft.



FIGS. 606 to 609.—Kelly pressure-filter.

The upper pipes carrying the filter-frames project at both ends. The front ends are coupled to the head and the rear ends rest on the back part of the carrier-frame. The lower pipes pass through the head and discharge the filtrate into a launder. There are the usual pipes for slime, wash-water, pressure-air, or steam, etc.

The mode of operating is as follows: the slime is drawn by means of a centrifugal pump from the supply-tank, in which it is kept in suspension by an agitator, and forced near the head into the closed cylinder until the overflow-pipe, also near the head, indicates that the cylinder is full, when the overflow-pipe is closed, but pumping continued. The solution passes off through the filters and slime-cakes form upon them until they have become about 1 1/2 in. thick, when the gauge at the top of the cylinder registers a pressure of about 50 lb. per sq. in. The pump is now stopped; the swinging-pipe, which is attached to the

discharge near the lower end of the cylinder, revolved until its end is over the slime-tank; the valve is opened, and air under a pressure of 2 to 3 lb. admitted at the top of the cylinder to hasten the discharge of the excess pulp. When empty, a weak KCN-solution is pumped into the cylinder to replace the strong solution in the slime-cakes; this requires a pressure of 30 to 60 lb. and considerable time. The filtrate and excess weak solution are collected in a solution-tank; the filter-cylinder is finally filled with wash-water, and the excess returned to the wash-water tank. A vacuum may be used to assist the replacement of the liquids in the cakes, also for holding in place the cakes in the next step, when the frame with the charged filters is withdrawn from the cylinder to a position over a receiving-hopper for leached slime. Lastly, a flexible connection for low-pressure steam, or water, or both is made to dislodge the cakes which drop into the hoppers (air cannot be used, as the filters are not braced). The filter-leaves are cleaned with water from a hose, and the frame is returned to the cylinder for a new charge.

At the Black Pearl mill, Idaho, where 1 cu. ft. dry slime weighs 128 lb., the press treated a charge of pulp (3.5 water : 1 dry slime) in 1 hr. 6 min.; at the Vindicator Concentrated Gold Mining Co.'s plant, Cripple Creek, Colo., a cylinder with eight filters 5 1/2 in. apart treated a charge of 2 tons roasted telluride ore with 60 per cent. water in 1 hr. One of the latest presses is now in use at the Treadwell Mines, Alaska.

308. Suction-filters.[1]—The original form of this class of filters, the vacuum-filter of industrial chemistry, is an air-tight vessel with perforated false bottom beneath which a vacuum is produced by a pump, an ejector or some other suction device. In metallurgical plants, the ordinary gravity-filtration of mixed sand and slime is sometimes accelerated in this manner as, *e.g.*, in the leaching of chloridized silver ore; slimes are rarely treated by this method of leaching. However, Drucker[2] applied it in his vacuum filter which he uses for cyaniding the pulp from a gold stamp-mill. The battery-pulp, with seven parts water : one dry pulp, is thickened in a conical spitzkasten to three parts water: one dry pulp and collected in a settler (15 ft. in diameter, 8 ft. in depth) having a false bottom covered with burlap and filter-cloth. Above this cloth is a muller with 26 cast-iron shoes that can be raised and lowered 2.5 ft. Milk of lime (8 lb. lime : 1 ton ore) is added to the 12- to 15-ton charge; the muller is stopped and raised; the pulp is settled for 5 hr., and the solution decanted, the pulp retaining about 40 per cent. water. The muller is now lowered to just above the perforated 1-in. boards, which protect the filter, rotated at the rate of 18 r.p.m., and vacuum-filtering started, which reduces the water-content of the pulp to 18 or 20 per cent. This thickened pulp is now ready for cyanide-treatment. It is stirred with KCN-solution, aërated from beneath the filter, settled as was the battery-pulp, etc. The decanted gold-solution is passed through sand

[1] Nichols, *Min. Sc. Press*, 1910, C, 395.
Young, *Tr. A. I. M. E.*, 1911, XLII, 752; *Eng. Min. J.*, 1911, XCII, 885; *Min. Sc. Press*, 1911, CIII, 552.

[2] *Min. Sc.* Press, 1908, XCVII, 458.

clarifying-vats before it enters the zinc-boxes. The 48 hr. required to treat a charge of 12 to 15 tons of dry pulp are distributed as follows: filling 12 hr.; extraction of water from pulp 7 (settling and decanting 5, vacuum-filtering 2); cyanide agitation and aëration 10; extraction with strong (0.20 per cent. KCN-) solution 6 (s.–d. 4, v.–f. 2); with weak (0.05 per cent. KCN-) solution 6 (s.–d. 4, v.–f. 2); discharging and sampling 1 hr. A similar manner of working has been suggested by Merrall.[1] Spilsbury[2] uses a diaphragm of silica sponge as a filtering medium. Plates, 12×12 or 12×20 in., secured to channel-iron bars, by 0.25-in. carriage bolts, form a false bottom; oakum is driven into the joints and these are made completely tight by means of liquid cement. Pulp with 1.5 to 2.0 parts solution to 1 part dry ore is agitated with air under a pressure of 2 to 5 lb.; the capacity of the sponge is 5.0 to 5.5 cu. ft. air per min. per sq. ft. area at 1 lb. pressure. After agitation is completed, compressed air is replaced by suction, and the solution filtered.

The fundamental features of the basket type of filters are those devised in 1902[3] by G. Moore, viz., the filtration of pulp by suction through an immersed filter-covered frame with resulting adhesion of cake on the outside of the filter, and subsequent displacement of slime cake by use of pressure inside the filter. In the practical adoption of these two principles the charged frame may either be removed from the unfiltered pulp and transferred to other vats filled with solvent or wash-water to be further treated, or the frame may be kept stationary and the unfiltered pulp replaced by solvents or wash-water; in both cases the treated cake is finally loosened from the filter by applying air pressure inside the filter-bag instead of suction, and then discharged.

Modern suction-filters[4] have been developed into four classes of apparatus: (1) the basket type (Moore, Butters); (2) the horizontal table type (Ridgway, Hunt, Ogle, Scherr, Hendry, Allen, Parrish, Rothwell, Grothe-Carter); (3) the drum type (Oliver, Portland); and (4) the wheel type (Just, Singer, Barnes, Holland).

The Moore suction-filter,[5] consists of a number of rectangular canvas-covered frames, Figs. 610 and 611, clamped together to form a basket which is suspended, Fig. 612, from an overhead traveling crane. The basket is moved over a series of tanks by one electric motor, and lowered into and lifted out of

[1] *Min. Rep.*, 1907, LV, 291.

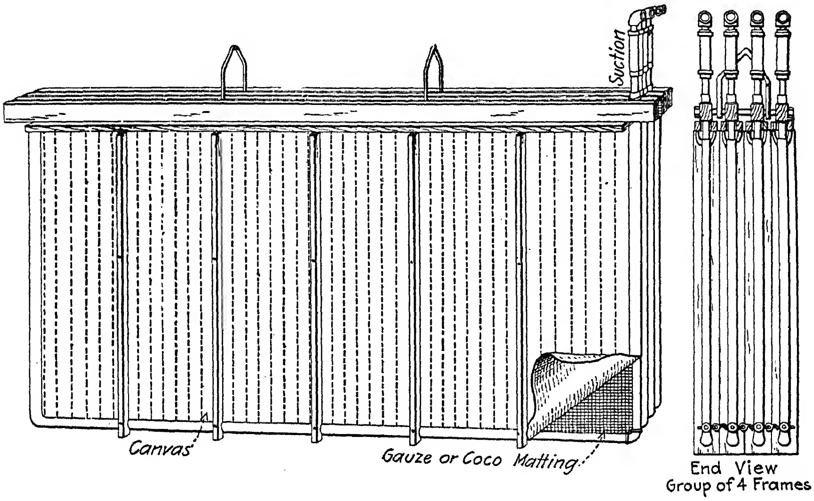
[2] *Tr. A. I. M. E.*, 1910, XLI, 367.

[3] According to Foote (*Min. Sc. Press*, 1908, XCVI, 159), E. L. Oliver and A. M. Nicholas developed the same features independently at a later date. Decision of U. S. Circuit Court of Appeals for the Moore Filter Co., *Min. Sc. Press*, 1912, CV, 663; *Eng. Min. J.*, 1912, XCIV, 923; *Met. Chem. Eng.*, 1912, X, 791.

[4] *Met. Chem. Eng.*, 1912, X, 708.

[5] Fulton, *Tr. A. I. M. E.*, 1905, XXXV, 610.
Brown, *Min. Sc. Press*, 1906, XCIII, 292.
Nutter, *loc. cit.*, p. 714.
Kirby, *op. cit.*, 1907, XCV, 46.
Tracy, *Eng. Min. J.*, 1906, LXXXII, 149.
Shaw, *op. cit.*, 1909, LXXXVII, 487.
Nichols *Min. Sc. Press.*, 1910, C, 495.

the series by another. In cyaniding, the basket is submerged in the filter-tank, Fig. 613, in which slime-pulp, that has been treated with KCN- solution, is kept



FIGS. 610 and 611.—Moore suction filter.

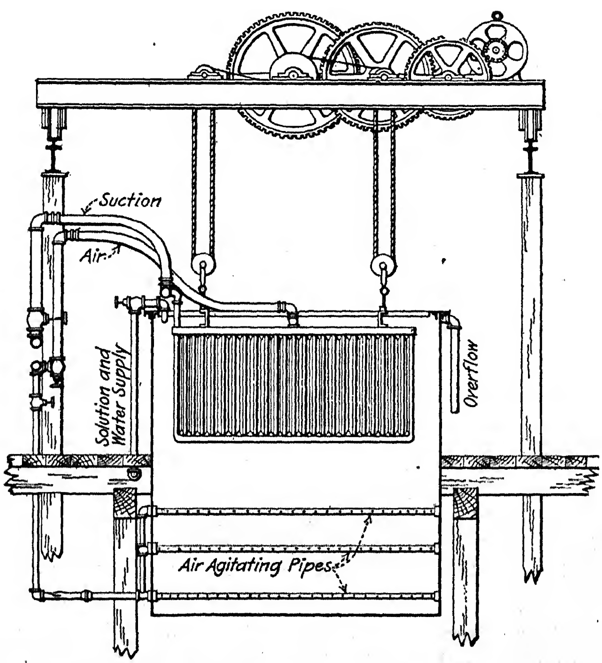


FIG. 612.—Moore suction filter.

in suspension by agitating with compressed air, Fig. 612, or some mechanical stirrer, or by withdrawing at the bottom and feeding at the top with a centrifugal

or an air-lift pump. The suction is applied through a common suction-pipe connected with the frames, and solution drawn through the filter-cloths[1] until a cake about 3/4 in. thick has been formed on both sides. The basket, charged with slime-cake and still under vacuum, is raised and transferred, Fig. 613, to the barren-solution tank and then to the wash-water tank, in each of which suction-filtering is repeated; lastly it is lowered into the tailings-discharge tank where the cakes are dislodged by applying air pressure. As moistening a slime-cake assists its loosening from the filter, it is often desirable to force some water through before the compressed air. For this purpose, a separate perforated pipe is introduced near the top inside of each frame. In places where there is an abundance of water, the compressed air may be turned into the filters after the washing is finished and the filters are still immersed in the wash-water tank; the tailings will then go to waste with the excess wash-water. Filtering to form a cake about 3/4 in. thick takes from 40 to 120 min.; suction in the barren-solution tank about 25 min., and in the wash-water tank 15 min.

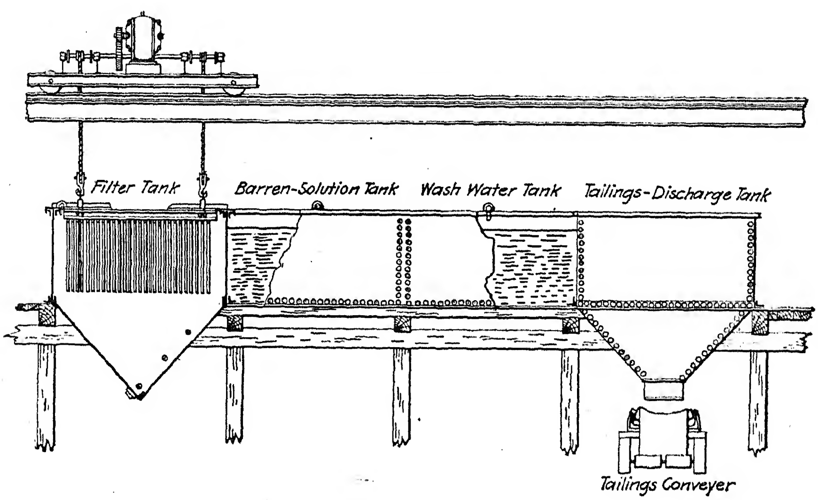


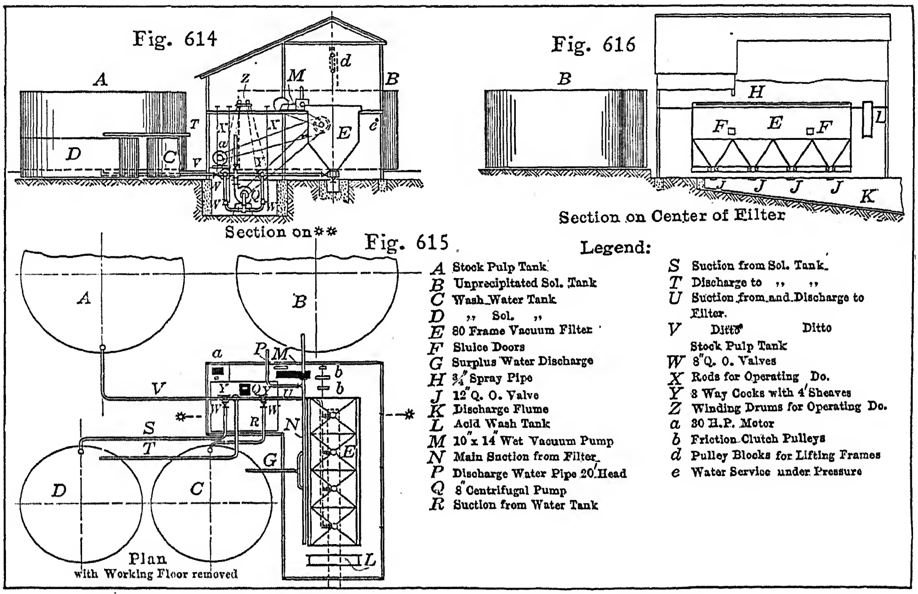
FIG. 613.—Moore suction filter.

Frames are commonly 6 by 12 ft. in size, and usually 24 frames with 6-in. centers are grouped in a basket, which gives a filtering-area of 3,456 sq. ft. The frame, as shown in Figs. 610 and 611, consists of a wooden top-piece, 2.5 in. wide by 4.5 in. high, and a U-shaped 3/4-in. pipe; the horizontal arm has 3/16-in. perforations for the passage of the liquid and one of the vertical arms extends through the wood to be connected with the suction. The enclosed space was formerly filled with cocoa matting, as seen in Fig. 610; this has been replaced by vertical separating wooden strips, 5/16 in. square and spaced 1.5 in apart; 16-oz. cotton duck, which forms the filter-cloth, is sewed between the strips and reinforced by extra stitching near the top and bottom. A two-motor traveling crane, serving three tanks, 14 by 14 ft. and 12 ft. deep,

[1] Wear of cloths, *Eng. Min. J.*, 1910, LXXXIX, 762.

of 3/16-steel with conical bottoms, has a span of 21 ft. between centers of track, a lifting-capacity of 20 tons and a lifting-height of 8 ft., the rate of lifting is 10 ft. per min., and the rate of travel 30 ft. per min. A plant with the above three tanks and two baskets will treat from 100 to 125 tons of dry slime in 24 hr., presupposing the slime-cakes to be 0.75 in. thick, and 7 tons of dry slime to be handled in each cycle.

The Butters-Cassell filter.[1] The filtering plant shown in Figs. 614 to 616 is a combination of the Moore filter, of the Cassell plan[2] of leaving the filters stationary and exchanging the liquids, and of the Butters general arrangement



FIGS. 614 to 616.—80-frame Butters-Cassell vacuum filter plant.

of parts. The filter-frames are approximately 5 by 7 ft. and are suspended at about 4-in. centers. The main changes from the Moore filter are that the frame has a cocoa mat; that the filter-cloths are stitched through with vertical seams 2 in. apart and have one horizontal seam near the top, and six or seven vertical strips of wood are placed outside of a leaf and screwed together through the filtering material.

In operating, the leaves are suspended in a rectangular vat, *E*, which has pointed pockets for the discharge of the cakes. The pulp (2 water : 1 dry slime) is drawn from the stock pulp-tank, *A*, and pumped into vat, *E*, near the bot-

[1] Pamphlet by C. Butters, 1906.
Hamilton, *Min. Sc. Press*, 1907, XCIV, 785, 818.
Lamb, *Tr. A. I. M. E.*, 1907, XXXVIII, 200.
Patterson, *Min. Sc. Press*, 1909, XCVIII, 428 (Pachuca).
Smith, *op. cit.*, 1909, XCIX, 65.

[2] Suggested to Cassell by Moore in 1903.

tom until the frames are submerged. The vacuum is applied through pipe, *N*, by means of pump, *M*, and the gold-solution drawn off into the gold- or unprecipitated-solution tank, *B*, the head of the pulp being kept constant by running the supply-pump on and off. When a cake 0.75 to 1 in. thick has been formed in from 15 to 60 min., a vacuum of 6 to 9 in. is kept in the frame, and the excess pulp drawn off and returned to the pulp-tank. Any light adhering slime is removed by turning on a spray of solution. The filtration is repeated with dilute KCN-solution, *D*, of which two to three times the amount held by the cake is drawn through it, and lastly with wash-water, *C*. The cake is now discharged from the filter-cloths into the pointed pockets of *E*, by introducing water through pipe *P*, under a pressure-head of 30 ft., into the interior of the leaves; the slime is removed from the pockets by sluicing through quick-opening valves, *J*, into the discharge-flume, *K*. The time required for a complete operation is about 70 min.

The interesting feature of the Butters plant is that all transfers of solutions in connection with filtering are effected by a single 8-in. centrifugal pump, *R*, with valves and levers so arranged that one man can manipulate them from one position.

In comparing the Moore and Butters-Cassell plants, Kirby[1] finds that the operating costs, maintenance and repairs, of either system are about equal; that the installation-cost of the Moore system is 35 per cent. greater than that of the Butters-Cassell for an equal filtering area; that the capacity is 50 per cent. greater, and that the efficiency for the recovery of gold-bearing solution with the least amount of loss and dilution is higher, as the losses with the Butters-Cassell system due to the mixing of residues in pipes and vats are absent with Moore. Other comparisons[2] have been made by Bosqui, Nutter, Lamb Pettis and Frier.

The Ridgway suction-filter,[3] Figs. 617 and 618, is the best-known representation of the horizontal-table type. It is probably the first continuous-acting moving filter, and consists of a series of 13 suspended filter-leaves traveling in a horizontal plane through an annular trough divided radially into three compartments; two of these hold pulp (slimes-wash) to be filtered and wash-water (water-wash), while the third, the smallest, has no bottom and serves as discharge. The arms of the framework holding the filter-leaves are pivoted near the center in a hollow revolving shaft, and supported at the opposite ends by 4-in. wheels which travel on a circular track, 12 ft. in diam. The hollow shaft is driven from below by a worm gear; it has a horizontal partition above the framework which divides it into two parts. The upper part is enlarged into a chamber for compressed air and has 13 openings that are supplied with

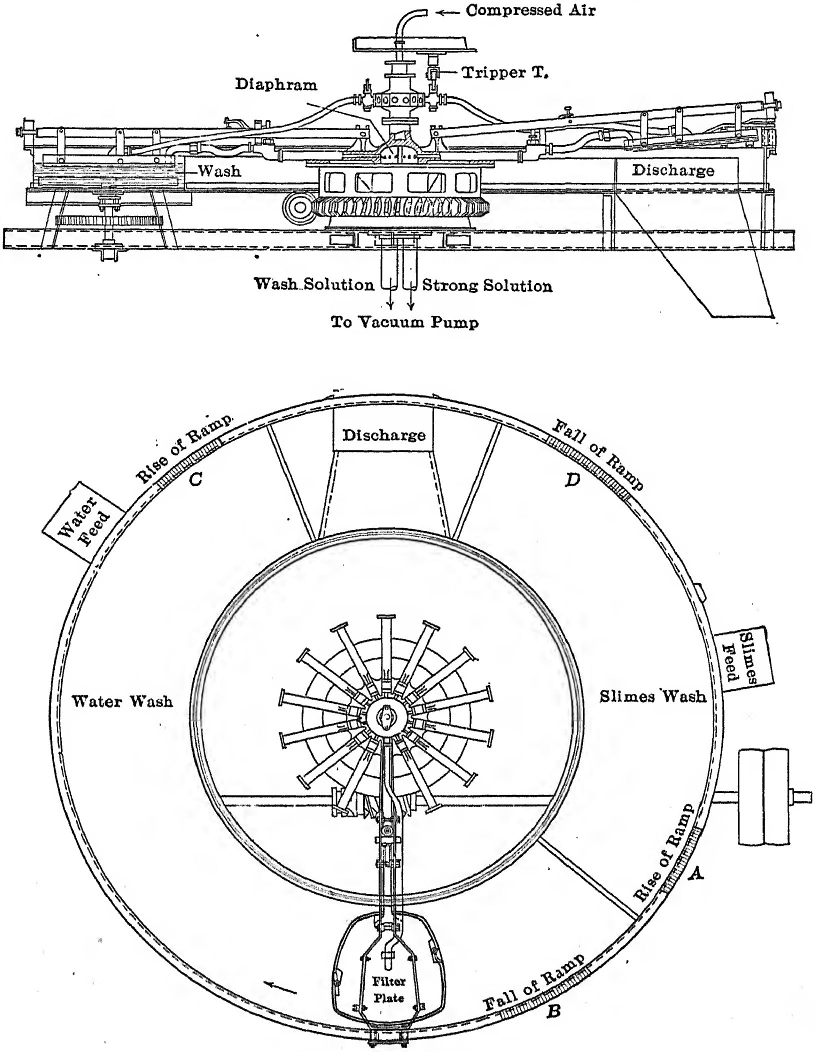
[1] *Min. Sc. Press*, 1907, xcv, 46.

[2] *Op. cit.*, 1907, xciv, 148 (Bosqui), 210, 491 (Nutter); 152, 400 (Lamb); 367 (Pettis); 432 (Frier).

[3] Fulton, *Min. Ind.*, 1907, xvi, 540.

Rice, *Eng. Min. J.*, 1908, lxxxvi, 120; *Min. Sc. Press.*, 1907, xciv, 181; 1908, xcvi, 769 (Lamb); 1909, xcviii, 51 (James).

mechanically-operated cocks; the lower part is divided into two equal compartments by a stationary vertical diaphragm; 13 hollow arms connected with the filter-leaves end in the chamber. The two compartments are joined to the vacuum-pump and carry off the strong and weak solutions. Each filter-leaf



FIGS. 617 and 618.—Ridgway suction filter.

has a filtering area of about 4 sq. ft. It consists of a cast-iron frame carrying a thin wooden box with a perforated bottom over which is stretched the filter-cloth, while the top is connected with the vacuum- and the compressed-air pipes, which have cocks that are opened and closed automatically. The supply of pulp (slimes-wash) and of wash-water is regulated by floats (not

shown); the pulp is prevented from settling by agitators rotated from below the trough.

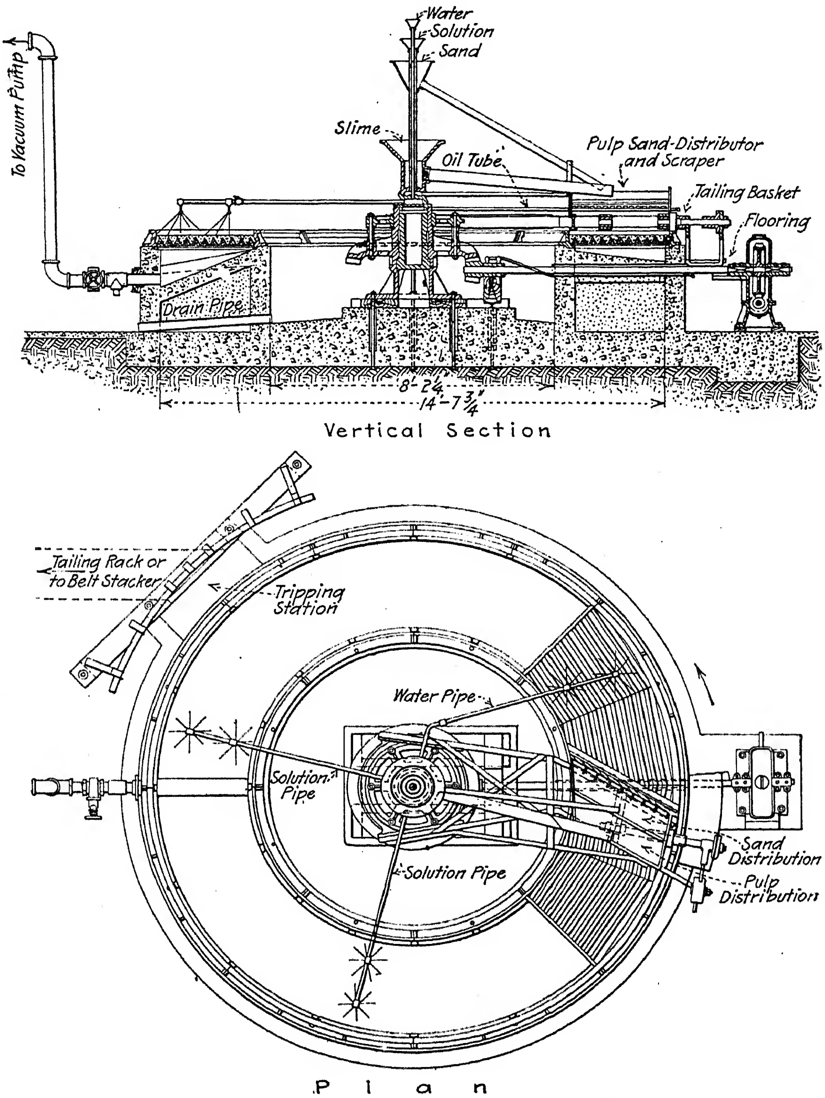
In operating, a leaf immersed in the pulp (1 water : 1 slime) with the vacuum-pipe open, travels through the slimes-wash compartment and collects on the lower side, with fairly sandy material, a cake 3/8 in. thick. As it approaches the end of the compartment it raises the leaf out of the pulp by climbing the rising track at *A*; the suction continues for one second to suck the solution from the filter and is then shut off by a tripper. The track now descends toward *B*, and the leaf following it is immersed in the wash-water, when the suction begins again. It now passes through the wash-water, rises over the second elevation, *C*, and is sucked dry; the suction is stopped and the compressed air turned on to discharge the cake which retains about 30 per cent. water. The bared leaf now descends again at *D*, and re-enters the slimes-wash compartment. A leaf is 13 sec. in pulp, 7 sec. on the elevation between pulp- and wash-water compartments, 30 sec. in wash-water, 7 sec. on the elevation between wash-water and discharge-compartments, and 3 sec. in connection with the compressed air over the discharge. The machine makes 1 r.p.m. and discharges 13 cakes per min.; it treats from 25 to 50 tons of slime in 24 hr., the amount varying with the clayey or sandy character of the pulp. A filter lasts 14 days with roasted slime. The machine requires 1/2 h.p. for rotation; the vacuum-pump with a piston-displacement of 80 cu. ft. per min. and a vacuum of 3.5 in., 3 1/2 h.p.; the agitators, etc., about 1 h.p.; total, 5 h.p.

The Hunt suction-filter,[1] shown in Figs. 619 and 620, is probably the first stationary filter which works continuously. It consists essentially of three parts: a stationary annular trough which is closed at the top by a filtering surface and connected near the bottom with a wet vacuum-pump; a central rotating cylinder to which are attached a pulp-distributor which spreads first sand and then slime in thin layers over the filter; two solution and one wash-water pipes, a scraper to remove the washed ore into a tailing-basket, and a tripper which dumps the tailing onto a belt-stacker. The operations of feeding, filtering and washing follow one another with the rotation of the cylinder.

The base, 22 ft. 4 1/2 in. in diam., and the annular trough, 3 ft. 9 3/4 in. in diam., which forms the vacuum chamber, are both of concrete; the bottom of the chamber slopes outwardly so as to drain to the outlet. The filter, 3 ft. wide, is a sand-filter; it has 16 divisions, the bottom of each is made up of triangular slats with bases 3/16 to 1/4 in. apart which are covered with graded sand. The spaces between the slats are filled with gravel of uniform size and just large enough not to fall through; over this is spread first a 1-in. bed of coarse sand, and then second a 1-in. layer of 8- to 12-mesh sand; the sides of the filter are enclosed by the concrete walls which carry an annular track for the traveling carriage which bears the distributor and scraper. In the center of the machine, 15 ft. 9 3/4 in. diam., is a stationary column with a collar which carries, on ball-bearings, a cylinder with bevel-gear driven at the rate of 1 r.p.m. by a horizontal shaft carrying at the opposite end a worm-gear, the worm of which

[1] Hunt, *Min. Sc. Press*, 1908, XCVII, 430; *Circular*, Hunt Filter Co., San Francisco, Cal.

runs in oil. The cylinder carries four superimposed concentric hoppers to receive water, solution, sand and slime. These are delivered on to the filter through radial pipes. The sand- and slime-pipes deliver their materials on to distributors, similar to those of the Frue vanner, Figs. 456 to 458, which



Figs. 619 and 620.—Hunt suction filter.

with the scraper are attached to the carriage. The pulp before reaching the machine is roughly classified into clean sand and sandy slime. The sand-distributor is placed in front of the slime-distributor, and the scraper in front of the former. The permanent filter-bed is thus first covered with a thin layer of clean

sand and then with sandy slime. The scraper, of the same width as the filter, is set at an angle of 60 deg. to the radius in order that it may discharge the residue into the tailing-bucket which is also connected with the carriage.

When the machine is in operation, the central cylinder rotates in the direction of the arrow; the scraper removes the filtered and washed ore into the tailing-bucket; back of it sand is spread over the permanent filter to be followed by sandy slime; the vacuum draws off the liquid, the residue is sprayed consecutively from two solution-pipes, and lastly from the wash-water pipe; the washed residue is now ready for the scraper; whenever the tailing-bucket arrives at the tripping station, it is overturned and its content discharged on to the belt-stacker. If water is abundant, the tailing-bucket is replaced by a tail-race which surrounds the machine. The entire filtrate (strong and weak solution and wash-water) goes from the vacuum-chamber to the pump. If it is desired to keep the strong and weak solutions separate, the chamber may be divided into sections; it is, however, preferable to have two machines working in tandem, as this does away with mechanically operated valves and the necessary attendance; the first machine will take the strong solutions and the second the wash-solutions. With the modern practice in the cyanide process of sliming and leaching by agitation, it is common to use one strength of solution, to displace the solution charged with precious metal with barren solution of the same strength, and to add at the end of a cycle just sufficient wash-water to displace the barren solvent.

The machine has a filtering surface of 113 sq. ft. If the filtered layer is 1/4 in. thick, the surface will carry 2.26 cu. ft. material; supposing this material to contain 50 per cent. water and to weigh 109 lb. per cu. ft., its total weight will be 246 lb. wet or 123 lb. dry. With the machine making 1 r.p.m., the output will be 3.69 tons of dry pulp per hr. or 88.56 tons per day. The machine is rated at 50 tons dry pulp per day. The power required for rotation is 1 h.p., and for the vacuum-pump 4 h.p. When once set, the machine requires very little attendance. The machine complete, including classifier, pump and connections, weighs 11,000 lb. and costs $4,000 f.o.b. San Francisco.

Other filters[1] are the Ogle, Scherr, Hendryx, Allen, Parrish, Grothe-Carter.

The two leading representatives of the drum-type machine are the Oliver and the Portland; to these may be added the Just.

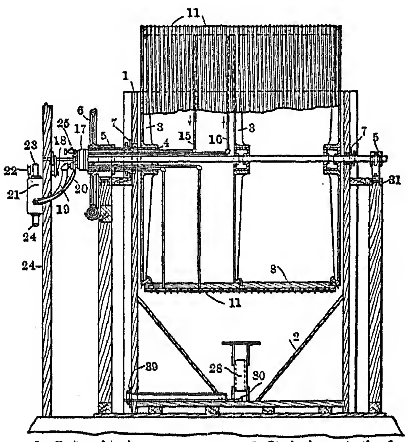
The Oliver continuous suction-filter,[2] Figs. 621 and 622, consists of a drum 11 ft. 6 in. in diam. and 8 ft. wide revolving partly submerged on a horizontal

[1] (Ogle, *Eng. Min. J.*, 1905, LXXIX, 372; (Scherr), *Min. Sc. Press*, 1908, XCVII, 195, 524; (Hendryx) *West. Chem. Met.*, 1907, III, 187; *Eng. Min. J.*, 1909, LXXXVII, 842; (Allen), *op.cit.*, 1909, LXXXVII, 1004; (Parrish), The Cyanide Plant Supply Co., 61 Fremont St., San Francisco, Cal; (Grothe-Carter), *Eng. Min. J.*, 1910, XC, 465.

[2] Martin, *Min. Sc. Press*, 1909, XCIX, 715; *Min. World*, 1909, XXXI, 1108.
Tweedy-Beals, *Tr. A. I. M. E.*, 1910, XLI, 349.
Kaeding, *Eng. Min. J.*, 1912, XCIV, 639.
Circular, Oliver Continuous Filter Co., San Francisco, Cal.
Nicholas, *Tr. A. I. M. E.*, 1912, XLIV, (priority).

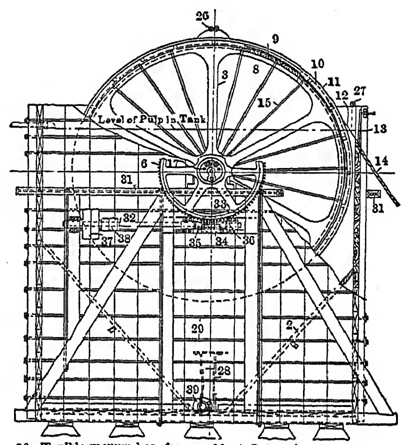
axis in vat (1), containing the pulp to be filtered. The pulp is agitated with compressed air (28) to prevent settling of solid parts.

The surface is built up of sections covered with filter-cloth (9) and connected on the inside with pipes (15 and 16) furnishing either a vacuum or air under pressure. When a section enters the vat, a suction is applied, and filtration begins; a cake of slime builds up on the surface, and when the section emerges from the pulp, the continued suction removes the solvent remaining in the cake; wash-water (26) is applied, the suction removes it and dries the cake until, just before re-entering the pulp, the suction is stopped and the cake dislodged by compressed air and falls off upon the tailing apron (14).



1 Redwood tank
2 False bottom
3 Cast-iron-spiders supporting filter drum
4 Hollow trunnion carrying spiders, piping, etc.
5 Main bearings
6 Worm-drive gear
7 Stuffing-boxes
8 Redwood staves for filter-drum
9 Filter medium
10 Division-strips dividing filter into sections
11 Steel-wire protection for filter medium
12 Steel scraper
13 Adjusting-screw and lever for scaper
14 Tailings-apron
15 Vacuum-pipes from filter-sections to automatic valve
16 Pressure-pipes from filter-sections to automatic valve
17 Automatic valve
18 Adjusting-lever for valve

FIG. 621.—Oliver continuous suction filter, transverse and elevation.



19 Flexible vacuum-hose for working-solutions
20 Flexible vacuum-hose for wash-solutions
21 Vacuum-chamber containing float check-valve
22 Vacuum-gauge
23 Pipe to dry-vacuum pumps
24 Pipe to solution-pump
25 Pressure-pipe for blowing filter-sections
26 Wash spray-pipe
27 Discharge spray-pipe
28 Agitator-pipe
29 Tie-rods
30 Agitating-air-nozzle
31 I-beam frame
32 Worm-shaft
33 Worm
34 Oil-well for worm
35 Thrust-collars for worm
36 Post-bearings for worm-shaft
37 Driving-pulleys
38 High-speed pulleys for wiring
39 Flange for drain

FIG. 622.—Oliver continuous suction filter, longitudinal section and elevation.

The drum is built up of 3.5-in. staves (8) bolted to three spiders. The perimeter of the drum is divided into 24 sections by 1-in. strips (10). Small strips, 0.5×0.5 in., nailed 1 in. apart form channels for the passage of solution in each section. Two 0.5-in. pipes (15 and 16) are connected to each section; they pass through the hollow shaft (4) to a plate having two circles with 24 holes each for 24 2-in. pipes. Facing this plate is a second plate (17) having a groove opposite the outer circle of holes. An adjustable bridge in this groove covers one hole, so that the groove being connected with the vacuum-pump, furnishes a vacuum to 23 of the sections. Air under pressure, passing through the pipe connected with the inner circle of holes in the plate, is admitted

to the section that is cut off from the vacuum by the bridge. As the drum revolves the vacuum is cut off from each section in turn, and air admitted.

A screen is placed on the 1-in. strips, forming the channels, which support a layer of burlap which in turn carries the canvas filter (9). The canvas is calked into a groove around the edge of the drum, and tied with steel wire (11) to make each section tight and to prevent bulging. The drum revolves once in 4 min., and is submerged for 3/5 of a revolution; the vacuum is maintained at 26 in.; the thickness of the cake varies from 3/16 to 3/8 in.; air under 5- to 10-lb. pressure agitates the pulp and dislodges the cake. The drum has a filtering surface of 290 sq. ft., and filters 60 tons of slime per day. The canvas lasts from 3 to 5 months, and it takes from 12 to 18 hr. to recover and re-wire a drum.

The Portland continuous suction filter[1] brought out by the Colorado Iron Works of Denver, Colo., has the general characteristics of the Oliver filter; it differs in the details of construction. A comparison of the two makes is not possible without detail drawings, and these are not available.

Another filter of this type is the Robinson.[2]

Wheel-type of Filter Presses.—In the wheel-type of suction filters a series of wheels mounted on a horizontal shaft revolves in a tank holding the pulp; the filtering members are leaves placed between the spokes. The filters[3] of Singer, Barnes and Holland belong to this class, as does also the Just.

309. Summary.—The conclusions which Young[4] has drawn from his exhaustive study of slime-filtration in the laboratory and in the mills are given in his own words:

1. The proportion of clayey material in ores which are to be subjected to "all-sliming" and filtration should be maintained at a minimum.

2. The slime-pulp should be as free as possible from sands coarser than a No. 150 screen, and as large a proportion of the pulp as possible should consist of material passing a No. 200 screen.

3. The slime-pulp before filtration should be settled to as thick a consistency as possible consistent with ready handling by pumps and in pipes.

4. The temperature of the slime-pulp should be maintained between 20° and 30° C. or higher.

5. The temperature of the wash-water and the pulp should be the same.

6. Vacuum-pressures should be varied until the proper intensity for the given slime is obtained.

7. Where very clayey slime is to be filtered, as much fine sand (limited as stated above) should be crowded into the pulp as it will carry without undue settling and clogging.

[1] *Circular*, Colorado Iron Works, Denver, Colo.
Lowden, *Min. Sc.*, 1910, LXII, 630.

[2] *Eng. Min. J.*, 1910, XC, 258.

[3] (Singer) U. S. Patent, No. 1017629, February 13, 1912; (Barnes) U. S. Patent, No. 1,036174, August 20, 1912; (Holland) U. S. Patent, No. 1036847, August 27, 1912; (Just) Spilsbury, *Tr. A. I. M. E.*, 1910, XLI, 367.

[4] *Tr. A. I. M. E.*, 1911, XLII, 783.

8. No. 10 canvas supported by slats gives the best all-round service for the thick cake, and No. 12 canvas on wire netting answers the requirements for the thin-cake filtering-machines.

9. With slimes containing a large proportion of colloid or clayey material pressures greater than those obtainable with vacuum apparatus are of questionable advantage.

10. With slimes containing a large proportion of clayey material the vacuum-filters should be used.

11. With slimes containing a small proportion of clayey material and much fine sand both vacuum-filters and pressure-filters could be used with perhaps equally good results.

12. With slimes containing much coarse and fine sand the chamber-filters with air-agitation and high pressures would perhaps give the best results.

13. Of the vacuum-filters, the thin-cake continuous filters (Oliver, Ridgway) are a decided improvement over the thick-cake filters (Moore, Butters).

D. GASES

AIR SUPPLY

310. Air-supply in General.—In all pyro-metallurgical processes it is necessary to supply furnaces with air to carry on the combustion which furnishes the heat, and to effect other processes of oxidation. Air is supplied either by draft or by blast, hence the apparatus will be of two kinds; one will create a draft by exhausting the products of combustion and thereby cause fresh air (fresh gas, with gaseous fuel) to flow in and fill the space partly evacuated; the other will force in fresh air and force out the products of combustion. The speedy removal of the latter is usually assisted by draft-apparatus. Draft is either natural or mechanical. Natural draft is obtained by chimneys; mechanical, or induced, draft by fans or injectors or ejectors.[1] As the last two devices serve also to furnish under-grate blast, often called forced draft, they will be taken up with the blowing apparatus.

The air supplied to a furnace is in most cases at ordinary temperature. In many instances it is preheated to make its action more energetic; in a few it is refrigerated to remove most of the water and thus obtain dry-blast with a low and uniform content of water. Air enriched in oxygen[2] has so far been used only in laboratory experiments; its application in large-scale work has been proposed, but has not yet been carried out.

311. Natural Draft. CHIMNEYS.[3]—A chimney is a vertical flue which is

[1] The practice of making a fire-room air-tight and maintaining therein the required pressure, common on board ships, is passed over.

[2] Lürmann, *Stahl u. Eisen*, 1912, XXXII, 609.

[3] Grüner, L., "Traité de Métallurgie Générale," Dunod, Paris, 1875, I, 273; Pietsch, F., "Der Fabrikschornstein," Craz and Gerlach, Freiberg, 1896; Bastine, P., "Berechnung und Bau Hoher Schornsteine," Felix, Leipsic, 1898; Christie, W. W., "Chimney Design and Theory," Van Nostrand, New York, 1902; Kent, W., "Mechanical Engineer's Pocket-book," Wiley,

connected at or near the bottom with a fireplace, and at the top is open to the air. The chimney-draft is caused by the difference in density and hence in weight between a body of heavy cold air P, Fig. 623, outside of the chimney and the same volume of light hot air, P', inside of it. The cold air displaces the hot air; on its way it passes through the fireplace and becomes heated; thus a constant suction is obtained, and P–P' represents the total suction or "head" h of the chimney. This head is defined as such a height of hot gas which, if added to the column of gases in the chimney, will produce the same pressure at the furnace as a column of outside air of the same area of base and of a height equal to that of the chimney. If H represents a column of air of the height of the chimney, α the expansion coefficient of air (0.00366 for 1 deg. C.), t' and t the temperatures inside and outside of the chimney, then $H(1+\alpha t') = (H+h)(1+\alpha t)$, or $h = \frac{H\times\alpha\times(t'-t)}{1+\alpha t}$. As $1+\alpha t$ is very small, it can be neglected. This leaves $h = H\times\alpha\times(t'-t)$.

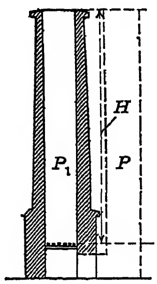


FIG. 623. Chimney-draft.

The theoretical velocity (ft. per sec.) with which the outside air would enter the chimney, provided there were no resistance, would be $v=\sqrt{2gh}$, in which g is the constant acceleration of gravity, 32.16 ft. per sec. Substituting the value for h, gives $v=\sqrt{2g\times H\times\alpha\times(t'-t)}$.

The quantity of air Q that passes through a chimney in a given time (per sec.) is proportional to the area of the chimney, A, the velocity of the gases v and their density d' at the chimney temperature t', or $Q=A\times v\times d'$. If d° is density of the air at zero deg., then $d'=\frac{d^\circ}{1+\alpha t'}$, hence $Q=A\times\sqrt{2g\times H\times\alpha(t'-t)}\times\frac{d^\circ}{1+\alpha t'}$. As A, d°, and $\sqrt{2gh}$ are constants, they can be expressed by C, and this gives $Q=C\sqrt{H}\times\frac{\sqrt{t'-t}}{1+\alpha t'}$.

The equation shows that with a given cross-section: (1) *The draft-power* varies as the square root of the height ($\sqrt{H}$). A chimney of double the height of another will have a draft-power of $\sqrt{2}$, or 1.4 times as large, of three times the height, $\sqrt{3}=1.73$ times as large. The ascent of the gases, however, is retarded by friction due to the roughness or smoothness of the walls. Its action may be considered as equivalent to a diminution of the actual area, A, of the chimney. Experience has shown that the effective area $A_e = A - 0.6\sqrt{A}$

New York, 1904, pp. 731–741; Richards, J. W., "Metallurgical Calculations," McGraw-Hill Book Co., New York, 1906, I, 157–171; Toldt, F., and Wilcke, F., "Regenerativ-Gasöfen," Felix, Leipsic, 1907, pp. 192–219; "Hütte," Engineer's Pocket-book, Ernst and Sohn, Berlin, 1908, II, 75; De Volson Wood, *Tr. Am. Soc. Mech. Eng.*, 1890, XI, 974; Kent, *op. cit.*, p. 985; Gale, *op. cit.*, pp. 451, 776, 980, 987; Rizer, *Clay-worker*, 1903, XL, 556; XLI, 602, 708; 1904, XLII, 20; *Report*, Prussian Commission, *Eng. Record*, 1901, XLIV, 52, 82; 1902, XLVI, 60, 495; Schiemann, F., *Proc. Phila., Eng. Club*, 1901, XVIII, 156; *Eng. Record*, 1901, XLIV, 271; Unger, C. H., *Eng. Record*, 1902, XLV, 195; Kingsley, F., *Eng. Record*, 1907, LVI, 679; Adams, *Eng. Mag.*, 1912, XLII, 824.

for square as well as for round chimneys.[1] As the friction increases with the height, there is a practical limit beyond which the draft will decrease instead of increase. Chimneys are therefore rarely made higher than 150 ft. A practical rule is to make the height of a chimney above the grate from 25 to 30 times the inner diameter as long as this does not exceed 8 ft.; the factor may be reduced to 20 times with a diameter over 8 ft.; in all cases the height of a chimney must be greater than that of neighboring buildings. Chimneys are built higher than 150 ft. only when obnoxious gases or vapors are to be dispersed high up in the air so that they may become sufficiently diluted to be harmless when they reach the earth. The chimneys of ordinary reverberatory furnaces are 50 or 60 to 100 ft. high.

TABLE 227.—SOME TALL BRICK CHIMNEYS

Works	Height, feet	Inside diam., feet		Outside diam., feet		Reference
		Base	Top	Base	Top	
Halsbrücken, Freiberg, Saxony.[2]	460	15.7	15.7	33	16	Christie, *op. cit.* *Oest, Zt. Berg. Hüttenw.*, 1891, XXXIX, 86.
Omaha & Grant, Denver, Colo.	350	16	16	33	20	Christie, *op. cit.*
Orford, Constable Hook, N. J.	360	20	10	32	11.83	Christie, *op. cit.*
Boston & Montana, Great Falls, Mont.[2]	506	62.5	50	78.5	53.75	*Eng. Min. J.*, 1909, LXXXVII, 156. *Eng. News*, 1908, LX, 583.
Anaconda, Mont.[2]	300	31.33	30	44.5	32.08	*Eng. Min. J.*, 1903, LXXVI, 962.

In boiler-practice the height of a chimney is varied with the rate of combustion that it is desired to attain per square foot of cross-section of chimney. Trowbridge[3] gives this relation in Table 228.

Thurston[4] expresses by $2\sqrt{H(\text{ft.})}-1$ the rate of combustion of anthracite

[1] Porter, *Eng. Min. J.*, 1906, LXXXI, 950.

[2] These three works are situated some distance below the bases of the chimneys, inclined flues serving as dust chambers leading the gases to them. The vertical distance from plant to base of chimney at Halsbrücken is 219 ft., at Great Falls, Montana, 275.5 ft., at Anaconda, 478.5 ft.

[3] Trowbridge, W. P., "Heat and Heat Engines," Wiley & Sons, New York, 1874, p. 153.

[4] *Tr. Am. Soc. Mech. Eng.*, 1890, XI, 991.

TABLE 228.—HEIGHT OF CHIMNEY AS RELATED TO RATE OF COMBUSTION

Height of chimney, feet	Pounds coal burned per hr. per sq. ft. cross-section of chimney	Pounds coal burned per sq. ft. of grate area; ratio of grate to cross-section of chimney, 8 : 1	Height of chimney, feet	Pounds coal burned per hr. per sq. ft. cross-section of chimney	Pounds coal burned per sq. ft. of grate area; ratio of grate to cross-section of chimney, 8 : 1
20	60	7 5	70	126	15.8
25	68	8.5	75	131	16 4
30	76	9 5	80	135	16.9
35	84	10 5	85	139	17 4
40	93	11.6	90	144	18 0
45	99	12 4	95	148	18 5
50	105	13.1	100	152	19.0
55	111	13.8	105	156	19 5
60	116	14.5	110	160	20.0
65	121	15 1	....	....	....

in lb. per sq. ft. of grate area per hr. with a chimney of given height. Table 229 gives data for heights H ranging from 50 to 200 ft. The data agree with those of Trowbridge.

TABLE 229.—HEIGHT OF CHIMNEY AS RELATED TO COMBUSTION OF ANTHRACITE COAL

Height (ft.)	50	60	70	80	90	100	110	125	150	175	200
$2\sqrt{H}-1$	13.14	14.49	15.73	16.89	17.97	19	19.97	21.36	23.49	25.45	27 28

(2) *The draft-power* of a chimney is proportional to $\frac{\sqrt{t'-t}}{1+\alpha t'}$; that is, it increases as the square root of the difference of internal and external temperatures, and decreases with the expansion of the gases by the rise of temperature. As the denominator increases more rapidly than the numerator, a point is reached beyond which a rise in temperature will not increase the draft.

Calculation of the maximum for $t=0$, gives $t'=273°$ C. Determining[1] the values for $\frac{\sqrt{t'-t}}{1+\alpha t'}$, for temperatures between 100 and 500 deg. with $t=0$, gives the data of Table 230, the results of which are shown graphically in Fig. 624. These show that with a chimney-gas of 100 deg. the draft is strong; that it increases somewhat with 200 deg., and hardly at all between 200 and 273 deg. A temperature of 200 deg. is therefore generally accepted as the highest mean temperature of the chimney; in other words, furnace-gases can be cooled down to 200 deg., their heat being utilized for raising steam or other heating purposes, and the draft-power of the chimney not interfered with.

[1] Grüner, *op. cit.*, I, 284.

TABLE 230.—RELATION OF TEMPERATURE AND DRAFT-POWER

t'	100	200	273	300	400	500
$\dfrac{\sqrt{t'-t}}{1+\alpha t'}$	7 4	8.16	8 255	8 251	8.110	7.89

The area, A, of a chimney required for a given case can be calculated from the quantity of gas Q' and the velocity v of the gas to be discharged per second as $Q' = A \times v$, hence $A = \frac{Q'}{v}$. If t' is the temperature of the gases, Q° their volume at o deg. C., then $Q' = Q^\circ\,(1+0.00366t')$ and $A = Q^\circ \times \frac{1+0.00366t'}{v}$. In general, 1 lb. bituminous coal burnt with the necessary excess-air, usually 24 lb. air, may be said to furnish about 275 cu. ft. of products of combustion at o deg. Assuming that the gases enter the base of the chimney at 275 deg., then according to $Q_t = Q_o \times (1+0.00366t)$, or $Q_{275} = Q \times 2$, or 1 lb. bituminous coal furnishes 550 cu. ft. of products of combustion at 275 deg. C.

The value for v shows a range of 10 to 23 ft. According to Grüner,[1] 16–17 ft. is the highest value for a gas-temperature of 200–300 deg. C. when there is no excessive friction, and 10–13 ft. the lowest value when there is excessive friction as, *e.g.*, in Siemens regenerative chambers. The last is a good average value for large flues of small cross-section or for large flues in which dust is to be deposited; v should never be less than 7 ft. with chimneys placed back of an ordinary reverberatory furnace.[2]

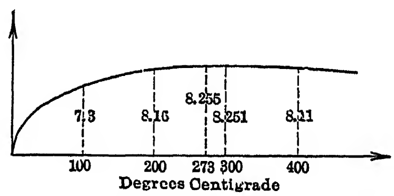


FIG. 624.—Chimney-draft and temperature of gases.

In practice it is common to have a definite relation between the sectional area of the chimney and the total grate area or the free area, *i.e.*, the area of grate interstices, as the rate of combustion is largely governed by it. Thus, with a 50- to 100-ft. chimney, its cross-sectional area with metallurgical furnaces is made from 1/2 to 1/5 the size of the total grate area or from 1/3 to 1/5 of the free area; the larger value refers to bituminous coal, the smaller one to lignite, peat or wood. With boilers, which burn fuel more slowly than metallurgical furnaces, the ratio is from 1 : 6 to 1 : 8.

With the same chimney a small grate will give rapid, and a large grate slow combustion. The former promotes the formation of CO_2, the latter of CO; hence if the grate is too large it causes waste, but this can be corrected by covering part of the grate with brick. In a similar manner, since it is permissible to reduce the effective area of a chimney by means of a damper or a gate, it is safer to err in making the sectional area a little too large than too

[1] *Op. cit.*, I, 295.
[2] Roitzheim, *Stahl u. Eisen*, 1912, XXXII, 969, 1098.

small. If the area is too large, however, descending air-currents are produced which interfere with the draft.

The *draft* is usually measured by means of a draft- or a pressure-gauge. The most common form is a U-shaped tube of which one end is open to the atmosphere while the other is connected with the space in which a different pressure exists. The tube is filled with a liquid; water, colored glycerine, oil, or phenol for small, and quicksilver for large differences in pressure. The excess of pressure in one limb over that in the other forces the liquid down in the first and causes it to rise a corresponding amount in the second; the difference in level represents the height of a column of liquid which will be balanced by the excess of pressure over the atmosphere. The pressure is expressed in inches of the liquid used or in oz. or lb. per sq. in. If the weight of 1 cu. ft. (1728 cu. in.) water at ordinary temperature is 62.33 lb., then a column of water 1 in. sq. and 1728 in. high will exert a pressure of 62.33 lb., or a column 27.7 in. high a pressure of 1 lb., or a column 1.73 in. high a pressure of 1 oz., or a column 1 in. high 1 in. head =0.578 oz. Table 220[1] gives the relation between water-column in in. and pressure in oz. per sq. in.

TABLE 231.—RELATION OF WATER-COLUMN IN INCHES AND PRESSURE IN OUNCES PER SQUARE INCH

Water column, inches	Decimal parts of an inch									
	0.0	0.1	0.2	0.3	0.4	0.5	0.6	0.7	0.8	0.9
0	......	0.06	0.12	0.17	0.23	0.29	0.35	0.40	0.46	0.52
1	0.58	0.63	0.69	0.75	0.81	0.87	0.93	0.98	1.04	1.09
2	1.16	1.21	1.27	1.33	1.39	1.44	1.50	1.56	1.62	1.67
3	1.73	1.79	1.85	1.91	1.96	2.02	2.08	2.14	2.19	2.25
4	2.31	2.37	2.42	2.48	2.54	2.60	2.66	2.72	2.77	2.83
5	2.89	2.94	3.00	3.06	3.12	3.18	3.24	3.29	3.35	3.41
6	3.47	3.52	3.58	3.64	3.70	3.75	3.81	3.87	3.92	3.98
7	4.04	4.10	4.16	4.22	4.28	4.33	4.39	4.45	4.50	4.56
8	4.62	4.67	4.73	4.79	4.85	4.91	4.97	5.03	5.08	5.14
9	5.20	5.26	5.31	5.37	5.42	5.48	5.54	5.60	5.66	5.72

In round numbers, 1 lb. or 16 oz. pressure per sq. in. =28 in. of water =2 in. of quicksilver =22 in. of glycerine (sp. gr. 1.28).

The draft of a chimney may be as much as 0.60 in. water in winter; this figure falls in summer to perhaps 0.40 in. In cases where the gases from a hearth at a red heat enter the chimney direct, the draft is twice and even three times as strong.

[1] Snow, W. B., "Mechanical Draft," Sturtevant Engineering Co., Hyde Park, Mass., 1899, p. 160.

Fig. 625 represents an ordinary form of draft-gauge. The U-tube is filled with water and provided with a scale having the zero mark in the center. This form lacks the sensitiveness necessary for measuring small drafts, such as are common in the filtering of gases. In the modern Wallaston draft-gauge,[1] Fig. 626, this disadvantage is remedied by having the narrow U-tube *A* surmounted with chambers *B* and *C* that are filled with a liquid which will not mix with that contained in *A*, and is lighter, but has approximately the same specific gravity. Thus dark-colored phenol and a saturated colorless solution of phenol in water; or olive oil, perhaps a lubricating oil, and a mixture of alcohol and water with a difference in specific gravity of 2 per cent. between oil and mixtures, are used for this purpose. In the gauge the fall of the line of demarcation *x* is proportional to the difference in the surfaces of the chambers *B* and *C* and the inside sections of the tubes *A*. If the ratio is as 10 : 1, a fall of 0.1 in. in B will cause a fall of the line of demarcation *x* of 1 in. The instrument is calibrated with the ordinary U-tube. The Fallis manometer[2] has a similar construction.

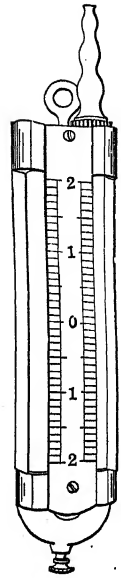


FIG. 625.—Ordinary form of draft-gauge.

The Ellison differentiated draft gauge,[3] Fig. 627 is another sensitive instrument. It consists of a case holding an inclined U-shaped glass tube, one upright limb having been enlarged to a cylindrical chamber, and a level. The rise and fall of the single liquid is proportional to the surface areas of the liquid in the tube and the chamber; thus a slight change of level in the chamber will show a considerable rise or fall along the scale placed against the tube. The case for gauges denoting a vacuum of 1/2, 1 and 2 in. is 4 in. wide by 1 in. deep by 13.2 in. long; that for a vacuum of 1 1/2 and 3 in. is 5 in. wide by 1 in. deep by 18.4 in. long. The scale for the 1/2-, 2-, and 3-in. gauges has 20 graduations to the inch, that for 1- and 1 1/2-in., has 10.

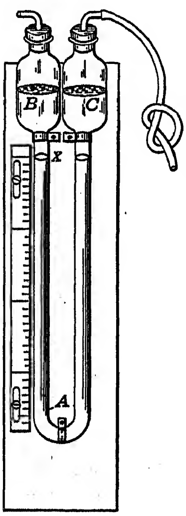


FIG. 626.—Wallaston draft-gauge.

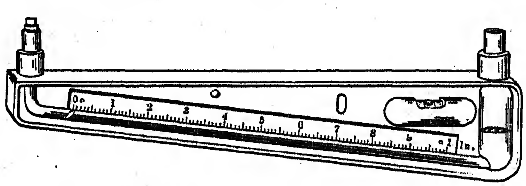


FIG. 627.—Ellison differential draft-gauge.

[1] *Tr. Am. Ceram. Soc.*, 1900, I, 88; *Thonind. Z.*, 1900, xxiv, 102, 1044.
[2] Iles, *Eng. Min. J.*, 1900, LXIX, 765.
Brown, *op. cit.*, 1900, LXX, 93.
[3] American Steam Gauge and Valve Mfg. Co., Boston, Mass.

Fig. 628 shows the Seymour draft-gauge[1] which is different in construction from the preceding two sensitive gauges, but also indicates the slightest changes in draft. It consists of a lower circular dish of sheet-metal, 12 in. in diam. and 3 in. high, half full of water; through the bottom projects a vertical tube, 2 in. long, that is connected with the chimney and is provided with a cock. In this dish is inverted an upper similar dish, 10 in. diam., which is suspended from a steel yard. Assume that the strongest draft to be measured is 1 in. of water. A column of water 10 in. in diam. and 1 in. high is easily calculated to weigh 2.832 lb. If the gauge is balanced under atmospheric pressure with the sliding weight at zero, the addition of 2.832 lb. on the inverted dish and balancing the gauge by sliding the weight on the arm will locate the point at which the draft will be equal to 1 in. water. Dividing the distance between this point and zero into 100 parts, each division will represent a draft of 1/100 in. of water, after the 2.832 lb. have been removed.

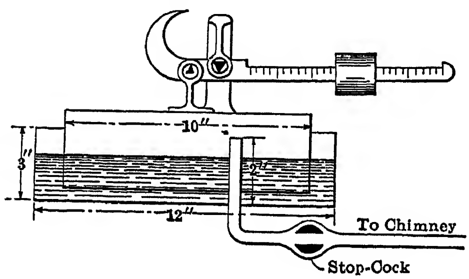


FIG. 628.—Seymour draft-gauge.

The draft-gauge of Kent[2] is similar. See also Dosch,[3] Weymouth,[4] Smallwood,[5] Stach,[6] Contzen.[7] Besides the gauges described which are designed for independent observation, there are gauges in use which furnish continuous records, *e.g.*, the mechanical draft recorder of the Crosby Steam Gauge & Valve Co., Boston, Mass., or Hohmann & Maurer Mfg. Co., Rochester, N.Y.[8]

Chimneys are built[9] of brick or steel plate. In recent years concrete[10] has replaced brick in many instances. Even wooden stacks are in use for carrying off blast-furnace gases, *e.g.*, at Mapimi, Mexico.[11] There is little choice be-

[1] *Coll. Engineer*, 1889–90, X, 195.

[2] Kent, W., "Steam Boiler Economy," Wiley & Sons, New York, 1901, p. 363.

[3] *Stahl u. Eisen*, 1910, XXX, 117.

[4] *Min. Sc. Press*, 1912, CIV, 562; *Met. Chem. Eng.*, 1912, X, 315.

[5] *Comp. Air*, 1912, XVII, 6288.

[6] *Glück auf*, 1910, XLVI, 1833, 1869; *Stahl u. Eisen*, 1911, XXXI, 1752, 1880.

[7] *Op. cit.*, 1912, XXXII, 573.

[8] See also Thomas, *Iron Age*, 1911, LXXXVII, 676; *J. Frankl. Inst.*, 1911, CLXXII, 411; Hydro Mfg. Co., Pittsburgh, Pa., *op. cit.*, 1910, LXXXV, 114; *Eng. Min. J.*, 1910, LXXXIX, 811.
Neuhauer, *Oest. Zt. Berg. Hüttenw.*, 1910, LVIII, 21; General Electric Co., Schenectady, N. Y., *Bulletin* No. 4720.

[9] Roch, P., "Baukunde für Berg- und Hüttenleute," Craz und Gerlach, Freiberg, 1901, pp. 281–293.

[10] Ransome system, *Eng. Rec.*, 1901, XLIV, 517.
Schuyler, *op. cit.*, 1903, XLVII, 374.
Butte Reduction Works, *op. cit.*, 1906, LIII, 124.
Custodis Chimney at Seaton, Carew, England, *Eng. Min. J.*, 1909, LXXXVIII, 777.
Jensen, *Eng. News*, 1912, LXVII, 866.
Stewart, *Eng. Min. J.*, 1911, XCII, 153.

[11] Magenau, *Min. Sc. Press.*, 1901, LXXXII, 820.

tween brick and steel as to draft, since the greater friction of the brick is offset by the more rapid radiation with the steel. Brick chimneys are more expensive[1] than steel ones; metallurgical fumes are often heavily charged with sulphurous gases which would attack the steel; however, steel chimneys are often lined in order to protect the metal and to diminish the loss in heat.

Chimneys are circular, octagonal or square in cross-section; large brick chimneys are usually circular, as is always the case in the steel chimneys. The circular form is the best, as it causes the least friction, has the smallest cooling surface, and requires less building-material than the other two forms. With square chimneys just the reverse is the case; the octagonal chimney stands between the two. As, however, the difference in draft between the circular and the square chimneys is only 2 per cent., small brick chimneys are usually built square, because they are cheaper, requiring only ordinary brick and not especially skilled masons.

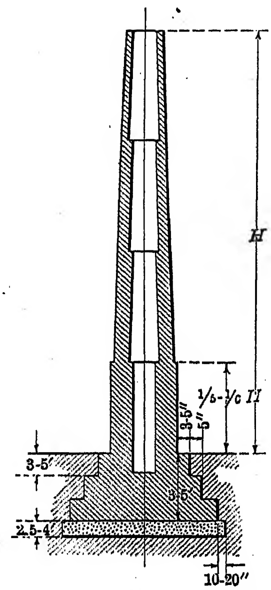


FIG. 629.—Vertical section through brick chimney.

The vertical section of the shaft-like flue is usually rectangular, rarely trapezoidal. Considering that the frictional resistance of the chimney is only about 1/10 that of the furnace, the slight taper from bottom to top that might be used increases the draft too little to make up for the increased cost.

The top of a small brick chimney, having a plain or bluff top, is usually covered with a stone or a metal plate; large chimneys have a cap which is concave toward the orifice that it may deflect the wind upward and thus help the draft. Chimneys are provided with lightning rods,[2] which must be well grounded. Ladder-ways are also provided.

The thickness of the brick wall and its outer form are governed by the temperature of the gases and the stability of the structure. Considering that the weight of the chimney increases from the top downward, the sectional area of the brickwork must be correspondingly enlarged. This gives the outer form a taper or batter which has a range of from 1/4 to 1/16 in. per ft. The brickwork is made one brick thick (8 or 9 in.) for the first 25 ft. from the top, and increases half a brick (4 or 4 1/2 in.) for each 25 ft. downward. If the diameter of a circular, or the inscribed circle of a square chimney exceeds 5 ft., the upper 25 ft. should be 1 1/2 brick thick; if it is less than 3 ft., the upper 10 ft. may be 1/2 brick thick. In order to save in material, and to permit the use of standard sizes of brick, a chimney is not built with a uniformly increasing thickness of wall, but the thickness is increased by a series of steps as shown in Fig. 629.

[1] Babcock & Wilcox Co., *Iron Age*, 1889, XLIII, 546.
Crane, *op. cit.*, 1898, LXII, Oct. 20, p. 16.

[2] Standards, of the U. S. Navy, *Eng. Rec.*, 1907, LVI, 291.

With chimneys from 90 to 120 ft. high, these steps are made from 17 to 25 ft. long. With circular chimneys perforated brick, Fig. 630, are frequently used in the outer shell. These perforations[1] are filled more or less with mortar and thus furnish an additional bond. The brick at the bottom are laid in ordinary mortar, at the top in cement, as mortar can stand more heat than cement, while cement is stronger than mortar; in the shaft the mortar is replaced stepwise by cement.

In many instances the chimney extends into the ground and thus forms the foundation; usually, however, it rests on a square or, with circular chimneys, on an octagonal base which has a curved rising opening serving as an inlet-flue. The height of the base, Fig. 629, usually is from 1/5 to 1/6 that of the height of the chimney and its walls extend 1 1/2 to 3 1/2 ft. beyond the outer chimney-line if prolonged to the foundation.[2] The shaft of the chimney ought to extend

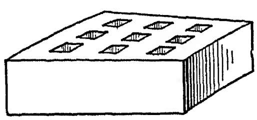


FIG. 630.—Perforated brick.

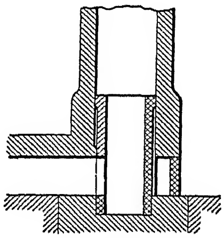


FIG. 631.—Flue-opening and port of chimney.

from 2 to 3 ft. below the flue-opening in the base, or the foundation, in order to form a pocket for collecting flue-dust. Opposite the flue-opening there is usually a port, Fig. 631, closed by a light wall, to furnish ready access.

Chimneys of any considerable size are built of two shells which are independent of one another, Figs. 632, and 636. The outer shell, or visible stack, must have sufficient strength to give stability to the structure; the inner shell, or core, may reach up only a short distance in the outer, but usually extends to the top or within a few feet of it. In the first case, the core is made half a brick thick and laid in clay; the air-space, 2 to 6 in. wide, is covered to prevent its becoming filled with flue-dust. In the second, it is built in steps, Fig. 636, as is the outer shell. The thickness at the top is about one-half that of the outer shell; it can be thinner, as it is protected from the wind by the outer shell, and has therefore to carry only its own weight. Clay-mortar is ordinarily used, at least for the lower part. At intervals of 10 or 20 ft. brick projections reach from the outer to the inner shell to furnish a lateral support. If more than one flue enters the base, vertical partition walls have to be erected in order that the gases in each flue may be deflected upward independently from those of the others before they meet and pass off together.

The foundation, if built on compact sand, loam or gravel, should not exert

[1] Radial brick chimney at Newark, N. J., *Eng. Rec.*, 1903, XLVIII, 46.
Christie, *Cass. Mag.*, 1906, XXIX, 269.

[2] Irwin, *Eng. Min. J.*, 1904, LXXVII, 203.

a pressure greater than 2 tons per sq. ft.; with solid rock this figure may be increased to 5 tons. With soft ground, it may be necessary to start with a layer of concrete 2 1/2 to 4 ft. thick which extends 10 to 20 in. beyond the foundations. It may even be necessary to drive piles before laying the concrete. As shown in Figs. 629, 632, and 633, the brick foundation is spread out so that the permissible pressure may not be exceeded; it is built in steps or in form of a trapezoidal block with sides forming an angle of 45 or often 60 degrees with the horizontal. In all cases the thickness beneath the bottom of the shaft should not be less than 3 to 5 ft.

The stability of a chimney depends upon the vertical pressure of the brickwork and the lateral pressure of the wind. The maximum pressure of wind is to

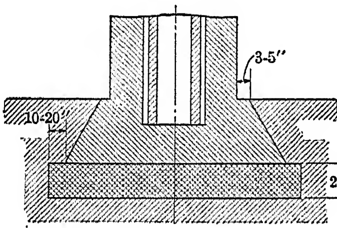


FIG. 632.—Base of chimney with inner and outer shell.

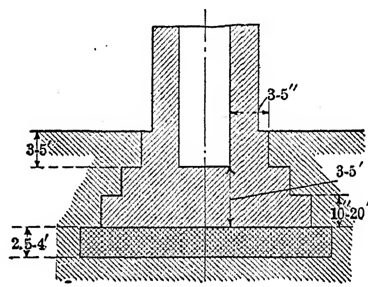


FIG. 633.—Foundation of chimney.

be 55 lb. per sq. ft. on a flat surface; this is in excess of the pressure exerted by an immense hurricane, or a velocity of 100 miles per hr. = 49.2 lb. per sq. ft. On a round chimney the pressure is assumed to be 0.67 that of the square chimney, with an octagonal chimney 0.71, on a hexagonal chimney 0.75. The wind-pressure on a flat-surface chimney, Fig. 634, is $A = H \times \frac{D+d}{2} = 90 \times \frac{12+6}{2} = 810$ sq. ft. The wind-pressure of 55 lb. per sq. ft. on a circular chimney is: $P = 0.67 \times 55 \times A = 29{,}848$ lb. The height of the center of wind-pressure on a trapezoid, $H_0 = \frac{H}{3} \times \frac{D+2d}{D+d} = \frac{100}{3} \times \frac{12+2\times 6}{12+6} = 39.9$ ft. The cubic content of the solid chimney, $Vs = \left[\left(\frac{D}{2}\right)^2 + \left(\frac{d}{2}\right)^2 + \frac{d}{2} \times \frac{D}{2}\right] \times \frac{H \times \pi}{3} = \left[\left(\frac{12}{2}\right)^2 + \left(\frac{6}{2}\right)^2 + \frac{6}{2} \times \frac{12}{2}\right] \times 100 \times \frac{3.14}{3} = 6030.8$ cu. ft., that of the core, $Vc = \left[\left(\frac{D'}{2}\right)^2 + \left(\frac{d'}{2}\right)^2 + \frac{d'}{2} \times \frac{D'}{2}\right] \times \frac{H \times \pi}{3} = \left[\left(\frac{8}{2}\right)^2 + \left(\frac{4.66}{2}\right)^2 + \frac{4.66}{2} \times \frac{8}{2}\right] \times 100 \times \frac{3.14}{3} = 2896.1$ cu. ft., hence the cubic content of the masonry, $V_m = V_s - V_c = 6030.8 - 2896.1 = 3134.7$ cu. ft., and its weight, $W = V_m \times 125$ (*wt* of 1 cu. ft. hard brick) lb $= 3134.7 \times 125 = 391{,}837$ lb. Its stability moment, $M_s = \frac{W \times D}{2} = \frac{391837 \times 12}{2} = 2351022$; its overturning moment, $M_o = P \times H_o = 29848 \times 39.9 = 1{,}190{,}935$, which gives its stability factor, $S = \frac{M_s}{M_o} = \frac{2351022}{1190935} = 1.98$, and this is close enough to 2 to make it abso-

lutely safe. The stability of a chimney can be calculated as a whole, or preferably each section separately.

A graphic method[1] for determining the stability is shown in Fig. 635. Here $ABCD$ represents the vertical section of a chimney or a division of it. In order to find the center of wind-pressure O, draw the diagonals AD and BC, and on BC lay off $CE=NB$; bisect AD which gives F, and join E and F; the point O where EF cuts the center-line of the figure is the center of wind-pressure, the point at which the wind is supposed to act. Draw the horizontal and vertical

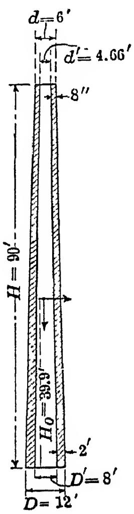


FIG. 634.—Stability—Calculation of a circular chimney.

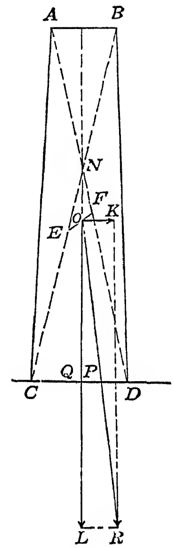


FIG. 635.—Stability—Construction of chimney.

components OK and OL of a parallelogram of forces, and lay off at the same scale on OK the intensity of the total wind-pressure, on OL that of the total weight of the chimney, and complete the parallelogram; the resultant OR will cut the base CD at P. The chimney is stable as long as QP with a square chimney is less than 1/3 CD, and with a circular chimney less than 1/4 CD. There will be no tension in the mortar on the windward side, and on the leeward side the highest compression will not be over twice the average compression due to that of weight alone. A safe load for hard-burnt brick is 10 tons per sq. ft.

Steel-plate chimneys are either self-supporting or guyed. With the former, the shell at the base is heavy (1/2-in. steel) and splayed out so as to give additional bearing on the foundation. The shell is bolted to a foundation-ring of cast-iron which is anchored. The chimney has a straight taper; the plates are riveted in rings 50 to 60 in. high, and grow thinner from bottom to top. Steel chimneys are either full-lined, half-lined, or unlined, depending upon the temperature and character of the gases. When lined with fire-brick, the lining is made half a brick thick at the top and increases in thickness downward. It

[1] P. Schwamb, "Notes on Stability of Chimneys," Mass. Institute of Technology, p. 13.

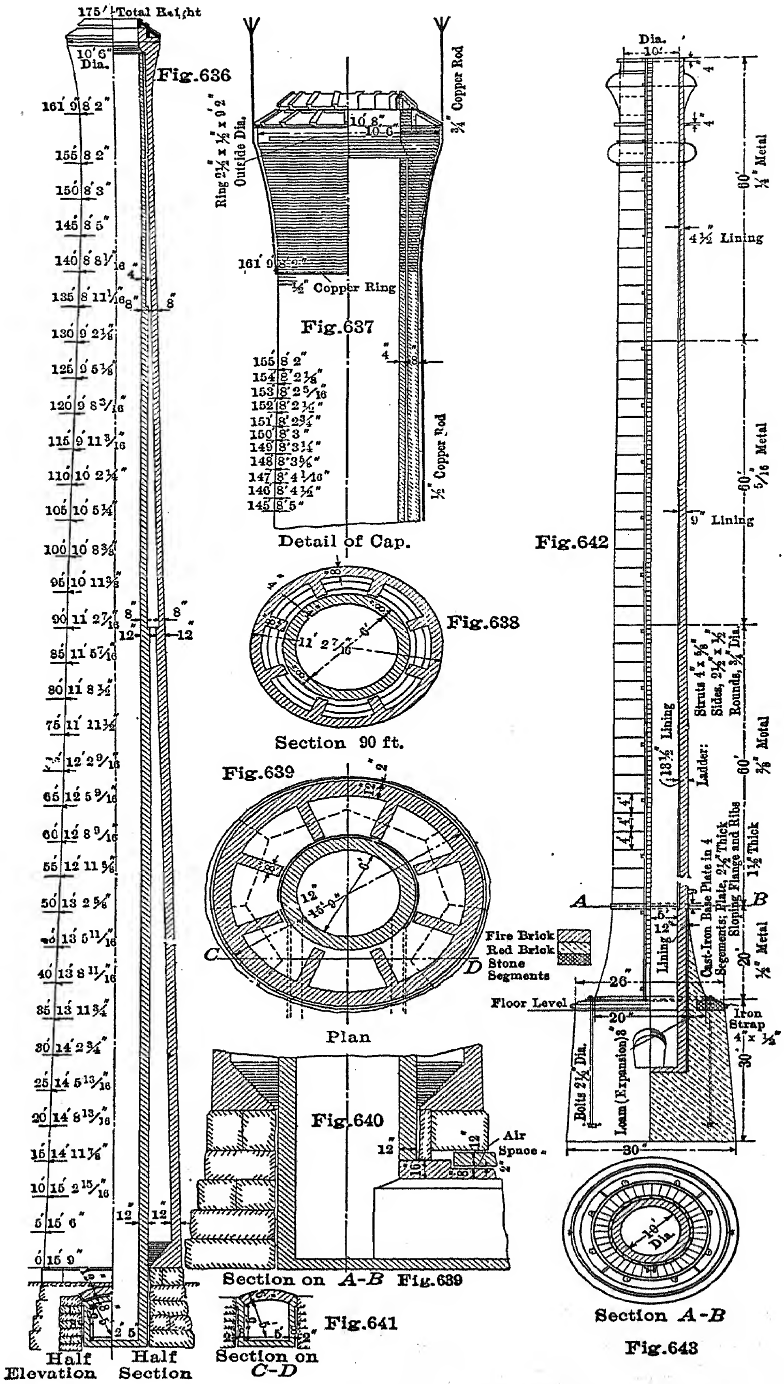
lies close to the steel, and being carried by the plate adds nothing to the strength of the chimney. The internal diameter is made uniform from bottom to top. The life of a steel chimney is estimated at about 12 years; the steel has to be painted about once in two years; the chimney should not stand idle, as it corrodes rapidly.

Figs. 636 to 641 represent a circular brick chimney at Fall River, Mass., 6 ft. in diam. and 175 ft. high, built by W. T. Henry to serve seven 72-in. horizontal tubular boilers, each of which has a grate-area of 33 sq. ft. The chimney is very slim and rests on bed-rock. The foundation is built of rough stone surmounted from the surface of the ground by dressed stone, Figs. 636 and 640. The chimney has an outer and inner shell built in steps, Fig. 636; the inner diameter of the core remains uniform throughout; the external diameter of the outer shell, 15 ft. 9 in. at the base, decreases to 8 ft. 2 in. at a height of 161 ft. 9 in.; the inner stack is carried down inside of the foundation of the outer stack; the brickwork of the outer stack, Figs. 636 and 640, is splayed out inside at the bottom to within 1 in. of the core so as to distribute the pressure over the stone foundation; at the top, Fig. 637, it is drawn in to the same diameter as the core; buttresses, Figs. 638 and 639, reaching from the outer to the inner shell give the latter the desired stiffness. The inner stack is 12 in. thick at the bottom and 4 in. at the top, while the outer stack, also 12 in. thick at the bottom decreases to 8 in. at the top; the latter is protected by two rows of special tiles, Fig. 637; and an iron hoop, 2 1/2×2 in. and 9 ft. 2 in. in outside diam., is built into the top to tie it together securely. There are two lightning rods of 3/4-in. copper which are united at the 161 ft. 9 in.-level to a 1/2-in. copper ring from which a 1/2-in. copper rod extends down the side of the chimney into the ground.

The flue, rectangular with arched top, enters the foundation, a large stone, Fig. 640, spanning the opening; a 16-in. arch, Figs. 640 and 642, supports the wall of the core, while the remainder of the face, Fig. 636, is arched over with an 8-in. wall and this with a 12-in. wall, there being a 2-in. air-space at the top and sides excepting where support is needed for the arch. The mortar used was composed of 1 vol. Rosendale cement and 2 vol. lime mixed with sharp sand. The brickwork is bonded through every eighth course.

Other examples: Plymouth Cordage Co., Plymouth, Mass.;[1] Presidential Building, Newark, N. J.;[2] Yuerba Buena Power House;[3] Plymouth Works, British Coalite Co.,[4] Easton, Pa.;[5] *Concrete Reinforced*,[6] Tacoma Smelting Co.;[7] Wiederhall type.[8]

[1] *Eng. Rec.*, 1901, XLIII, 466.
[2] *Op. cit.*, 1901, XLIV, 567.
[3] *Op. cit.*, 1904, XLIX, 592.
[4] *Stahl u. Eisen*, 1909, XXIX, 965.
[5] *Clay Worker*, 1911, LVI, 277.
[6] Weidner, *Stahl u. Eisen*, 1909, XXXII, 1508.
Weber, *Iron Age*, 1910, LXXXV, 330.
Faber, *Engineering*, 1911, XCII, 824.
[7] *Min. Ind.*, 1905, XIV, 178.
[8] *Iron Age*, 1911, LXXXVIII, 689.



FIGS. 636 to 641.—Circular brick chimney.
FIGS. 642 and 643.—Lined steel-plate chimney.

Figs. 642 and 643 represent a lined steel-plate chimney, 10 ft. in diam. and 200 ft. high, designed by A. Laughlin of Pittsburgh, Pa. The foundation, Fig. 642, is red brick; at the floor level it is covered with a course of stone that is tied by a wrought-iron strap made in two parts and bolted together, Fig. 643. The stone carries the cast-iron foundation-ring, 2 1/2 in. thick and made in four segments. The shell is bolted through a steel bell-plate, 1 1/2 in. thick, and the cast-iron foundation-ring by means of eight 2 1/2-in. rods which are connected with anchor-plates imbedded in the foundation. The base is of 1/2-in. steel and is splayed at the bottom, and the stack is slightly tapered; the plates are 4 ft. wide and decrease in thickness from 3/8 to 1/4 in. The stack is full-lined; the self-sustained lining, 18 in. in the lower part, decreases by inner steps to 4 1/2 in. at the top; a 3-in. expansion-space filled with loam is left between the lining and the steel shell. The top of the chimney has an external finish of light plate. Other examples are cited in the footnote.[1]

312. **Mechanical (Induced) Draft.**[2]—This form of draft is one which induces a flow of air by means of exhausters, usually centrifugal fan-blowers. In special cases steam-ejectors are used. In steam-boiler practice artificial draft is replacing the common natural draft. The usual plan of operating is to have a large fan, with a speed equal to the desired velocity of the gases, to draw the products of combustion through a heat economizer and discharge them through a steel chimney of sufficient height (30 to 50 ft.) that they may be sufficiently dispersed so as not to become a nuisance. The advantages claimed for fan-draft are: cheapness of first cost of plant; cheapness of operation in connection with heat-economizer, as a fan requires only 1 per cent. of the power supplied to the engines by the boiler; use of heat economizers to an extent not possible with chimneys; brisk fire with thick fuel-bed and hence less excess air; control of air-supply and hence of combustion, and with flexibility permitting increase or decrease of draft; independence of the condition of weather; use of low-grade fuel; increase of capacity of existing plant.

The main disadvantage is that the economizer tubes must be filled with warm water, as cold water inside the tubes causes condensation of products of combustion on the outside; hence the gases leave the tubes at 150 to 200°. C., and a chimney with this average gas temperature shows a draft-power not far from its maximum.

[1] Examples: Sparrow's Point, Md., *Iron Age*, 1889, XLIII, 546; Denver, Colo., *Eng. Rec.* 1904, L, 314; Tacoma, Wash., *Eng. Min. J.*, 1905, LXXX, 631, 1076; Christie, *Cass. Mag.*, 1906 XXIX, 267; Stuctural and Corrugated Steel, Barbour, *Eng. Min. J.*, 1912, XCIII, 303. Vitribestos lining *Eng. Min. J.*, 1913, XCV, 741.

[2] Snow, W. B., *Iron Age*, Dec. 1, 1898, p. 9 "Mechanical Draft," B. F. Sturtevant Co., Hyde Park, Mass., 1899.

Booth, *Cass. Mag.*, 1901–02, XXI, 130.

Snow, *op. cit.*, 1906, XXIX, 399.

Cobb, *Eng. Min. J.*, 1905, LXXX, 676.

Boskowitz, *Thonind. Z.*, 1908, XXXII, 2197 with cross-references.

Gaab, *Stahl u. Eisen*, 1907, XXIX, 389.

Brinkerhoff, *J. Frankt. Inst.*, 1911, CLXXI, 463.

Künzel, *Oest. Zt. Berg. Hüttenw.*, 1909, LVIII, 654.

In metallurgical apparatus artificial draft is employed with furnaces from which fumes are to be recovered by dry-filtering, as is the case with some lead, antimony and zinc-white furnaces, or gases to be washed as with some iron blast furnaces, or with wet-condensation in general. Further, in the manufacture of sulphuric acid the gases from roasting-kilns are drawn through dust-chambers before they are delivered to the lead chambers.

313. **Blast in General.**[1]—The air-supply furnished a furnace by natural draft may have a harmful effect upon the metallurgical process that is being carried on; it may also be insufficient in quantity. In either case blast will correct the evil.

In a reverberatory furnace worked by draft, air is sucked in through imperfectly closed working doors, cooling the charge and having an oxidizing effect upon it. Reverberatory roasters which require little heat, but much air, are therefore preferably worked with natural draft. By closing the ash-pit and blowing air underneath the grate, the pressure of the gases inside the furnace becomes greater than that outside; this causes some smoke or flame to pass outward through the cracks between the door-frames and the doors, and prevents air from entering. Thus, direct-fired heating furnaces for packets of muck-bar, or for steel ingots or billets, are usually worked with undergrate-blast. This so-called forced-draft or closed ash-pit system has advantages similar to those of induced draft when compared with natural draft. The main disadvantage is that vigorous local action of unevenly distributed blast causes temperatures to be reached which are sufficiently high to attack the walls of the fire-chamber and to cause the formation of hard clinkers. This is corrected in part by admitting with the compressed air some steam which keeps the grate cool and softens the clinkers. Nevertheless, most copper reverberatory smelting-furnaces are worked with natural draft, as the hearth from fire-bridge to flue-bridge is more evenly heated than with forced draft; false air is excluded from the hearth by carefully luting the side-doors with clay. In shaft-furnaces only coarse-ore charges can be worked with natural draft; fine-ore charges require blast whether they are to be roasted or smelted.

The mechanism of most blast-apparatus is similar to that of moving liquids. The modifications in the details of construction are caused by the differences in density and compressibility of liquids and gases. Thus there are employed injectors, exhausters, centrifugal fans, rotary pressure-blowers, and reciprocating blowing-engines. The older apparatus, such as bellows, the *trompe* and other hydraulic blowing machinery may be passed over. Besides the apparatus for moving air, the blast-mains with branchings, dampers, gates, etc., the blast-equalizers and other auxiliary apparatus will receive some notice.

Whatever apparatus may be employed for forcing air into a furnace, it is important that the air be drawn from the outside of the furnace or engine-building and from a cool place. The air-inlet pipe is covered with a coarse screen to prevent solid particles from being drawn in.[2]

[1] Birkinbine, *Cass. Mag.*, 1896–97, XI, 109.

[2] Moeller, Air-filter at intake, *Iron Age*, 1909, LXXXIV, 1402.

314. Air Injectors and Exhausters.—These apparatus act on the same principle as the water-injector (§301) used to feed a boiler. A jet of steam issuing under pressure from a contracted nozzle into a larger pipe, moves the air contained in the latter. Compressed air can take the place of pressure-steam. Fig. 644 represents a longitudinal section through an older form of Koerting jet-blower. The steam enters the blower through a conical nozzle provided with a needle-valve; it then passes through from three to six conical nozzles, which have increasing areas and are connected with a single air-chamber, draws in air through the ports and forces it through the neck of the last nozzle into the discharge-pipe. The latest form is shown in Fig. 645 in which *A* is the air-inlet chamber; *D*, air-discharge; *M*, nozzle; *R*, regulating spindle; and *S*, steam-connection. Table 232 is made up from data given in the catalogue of the Schutte & Koerting Co., Philadelphia, Pa.

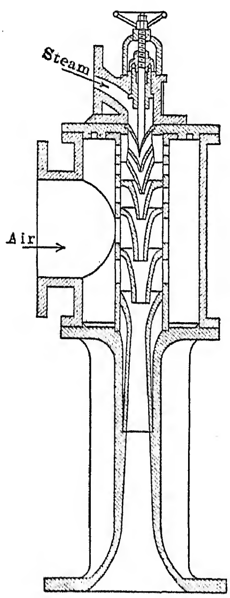


FIG. 644.—Koerting jet-blower, old form.

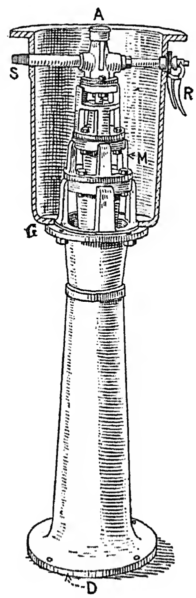


FIG. 645.—Koerting jet-blower, new form.

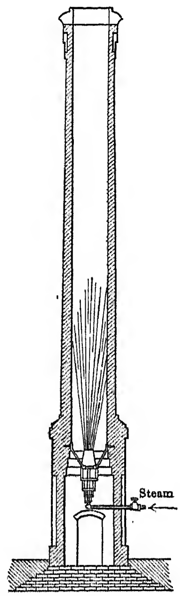
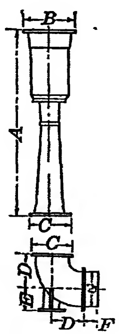


FIG. 646.—Koerting exhauster.

Air-injectors furnish a large volume of moist air at a low pressure. The volume of air given in the table refers to a steam-pressure of 60 lb. per sq. in. and a counter-pressure of 1/2 to 2 in. water. If the air is to be heated by the pressure-steam to 20° C., 100 lb. of air will contain about 1.3 lb. water.

These blowers are especially adapted to furnish undergrate blast to reverberatory furnaces and gas-producers, as the initial heat of the air is utilized and the decomposition of the water vapor into H_2 and CO assists in softening the clinkers. The efficiency reaches 75 per cent. In some instances jet-blowers are used for furnishing surface blast in oxidizing fusions, such as cupelling lead bullion, and fining black copper, etc.

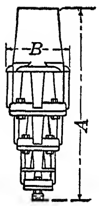
TABLE 232.—KOERTING JET-BLOWER



Size, Diameter of discharge	Blower				Elbow					Capacity in cubic feet per hour	Diameter in inches		Max. coal burnt per hour in pounds
	Inches			Pounds	Inches				Pounds				
	A	B	C		D	E	F	G			Steam	Inlet	
3	19¼	7	5½	45	4¾	3	9	3½	15	10,000	⅜	4	50
4	26¼	8	6½	60	5⅝	3½	9	4½	30	20,000	½	5	100
5	34⅝	11	8½	75	6¼	4	9	5½	38	30,000	½	8	150
6	41⅝	12	9½	110	7	4½	12	6½	45	45,000	¾	9	225
7	46¾	14	11	132	7½	5	12	7½	60	60,000	¾	11	300
8	57¾	15	12	215	8	6	16	8½	80	90,000	¾	12	450
10	65⅜	17½	13	285	10	7	16	10½	150	120,000	1	14	600
12	75⅜	19½	15	396	12	8	16	12¾	200	180,000	1	16	900
14	86¼	21½	17½	530	14	9	16	14¾	260	240,000	1¼	18	1,200
18	111	23	23	980	18	12	16	18¾	340	500,000	1½	24	2,500

A Koerting exhauster or chimney ventilator is shown in Fig. 646. It furnishes a simple means for remedying the lack of draft in a given chimney, due perhaps to faulty construction, to settling of dust in flues or other causes. Table 233 gives the leading facts.

TABLE 233.—KOERTING EXHAUSTER



Size number	Size of stack, inches	Diameter of steam-pipe	Blast nozzle		
			A, inches	B, inches	Pounds
00	Up to 8	$\frac{3}{8}$	$8\frac{1}{4}$	$3\frac{1}{2}$	10
0	9 to 11	$\frac{1}{2}$	$11\frac{1}{2}$	$4\frac{1}{2}$	15
1	12 to 15	$\frac{1}{2}$	$15\frac{1}{4}$	$5\frac{3}{4}$	25
$1\frac{1}{2}$	16 to 19	$\frac{3}{4}$	$18\frac{1}{2}$	$6\frac{1}{2}$	35
2	20 to 23	$\frac{3}{4}$	$21\frac{1}{2}$	$7\frac{1}{2}$	50
$2\frac{1}{2}$	24 to 28	$\frac{3}{4}$	$26\frac{3}{4}$	9	70
3	29 to 33	1	31	10	100
$3\frac{1}{2}$	34 to 40	1	36	12	150
4	41 to 50	$1\frac{1}{4}$	42	14	200
6	51 to 72	$1\frac{1}{2}$	60	18	400
8	73 to 96	2	$82\frac{1}{2}$	$25\frac{1}{2}$	600

315. Fans.[1]—In metallurgical plants there are in operation two types of fans, the disc and the centrifugal. A disc- or propeller-fan, Fig. 647, has propeller-shaped blades which act upon the principle of the inclined plane; the air moving in lines parallel to the axis enters on one side and leaves by the other. It serves for ventilating rooms, for carrying off the moisture-laden air from drying-chambers, for moving the gases in condensing plants, etc. It does efficient work when it acts against slight resistances; it is useless when it has to blow against any pressure. The fan may be driven by a belt or a direct-connected, dust-proof, electric motor. It is made in sizes varying from 18 to 120 in. in diam.; the following six taken from the catalogue of the B. F. Sturtevant Co., Hyde Park, Mass., may serve as examples.

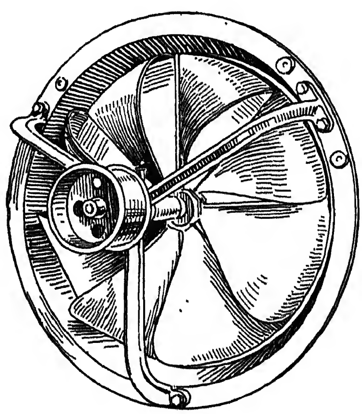


FIG. 647.—Disc-fan.

[1] Snow, *J. Am. Foundrymen's Assoc.*, 1897–98, IV, 129; *Cass. Mag.*, 1906, XXIX, 217, 1906–07, XXXI, 63.

TABLE 234.—STURTEVANT DISC-FANS

Diameter of fan in inches	Revolutions per minute		Pulley		Volume of air per revolution, cubic feet
	Minimum	Maximum	Diameter	Face	
30	325	650	6	$3\frac{1}{2}$	....
36	275	550	7	$4\frac{1}{2}$	....
42	235	470	8	$5\frac{1}{2}$	....
48	200	400	8	$5\frac{1}{2}$	....
54	175	350	9	$5\frac{1}{2}$	....
60	165	330	10	$6\frac{1}{2}$	....

A centrifugal fan[1] (fan- or pressure-blower) is frequently used with furnaces requiring a large volume of air at low pressure (< 16 oz.), *e.g.*, direct-fired reverberatory furnaces with closed ash-pits and cupola furnaces. Left- and right-hand fans are shown in Figs. 648 and 649. They consist of an upright spiral-shaped cast-iron or steel casing with central air-inlet on either side, a single peripheral outlet, and a central horizontal shaft with blades, which revolves at a high speed. The blades on the radial arms, present a flat surface to the air in the casing.

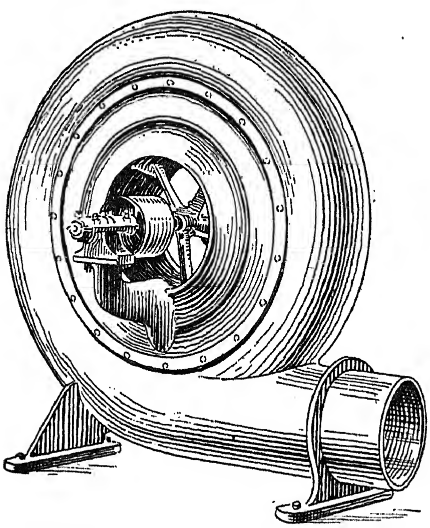


FIG. 648.—Left-hand centrifugal fan.

Fig. 650 represents the fan-wheel of the Buffalo Forge Co. Five plates are joined at the edges to a continuous flange, thus forming five compartments, the area of each of which at the circumference is equal to the area of the opening in the center. The air is given a centrifugal motion by the rapidly revolving fan-wheel and delivered from the tips of the blades to the tangential outlet. The vacuum created in the center is filled by the inrush of fresh air.

Exhausters, constructed on the same general principles, have but a single inlet on the opposite side of the pulley, and are provided with a collar to receive the suction-pipe.

A pressure-blower may be driven by one or two belts, by a direct-connected electric motor, or by a steam turbine. For high speeds, *i.e.*, high pressures, the belt drive used to be the only suitable means of transmitting power, and with this it is necessary to have a counter-shaft in order to obviate the use of an

[1] Buckle, *Proc. Inst. Mech. Eng.*, 1847, p. 3.

abnormally large pulley on the engine- or line-shaft. Smaller sizes have one, larger sizes two pulleys. In recent years the steam turbine direct-connected has replaced the belt-transmission in many instances. For medium pressures,

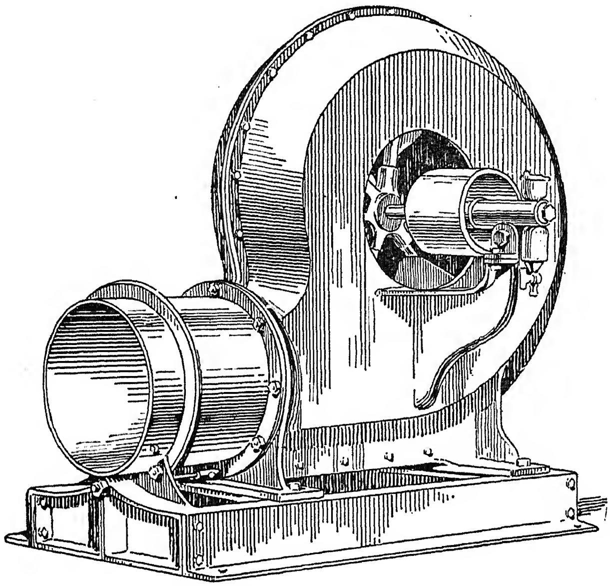


FIG. 649.—Right-hand centrifugal fan.

the belt can be run direct-connected with the pulley of the engine-shaft. A direct-connected electric motor has taken the place of the former high-speed engine; however, the General Electric Co., Schenectady, N. Y.,[1] has on the market a direct-connected machine which running at a speed of 3,450 r.p.m., furnishes 10,200 cu. ft. of air per min. at a pressure of 3 1/4 lb.

The relation[2] that exists between the revolutions of a fan, the volume of air discharged, the pressure created, and the horse-power required is shown in Fig. 651. The curves are based upon three facts, that the volume varies directly as the speed, the pressure as the square, and the horse-power as the cube of the speed. They show that the fan has to be proportioned for the work that it is to do, and that speeding it up to obtain either greater volume or greater pressure causes a tremendous increase in the consumption of power. A fan to show high mechanical efficiency (about 40 per cent.) ought to be driven up to its maximum

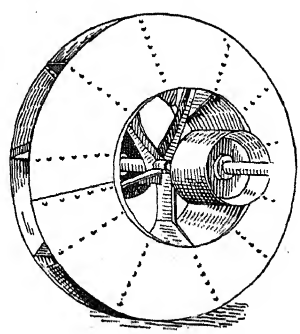


FIG. 650.—Fan-wheel of centrifugal fan.

[1] *Iron Age*, 1908, LXXXII, 1440.

[2] Snow, "Mechanical Draft," p. 203.

capacity. With a given pressure, which means a certain peripheral speed, if great quantities of air are required, it is advisable to use several narrow fans instead of one wide fan. This arrangement makes it possible to meet conditions in which variations in quantity of blast arise, and maintain at the same time a high efficiency. When several fans of different size and speed are in operation, a higher efficiency is obtained, if the blast of each is conducted separately to the furnace where it is to be used, than if they all deliver into a single main with branches. The experiments of Howe[1] have shown that for a given speed of fan, any diminution in the size of the blast-orifice decreases the consumption of power and at the same time raises the pressure of the blast, but

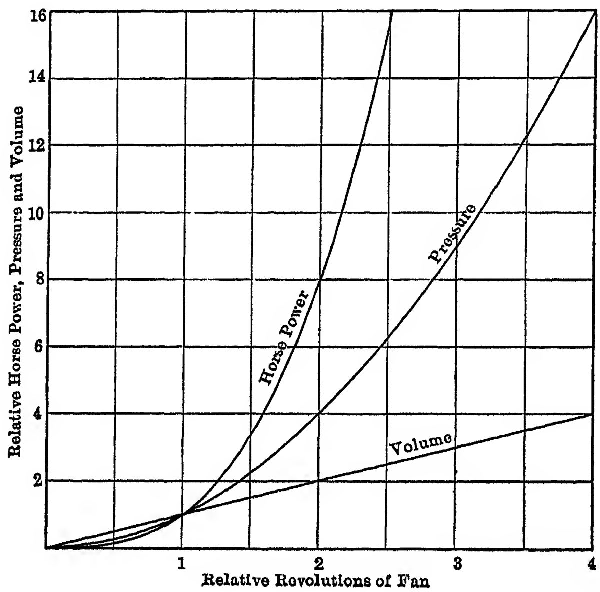


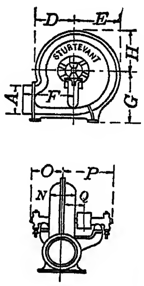
FIG. 651.—Revolutions and power of fan, volume and pressure of blast.

increases the consumption of power per unit of orifice for a given pressure of blast. When the orifice has been reduced to the maximum normal size for any given fan, further diminishing it causes but slight elevation of the blast-pressure. And when the orifice becomes comparatively small, further diminishing it causes no sensible elevation of the blast-pressure, which remains practically constant, even when the orifice is entirely closed.

Table 235 gives the leading dimensions and the work of some of the Monogram blowers of the B. F. Sturtevant Co. as applied to boilers; it is based upon the operation of the blower at 1 1/2-oz. pressure at the fan-outlet, and a combustion rate of 15 lb. of coal per hr. per sq. ft. of grate area.

[1] *Tr. A. I. M. E.*, 1882, x, 489.

TABLE 235.—STURTEVANT FAN-BLOWERS



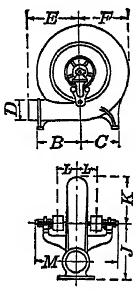
No.	Dimensions, inches										Area of grate in square feet	Revolutions per minute of blower necessary to produce the required pressure
	A	D	E	F	G	H	N	O	P	Q		
1	$5\frac{3}{4}$	$7\frac{3}{4}$	9	$10\frac{7}{8}$	$10\frac{3}{4}$	$8\frac{3}{8}$	$4\frac{1}{2}$	$6\frac{1}{2}$	10	$4\frac{1}{8}$	6	2,218
2	$7\frac{1}{2}$	$9\frac{1}{4}$	$10\frac{3}{4}$	$13\frac{1}{2}$	$12\frac{1}{2}$	10	$5\frac{5}{8}$	$7\frac{3}{8}$	$11\frac{1}{2}$	$5\frac{1}{8}$	8	1,969
3	9	$10\frac{3}{4}$	$12\frac{3}{4}$	$15\frac{5}{8}$	$14\frac{3}{4}$	$11\frac{3}{4}$	$6\frac{5}{8}$	$8\frac{1}{4}$	$13\frac{1}{4}$	$5\frac{7}{8}$	11	1,635
4	$10\frac{5}{8}$	$12\frac{1}{2}$	15	$17\frac{3}{4}$	$17\frac{1}{4}$	$13\frac{3}{4}$	$7\frac{3}{4}$	$9\frac{1}{4}$	$15\frac{1}{4}$	$6\frac{3}{4}$	16	1,378
5	$12\frac{1}{4}$	$14\frac{7}{8}$	$17\frac{5}{8}$	$20\frac{1}{4}$	20	$16\frac{1}{4}$	9	$10\frac{3}{8}$	$17\frac{3}{4}$	8	20	1,206
6	$14\frac{5}{8}$	17	$20\frac{1}{2}$	$22\frac{3}{4}$	$23\frac{1}{8}$	$18\frac{3}{4}$	$10\frac{1}{2}$	$11\frac{7}{8}$	$20\frac{1}{4}$	$9\frac{1}{2}$	35	1,026
7	$16\frac{5}{8}$	20	24	26	28	22	$13\frac{3}{4}$	$14\frac{1}{2}$	$23\frac{3}{4}$	$11\frac{3}{4}$	50	893
8	$18\frac{3}{4}$	$23\frac{1}{8}$	$27\frac{1}{8}$	$29\frac{1}{4}$	$31\frac{1}{2}$	$25\frac{1}{8}$	$16\frac{1}{4}$	$17\frac{1}{2}$	28	$13\frac{1}{2}$	70	780
9	$21\frac{3}{4}$	26	$31\frac{1}{2}$	33	37	$28\frac{3}{4}$	19	$19\frac{1}{2}$	31	$15\frac{3}{8}$	110	603
10	$24\frac{5}{8}$	$31\frac{1}{2}$	$37\frac{1}{2}$	$37\frac{1}{2}$	43	$34\frac{1}{2}$	$21\frac{1}{4}$	$21\frac{3}{4}$	$38\frac{3}{4}$	$19\frac{3}{8}$	150	524

Tables 236 and 237 give similar data about the Buffalo steel pressure-blowers as applied to cupola work.

Turbo-blowers[1] are multi-stage high-pressure fan-blowers constructed on the same principle as compound turbine pumps (§299). From four to six centrifugal pressure-blowers on a single shaft are connected in series; the discharge from one fan-wheel forms the feed of the next following. This arrangement has opened the field for fan-blowers so far occupied by blowing engines. It permits the use of direct-connected electric motors, as the usual standard speed for medium pressure-blowers need not be exceeded. The first experiments with turbo-blowers were conducted in 1900 by Rateau. In 1908 there were in operation 30 turbo-blowers furnishing blast to foundry-cupolas, copper and iron blast-furnaces. The maximum air capacity was 42,000 cu. ft. per min.; the greatest pressure, 18 lb. per sq. in., the shaft making 4,000 r.p.m. At the Mt. Lyell Mining Co.'s plant, Tasmania, a blower driven by a steam turbine furnishes 18,000 cu. ft. air per min. at a pressure of 5 lb. per sq. in. to a copper blast-furnace, the shaft making 4,000 r.p.m. The iron blast-furnace of the Gute Hoffnungs works has a 4-stage blower driven by an electric motor, which furnishes 35,700 cu. ft. of air per min., at a pressure of 10 lb. per sq. in., the shaft making 1,500 r.p.m. The General Electric Co., Schenectady, N. Y., has in the market a turbo-blower delivering 3,000 cu. ft. of air per min. at 15 lb. pressure, when the machine is running at 3,750 r.p.m.

[1] Rateau, Rev. Mét., 1907, IV, 751; *Bull. Soc. Ind. Min.*, 1908, IX, 569; *Rev. Univ.*, 1908, XXIV, 137; Parsons, *Iron Age*, 1907, XC, 485; Buffalo Two-stage Blowers, *Iron Age*, 1907, LXXIX, 29; 1908, LXXXI, 1855; *Stahl u. Eisen*, 1908, XXVIII, 73 (Langer), 1729 (Regenbogen, also *Iron Age*, 1909, LXXXIII, 1846); 1909, XXIX, 493 (Naville), 544 (Scherbius); Rice, *Iron Age*, 1911, LXXXVII, 538; *Iron Trade Rev.*, 1911, XLVIII, 459; Zur Nedden, *Eng. Mag.*, 1912, XLIII 587, 699; Johnson, *Met. Chem. Eng.*, 1911, IX, 393.

TABLE 236.—BUFFALO STEEL PRESSURE-BLOWERS

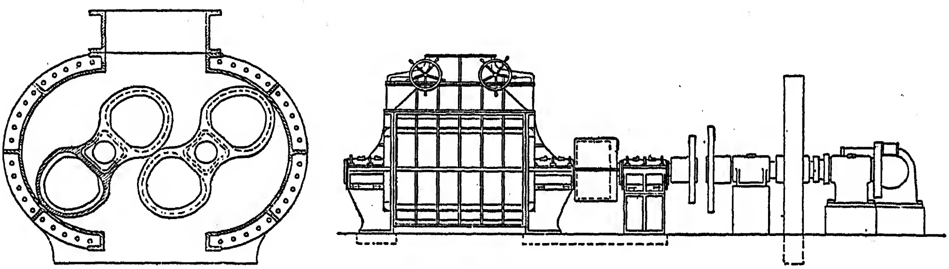


Size number	Dimensions, inches										
	B	C	D	E	F	J	K	L	M	Pulley	
										Diameter	Face
.4	11⅝	5½	5	13⅛	10¼	13½	10	5	25¼	4	3
5	12½	5¾	5⅜	14¼	12	14¼	11½	4⅝	24½	4¼	3
6	14	7½	6¼	15¾	13¼	16¾	13	5⅝	27¾	4½	3½
7	14½	10½	7¼	16¼	14¾	19¼	14	6½	33¾	5	4½
8	17	12½	8¾	19	17	21½	16½	9	40	6	4½
9	19½	16	10	22	20	25	19	9⅜	41	7	5
10	23¾	20	12¼	28	25½	30½	25½	10½	45¼	8	5¾
11	27½	25	14½	28	30½	36½	29	12	50¾	8½	6½
11½	30	28	16½	33	36	41⅝	34	12⅜	53	10	7
12	30	28	18	33	36	41⅝	34	12⅜	53	10	8

TABLE 237.—SPEEDS AND CAPACITIES FOR CUPOLA SERVICE
(Buffalo Steel Pressure-blower)

Number of blower	Inside diameter of cupola	Pressure in ounces	Revolutions per minute	Melting capacity in pounds per hour	Cubic feet of air required per minute
4	18	8	4,732	1,545	666
5	23	8	4,209	2,321	773
6	27	8	3,660	3,093	951
7	32	8	3,244	4,218	1,486
8	37	8	2,948	7,500	2,199
9	42	10	2,785	10,000	3,000
10	48	10	2,195	14,000	4,500
11	60	12	1,952	22,000	6,500
11½	70	12	1,647	30,000	8,500
12	84	12	1,625	40,000	10,000
4	18	9	5,030	1,647	717
5	23	10	4,726	2,600	867
6	27	10	4,108	3,671	1,067
7	32	10	3,642	5,000	1,668
8	37	10	3,310	8,800	2,469
9	42	12	3,260	12,000	3,000
10	48	12	2,413	17,000	5,000
11	60	14	2,116	26,000	8,000
11½	70	14	1,797	35,000	10,000
12	84	14	1,775	45,000	12,000

316. Rotary Blowers.—Two similar or dissimilar impellers attached to parallel shafts revolve in opposite directions in a casing having an air-inlet and outlet. The revolving bodies being in tangential contact with one another and with the casing, draw in with every revolution a given volume of air on one side of the casing and discharge it by displacement on the other into a blast-pipe. In a rotary blower, *i.e.*, a positive displacement-machine, the amount of air supplied and the horse-power required are proportional to the speed and pressure under which it works. A blower is run to make from 95 to 270 r.p.m.; the rapid movement of the impellers overcomes the small fluctuations due to leakage and slip, and gives a practically continuous air-blast. The volume of air furnished is large, and the pressure attained reaches 3 lb. per sq. in. For higher pressures blowing engines are necessary.[1] The power[2] required is assumed to be 5 h.p. for every 1,000 cu. ft. free air discharged at 1 lb. pressure; the mechanical efficiency is about 45 per cent.,[3] and the discharge efficiency



Cross-section. Side-elevation.

FIGS. 652 and 653.—Roots blower.

about 75 per cent. The cost of the apparatus is about 1/6 that of a blowing engine; it has become the leading blowing machine for lead, copper and similar medium-pressure blast-furnaces, and furnishes blast to foundry cupolas.[4]

The leading forms of rotary pressure-blowers are the Roots, Cycloid, Baker and Sturtevant.

The Roots blower[5] came into use in 1866.[6] A side-elevation of a single-geared direct-driven machine with top-discharge and double-acting blast-gate is given in Fig. 652; a cross-section in Fig. 653, and a perspective view of the impeller in Fig. 654. The outlines of the impellers form epicycloidal curves. The impellers are so constructed as to revolve as closely as possible to each other

[1] Discussions of blowers and blowing engines, *Min. Ind.*, 1901, x, 432.

[2] Field, *Foundry*, 1904, xxv, 229.

Snow, *op. cit.*, 1907, xxi, 181; *Iron Age*, 1907, lxxxi, 21.

[3] Ledebur, A., "Die Bearbeitung der Metalle auf Mechanischem Wege," Vieweg, Brunswick, 1879, p. 81.

[4] Caldwell, *J. Am. Foundrymen's Assn.*, 1898, iv, 200.

[5] Oerzen, *Stahl u. Eisen*, 1904, xxiv, 888.

Rice, *Eng. Min. J.*, 1909, lxxxvii, 352.

Austin, *Min. Sc. Press*, 1909, xcix, 432.

[6] Editor, *Engineering*, 1867, iii, 614; iv, 508.

and to the casing without touching at any point; some volumetric efficiency is sacrificed in order to simplify the adjustment and to give latitude for wear. As the impellers are always in clearance contact with one another and the casing, they form enclosed spaces bordered by the lobes and the casing. By the rotation of the lobes in opposite directions these spaces change their locations. At every revolution a certain volume of air is drawn from below into the casing, enclosed between the lobes and the casing, and then discharged above. Fig. 652 represents a single-geared blower driven direct from a Corliss

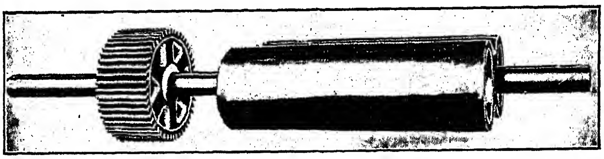


Fig. 654.—Impeller Roots blower.

engine; this type is often driven from an electric motor by means of a belt. Double-geared blowers have the advantage that only part of the load is transmitted through the gears; they have been successful only with electric motors which respond quickly to any changes in the load and divide it so as to keep the tension on the belts equal. The successful operation of a blower depends largely upon keeping the shafts of the impellers in correct alignment. Table 238 gives the chief data in regard to the common sizes of the machine.

Table 238.—Roots Positive Pressure Blowers

No.	Displacement per rev., cu. ft. air	Rev. per minute	H.p. per 1,000 cu. ft. displacement per minute at 2.5 lb. pressure	Dis-charge diam., inches	Pulley		Machine	
					Diam., inches	Face, inches	Length by width by height, inches	Weight, pounds
4	12.2	270	41	14	32	6	97⅞×41½×42⅛	5,000
5	18.27	230	51	16	36	8	115⅝×49 ×48⅛	8,500
5½	26.1	200	62	18	42	10	126½×54½×54½	10,000
6	35.7	190	80	20	48	12	140⅝×59 ×57	13,500
6½	47.5	175	95	22	66	12	159⅝×65½×63¾	16,000
7	56.9	160	110	24	70	14	168½×73 ×55¾	26,000
7½	81	145	140	24	84	16	185 ×79½×62	36,000
8	111.2	135	175	30	106	18	202¾×90 ×67½	40,000
8½	147.9	125	220	30	120	22	220⅛×100×77½	52,000
9	192	120	260	36	136	24	....	70,000
9½	244	110	320	36	160	26	....	85,000
10	304.9	100	360	42	184	28	....	115,000
11	401	95	450	42	216	32	....	130,000

The Connersville Blower.—This machine which has found much favor among smelter men is similar to the Roots; it differs from it in details such as journals, bearings, gears, etc. The Tables 239 and 240 give the chief data in regard to the blowers used in blast-furnaces and cupolas.

TABLE 239.—CONNERSVILLE SMELTER BLOWERS

Displacement per rev., cu. ft. air	Rev. per. min.	Discharge diam., in.	Pulley		Machine, length by width by height, in.	Power
			Diam., in.	Face, in.		
13	250	14	42	7	$99\frac{3}{4}\times45\frac{1}{2}\times39$	5 h.p. per
17	225	16	42	8	$117\times48\times41$	1,000 cu.
24	200	18	48	10	$123\times54\frac{1}{2}\times44$	ft. dis-
33	190	20	60	12	$147\times60\times49\frac{1}{2}$	place-
45	180	20	66	14	$160\frac{1}{4}\times65\frac{1}{2}\times55$	ment
57	170	24	72	16	$170\frac{3}{4}\times70\frac{1}{2}\times59$	per min.
65	160	24	84	16	$181\times76\times63$	at 1 lb.
84	150	27	84	20	$192\times81\frac{1}{2}\times67$	pressure.
100	140	30	96	20	$210\frac{1}{2}\times87\frac{1}{2}\times70$	
118	130	30	120	20	$213\frac{3}{4}\times94\times76$	

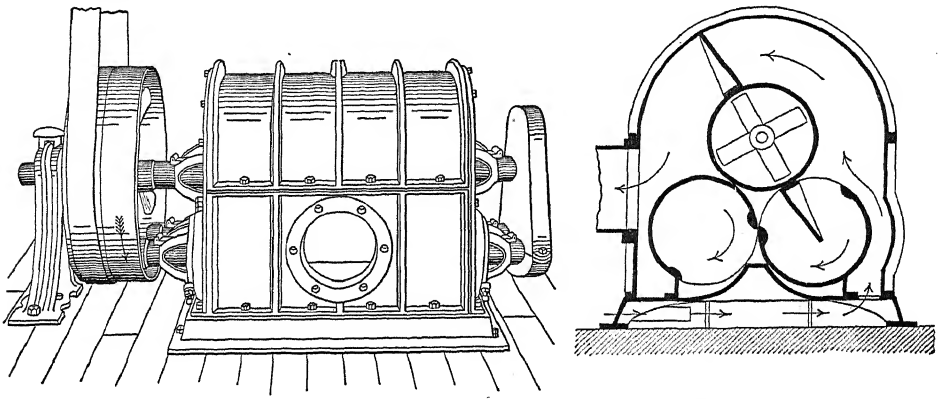
TABLE 240.—CONNERSVILLE CUPOLA BLOWERS

Displacement per rev., cu. ft. per minute	Diam., cupola, inches	Tons pig iron melted per hour at rev. per min.			Discharge diam., inches	Pulley		Machine, length by width by height, inches	Power
						Diam., inches	Face, inches		
$1\frac{1}{2}$	18–20	$\frac{1}{2}$ at 200	1 at 400	....	6	20	4	$49\frac{1}{2}\times16\times20\frac{1}{2}$	
3.3	24–27	1 at 175	2 at 335	....	8	24	4	$57\times18\times25\frac{1}{2}$	
6	28–32	2 at 185	3 at 275	....	10	27	4	$65\frac{3}{4}\times24\times29$	
10	32–38	4 at 200	5 at 250	....	12	30	5	$76\frac{3}{4}\times28\times34$	5 h.p. per
13	32–40	4 at 150	5 at 190	$6\frac{1}{2}$ at 275	14	34	5	$99\frac{3}{4}\times45\frac{1}{2}\times39$	1,000 cu.
17	36–45	5 at 150	$6\frac{1}{2}$ at 205	$8\frac{1}{2}$ at 250	16	36	6	$117\times48\times41$	ft. dis-
24	42–54	8 at 166	10 at 200	12 at 240	18	42	7	$123\times54\frac{1}{2}\times44$	place-
33	48–60	10 at 150	12 at 180	14 at 210	20	42	8	$147\times60\times49\frac{1}{2}$	ment per
45	54–66	12 at 135	15 at 165	18 at 200	20	48	10	$160\frac{1}{4}\times65\frac{1}{2}\times55$	minute
57	60–72	15 at 130	18 at 155	21 at 185	24	54	10	$170\frac{3}{4}\times70\frac{1}{2}\times59$	at 1 lb.
65	66–84	18 at 140	21 at 160	24 at 185	24	60	12	$181\times76\times63$	pressure.
84	72–90	21 at 125	24 at 145	27 at 160	27	66	14	$192\times81\frac{1}{2}\times67$	
100	84–96	24 at 120	27 at 135	30 at 160	30	72	14	$210\frac{1}{2}\times87\frac{1}{2}\times70$	
118	60–66[1]	27 at 115	30 at 130	33 at 140	30	72	16	$213\frac{3}{4}\times94\times76$	

[1] Two cupolas.

The Baker Blower, 20 years ago, was the principal machine for supplying copper and lead blast-furnaces with air; it has been replaced to a considerable extent by the Roots and Connersville machines. Fig. 655 *a* and *b* give a perspective view and a cross-section. The cast-iron casing, with air-inlet at the bottom and -outlet at the side, has three revolving drums; the shaft of the upper drum carries at one end the driving-pulley and at the other a spur-gear which meshes with the smaller gears of the lower drums and causes these to revolve twice as fast. The upper drum has two arms, which draw in air at the lower

right hand and deliver it to the outlet at the left. Each of the lower drums has a slot for the passage of these arms and acts as a valve to prevent air from escaping or returning. However, as soon as the upper arm passes the upper edge of the outlet, the compressed air in the latter fills the upper part of the casing and the drum near the outlet. This decreases the efficiency of the machine.



FIGS. 655a and 655b.—Baker positive pressure blower

TABLE 241.—BAKER POSITIVE PRESSURE-BLOWERS [1]

No.	Displacement per rev., cu. ft.	Minimum speed	Maximum speed	Diam. of discharge pipe	Size of pulley	Weight, pounds	Horse-power required
¼	1	....	....	3	12× 2½	350	....
½	2	....	....	5	20× 4	750	1
1	3½	168	240	6	24× 6	1,200	4
2	6	123	200	7½	30× 6	1,700	4½
3	9	112	180	8¾	36× 6	2,250	7½
4	12	106	165	11¾	40× 7	3,000	9½
4½	16½	88	150	11¾	48× 7	3,700	12½
5	25	87	130	15	54× 7½	5,000	15
5½	30	100	125	17	60× 7½	6,350	17
6	45	80	120	21	62×12	9,700	25
7	60	85	120	24	72×13	12,800	28
7½	72	88	120	24	72×15	15,000	32

The Sturtevant impeller blower[2] is a complicated machine and is represented in Figs. 656 to 662. Fig. 656 shows the cast-iron shell, consisting of two partially intersecting cylinders, to be closed at the ends by four cover-plates which carry the lower journal-boxes of the two revolving shafts. The lower shaft is

[1] This blower is also sometimes connected to an engine direct, and mounted on same bed-plate.

[2] *Eng. Min. J.*, 1906, LXXXI, 365.

connected with the power and drives the upper shaft by gearing, both making the same r.p.m. The figure gives the air-outlet, opposite which is the inlet. In Fig. 657, the lower cylinder of the casing contains the impeller (a central web keyed to the steel driving-shaft carrying three diamond-shaped bars) and the fixed core, *G*, Figs. 658, 659 and 662, which forms part of the cover; the upper cylinder holds the idler (three hollow vanes cast in one piece with shaft).

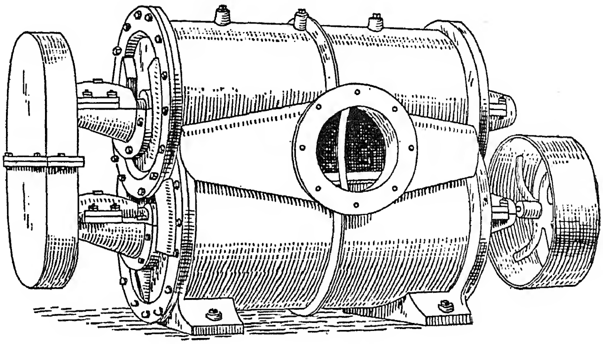


FIG. 656.—Sturtevant impeller blower, perspective view.

The operation, Figs. 659–662, is as follows: In Fig. 659, the entering air fills chambers *X* and *D*, while chambers *E* and *Z* are discharging into the delivery-pipe; just before reaching this position, the pressure of the air held in the pocket, *Y*, has been released through the leakage-passage, *N* (see below). After the impeller and idler have passed from positions held in Fig. 659 to those of Fig. 660, air enters *F*; chamber *X* is shut off from the inlet and placed in communication with *N*; chamber *D* is also cut off from the inlet and bordered by blades *A* and *B*; chambers *E* and *Z* are still in connection with the delivery-pipe and increase the pressure in *Y* through leakage-passage *O*. In position, Fig. 661, the principal change is that *E* alone is discharging, while pocket *Z* filled with pressure-air gives up part of this through leakage-passages *N* and *O* to pockets *X* and *Y*. When the fourth position, Fig. 662, has been reached, pockets *F* and *Z* will be filling, *E* and *Y* discharging, and some compressed air flowing through leakage-passage *M* into pocket *D*. Upon leaving position 4, the impeller and idler quickly reach positions similar to those shown in Fig. 659. Clearance to the amount of 1/8 in. with small, and of 1/2 to 3/4 in. with large blowers is given the moving parts and the casing. Table 242 gives some of the leading data.

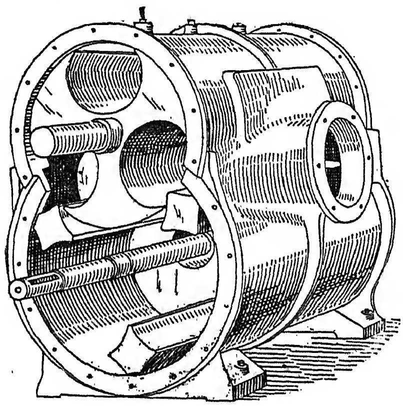


FIG. 657.—Sturtevant impeller blower casing, with ends and fixed central core removed.

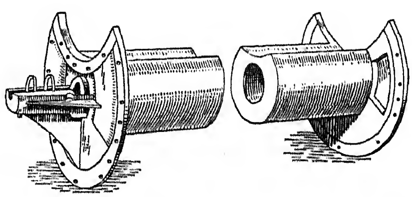
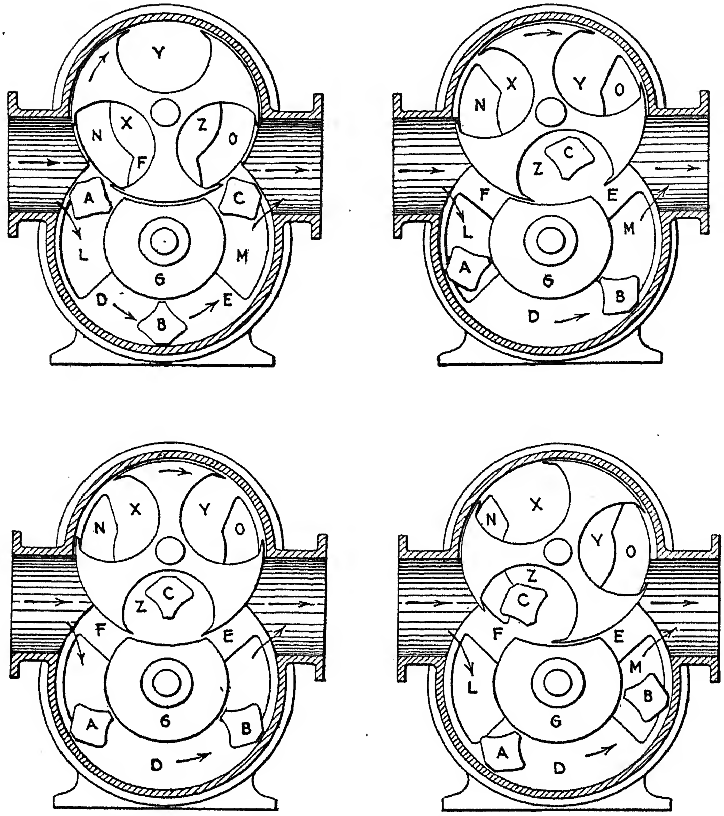


FIG. 658.—Sturtevant impeller blower fixed central core and end-plate of lower cylinder.



FIGS. 659 to 662.—Sturtevant impeller blower, action of blower.

TABLE 242.—STURTEVANT IMPELLER BLOWERS

No.	Capacity, cu. ft. per minute, ½ lb. pressure	R.p.m.	Inlet and discharge, inside diam. inches	Pulley ½ lb. press		Pulley 3 lb. press		Pulley 5 lb. press	
				Diam., inches	Face, inches	Diam., inches	Face, inches	Diam., inches	Face, inches
4	325–560	350–565	6	12	3	12	5	12	6
5	560–1,030	300–475	8	18	3	18	5	18	6
6	1,030–1,540	290–415	10	20	5	20	8	20	10
7	1,540–2,300	280–410	10	24	6	24	8	24	12
8	2,300–3,300	265–375	12	24	8	24	12	24	16
9	3,300–4,700	250–350	16	32	8	32	12	32	16
10	4,700–6,000	260–330	16	36	8	36	12	36	16
11	6,000–8,500	220–310	20	44	10	44	15	44	18
12	8,500–11,300	190–250	24	60	10	60	15	60	20
13	11,300–15,500	190–260	30	66	12	66	18	66	24

As applied to furnishing blast for cupolas, Table 243 gives some information. The figures in the third column are based upon the assumption of 30,000 cu. ft. of air being required per ton of pig iron per hr.

TABLE 243.—STURTEVANT IMPELLER BLOWERS IN CUPOLA WORK

Tons of iron per hour	Usual inside diameter of cupola for the work, inches	Cubic feet of air per minute	Number of blower	Revolutions per minute
1	23	500	4	450
2	27	1,000	5	430
3	30	1,500	6	380
4	32	2,000	7	350
5	36	2,500	8	280
6	39	3,000	8	330
7	42	3,500	9	260
8	45	4,000	9	290
9	48	4,500	9	330
10	54	5,000	10	270
11	54	5,500	10	300
12	60	6,000	10	325
13	60	6,500	11	230
14	60	7,000	11	250
15	66	7,500	11	270
16	66	8,000	11	290
17	66	8,500	11	305
18	72	9,000	12	200
19	72	9,500	12	215
20	72	10,000	12	225
21	78	10,500	12	235
22	78	11,000	12	245
23	78	11,500	12	255
24	84	12,000	13	200
25	84	12,500	13	210
26	84	13,000	13	215
28	90	14,000	13	235
30	90	15,000	13	250

317. Blowing Engines.[1]—Of the various types of blowing engines, only the

[1] Lohse, History, *Stahl u. Eisen*, 1911, XXXI, 173, 348, 429.
Gordon, *Am. Inst. Mech. Eng.*, 1891, XII, 676.
Klein, *Stahl u. Eisen*, 1891, XI, 98.
Kennedy, *Tr. A. I. M. E.*, 1893, XXII, 709.
Ihering, *Oest. Jahrb.*, 1899, XLII, 177.

double-acting piston-blowers are in use at present. They serve for furnishing large volumes of air at high pressures to iron blast-furnace, steel and matte converters. In an iron blast-furnace from 5 to 6 tons of air are necessary for 1 ton of pig iron; a coke furnace may produce 500 tons of pig iron in 24 hr.; or from 60 to 65 cu. ft. air will burn 1 lb. coke, and on the average 1 ton coke is required for 1 ton pig, the blast-pressure in a coke-furnace, usually from 12 to 16 lb., may have to be raised to 30 lb. The engine has to work constantly under irregular strains with little time for examination and repair; it therefore must be built strongly. In a Bessemer converter, which works intermittently, the volume of air is much smaller than in a blast-furnace, but the pressure is higher, viz., 25 to 30 lb. per sq. in. In a matte converter one has to deal with smaller units than in a steel converter.

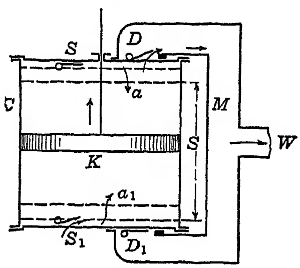


FIG. 663.—Operation of double-acting piston blowing engine.

The operation of a double-acting-piston blowing engine is shown diagrammatically in Fig. 663. In the cylinder C, provided with air-inlet valves S, S' and outlet valves DD', the solid piston, K, is actuated by an engine and receives a reciprocating motion. With the stroke in the direction of the arrow, outside air is drawn in through S' and compressed air expelled through S into branch M and main W; upon reversing the stroke, S and D' are opened, while S' and D are closed. The cylinder is double-acting, as at every stroke outside air is drawn in and compressed air forced out. As the speed of the piston is greatest at the center of the cylinder and zero at the ends, the blast is not uniform like that delivered by a fan or a rotary blower, and the pulsations have to be equalized. This is done with blast-furnace work by compound engines or by having large receivers (equalizers, regulators); hot-blast stoves often fulfill the purpose; and for converting work accumulators are often used. Blowing engines have been driven until recently by steam-engines, water-power being used only occasionally. At present gas-engines driven by washed iron blast-furnace gases have replaced many steam-engines. This innovation was started in Europe, and in this country the Lackawanna Steel Co. of Buffalo, N. Y., was the pioneer.[1]

1. Parts of the Blowing Engine.—The leading parts of a blowing engine are the cylinder, the piston and the valves. The cylinder, or tub, was formerly made of cast-iron, but in most engines this is now replaced by cast steel. The cylinder is built without any provision for reducing the rise of temperature of the air, due to the partial conversion into heat of the work done in compression.

Snyder, *Proc. Eng. Soc. West. Pa.*, 1900, XVI, 190.
Hilgenstock, *Stahl u. Eisen*, 1902, XXII, 203.
Roberts, *Proc. Engl. Inst. Mech. Eng.*, 1906, p. 375; *Iron Age*, 1906, LXXVIII, 1082.
Hauer, J., Ritter von, "Die Hüttenwesen-Maschinen, Felix, Leipsic, 1876; supplement, 1887.
Ihering, A. von, "Die Gebläse," Springer, Berlin, 1913.
Düsseldorf Mining and Metallurgical Exposition, *Stahl u. Eisen*, 1910, XXX, 1043.

[1] *Iron Age*, Jan. 7, 1904, p. 49.

In the heads are the openings for the valves, the inlet-valves occupy from 1/4 to 1/5 of the surface, the outlet-valves from 1/6 to 1/10. The length of the cylinder must be equal to the length of stroke and thickness of piston plus the clearances at the ends, *i.e.*, the distances, *a* and *a'*, Fig. 663, between the piston and the heads of the cylinder when the former is at the end of a stroke. This space, while necessary to prevent shocks, is made as small as possible, and usually does not exceed 5 per cent. of the volume of the cylinder. The diameter varies with the intended capacity. With iron blast-furnace engines, the length of the air-cylinder ranges from 36 to 60 in., and the diameter from 66 to 90+ in.; common combinations are 36×66, 48×66, 48×72, 48×84, 58×84, 60×84, 66×84, 60×90 in., etc. With steel converter blowing engines the length of the cylinder is similar to that of blast-furnace engines, but the diameter is smaller, *e.g.*, 48×54, 60×54, 48×58, 60×58 in., etc.

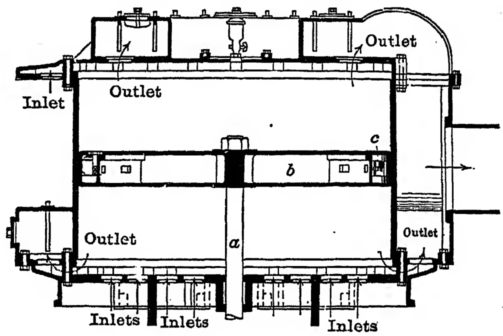


FIG. 664.—Details of blast cylinder, 84-in. diam., 60-in. stroke.

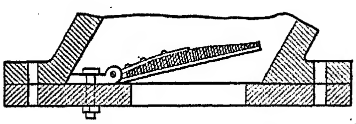


FIG. 665.—Flap- or clack-valve.

The piston, Fig. 664, consists essentially of three parts, the solid rod, *a*, the hollow head, *b*, which are of steel, and the packing, *c*, which may be of wood, leather, hemp or metal (brass, steel). The piston-speed is governed to a certain extent by the character of the air-valves; with automatic valves it is usually given as 400 ft. per min., which presupposes the engine to make 30–40 r.p.m.; with mechanical (positive) valves the number of revolutions has been doubled.

The valves[1] used are of two kinds; automatic and mechanical. The automatic, or self-acting valve, is one which is operated by the flow of air; it is the older, is simple and cheap, but is likely to leak air. One form is that of the flap- or clack-valve, Fig. 665, a piece of leather backed by iron and hinged on one side; another, that of the straight-lift valve,[2] Fig. 664, also of leather or vulcanized rubber and backed by iron, which rises from

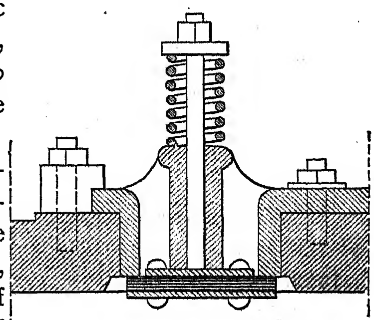


FIG. 666.—Poppet-valve.

[1] *Stahl u. Eisen*, 1897, XVII, 941 (Hörbiger); 1066 (Daelen); 1898, XVIII, 21 (Hörbiger); 1908, XXVIII, 518 (Kiesselbach); *Tr. Am. Inst. Mech. Eng.*, 1899, XX, 967; *Iron Age*, July 13, 1899, p. 9 (Gordon).

[2] Gordon, *Tr. Am. Inst. Mech. Eng.*, 1891, XII, 676.

its seat. The outlet-valve is usually heavier than the inlet-valve, and often of steel.

These valves have been replaced in part by others held in their seats by springs, as *e.g.*, the poppet-valve,[1] Fig. 666; this is opened by the pressure of the air and closed quickly by the spring upon the reversal of the stroke. The

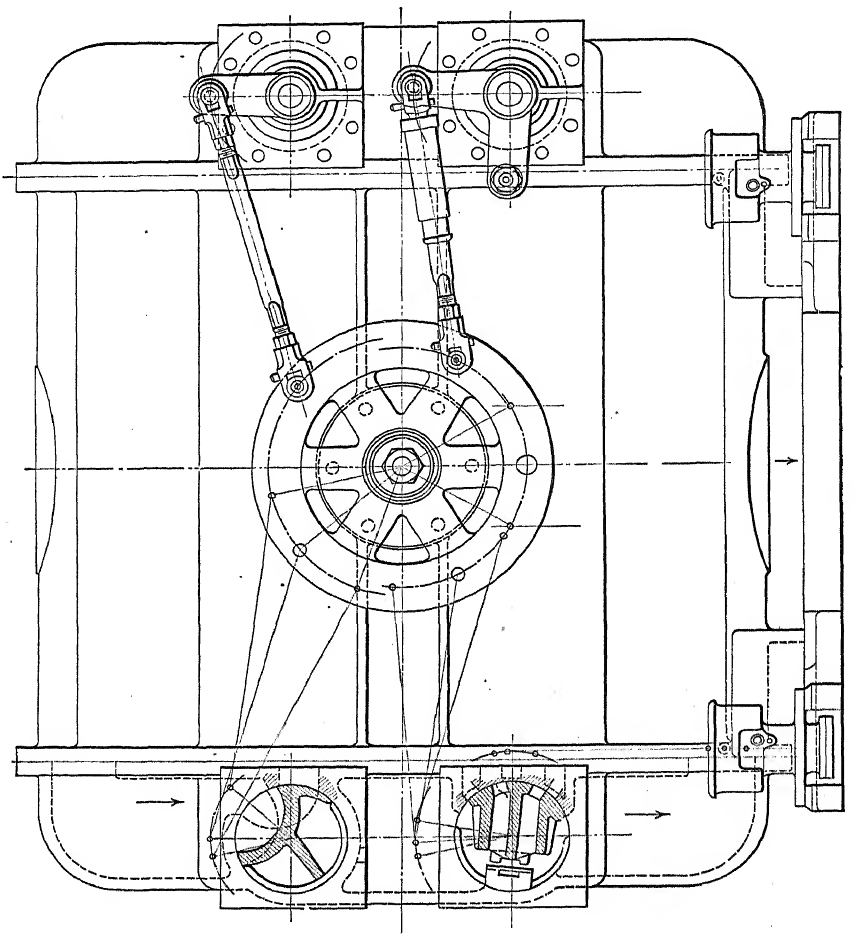


FIG. 667.—Rotary valves.

disadvantage is that the ingoing air-current must overcome the resistance of the spring.

Mechanical valves are rotary, or have the form of a piston or a slide; they are controlled by a mechanism connected with the steam-engine and can be opened and closed just at the desired moment. They permit running the blowing engine at an increased speed and give at the same time the best air-efficiency.

The rotary valves shown in Fig. 667 are built by the E. P. Allis Co., Mil-

[1] Schwanecke, *Stahl u. Eisen*, 1910, xxx, 623.

waukee,[1] Wis., for a 60×60-in. air-cylinder of a bessemer blowing engine. They are driven from a wrist-plate; the inlet-valves are plain rotary, the outlet-valves are triple-port. The Gordon[2] piston-valves, two in each head, are shown in Fig. 668. The four are driven from a single eccentric by two rock-arms; they are in balance as to pressure and weight of parts so that a high speed of engine is permissible.

The slide-valves of the Southwark Foundry & Machine Co., Philadelphia, Pa., are of the gridiron-type. They have two forms; the rectangular is shown

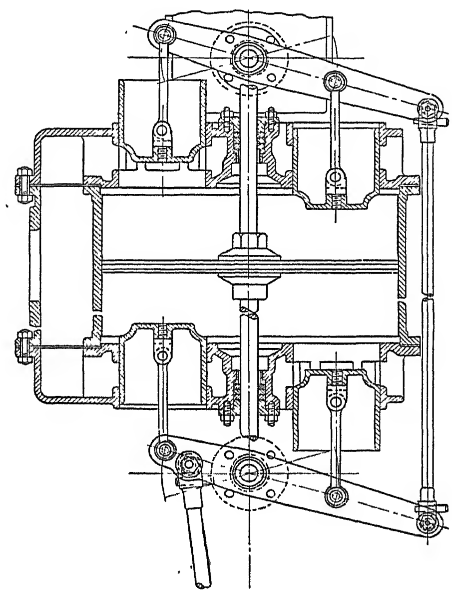


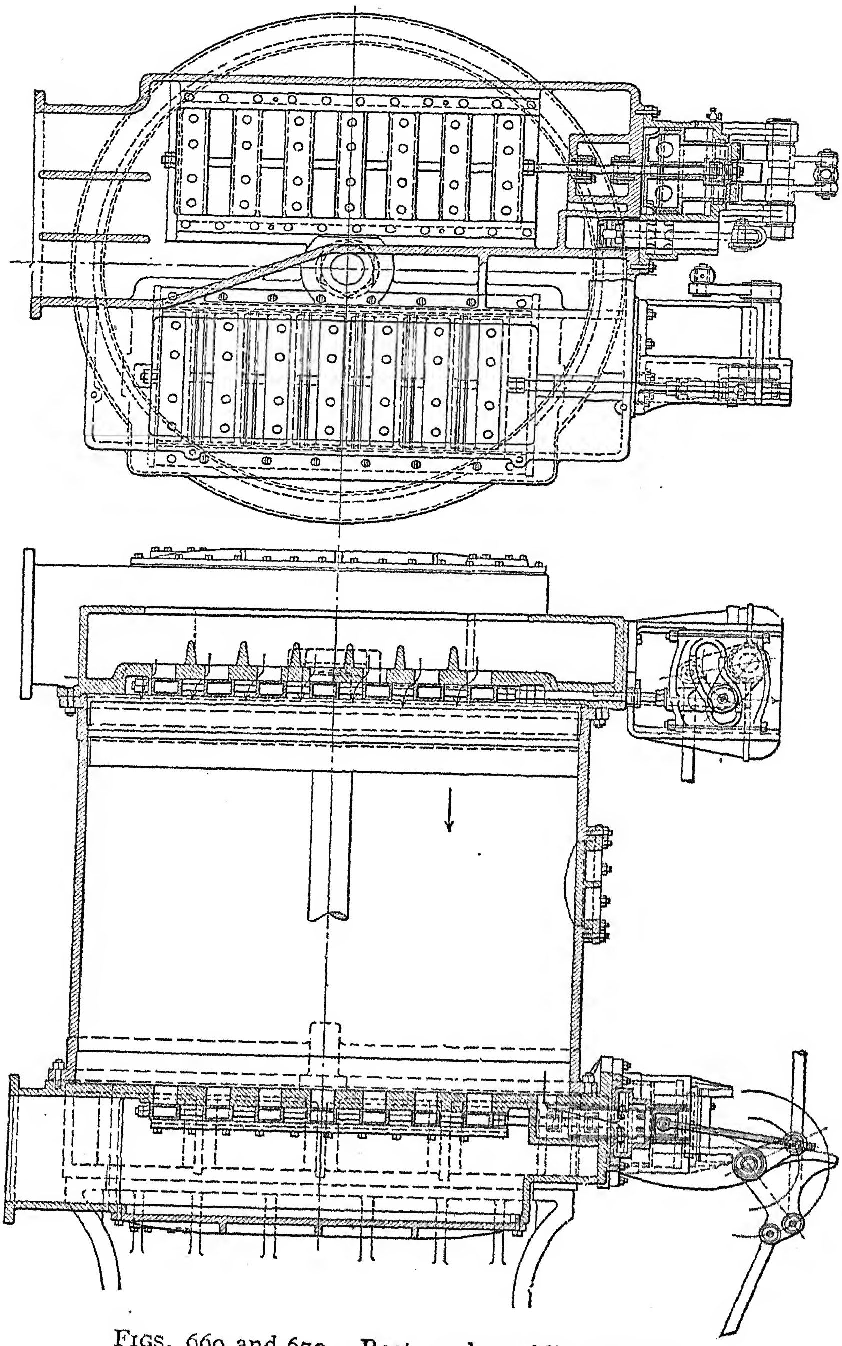
FIG. 668.—Gordon piston-valve.

in Figs. 669 and 670; the fan-shaped in Figs. 671 and 672. By a small movement of the valve an ample port-opening is obtained. The valves are lifted from their seats the moment they move, and rest on the back surfaces until the movement is completed; they are then forced back into their seats by the incoming or outgoing air as the case may be. The inlet-valves are positive in their movements, being actuated by a cam; the outlet-valves are automatic, being operated by a small auxiliary cylinder.

2. GENERAL ARRANGEMENT OF BLOWING ENGINES.—The air-cylinders of all modern blowing engines are stationary. As to the number of air-cylinders, the engines are single, more frequently duplex, sometimes triplex; and as to the position of the axis they are either horizontal or vertical. With steam as motive power, the engines are direct-connected or driven by a beam. In most cases the axes of the air- and steam-cylinders are parallel; a recent innovation,

[1] *Tr. A. I M. E.*, 1893, XXII, 542.

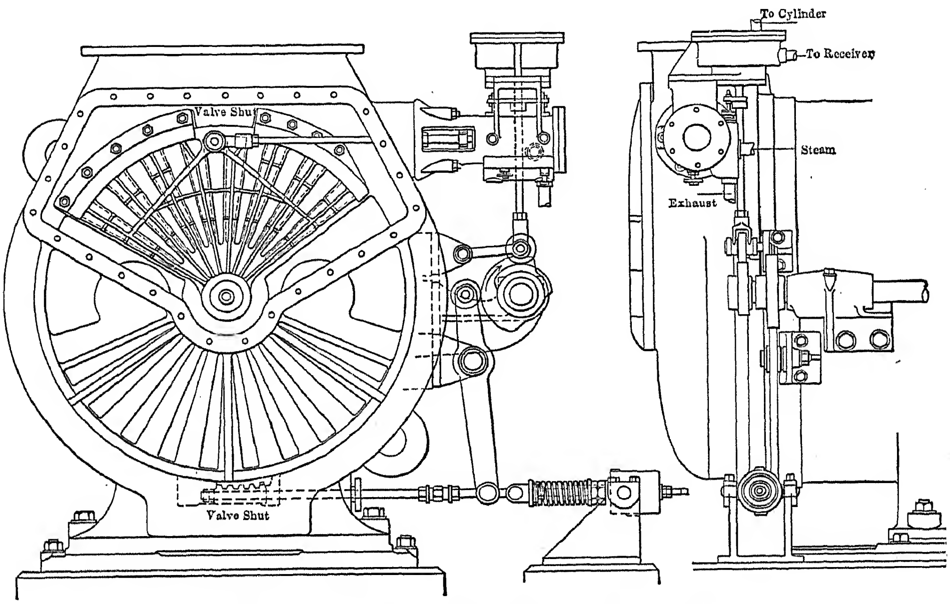
[2] *Tr. Am. Soc. Mech. Eng.*, 1899, XX, 967; *Iron Age*, July 13, 1899, p. 9.



FIGS. 669 and 670.—Rectangular gridiron-valve.

called vertical-horizontal, is to have a vertical air-cylinder crank-driven from a horizontal steam-cylinder. Single engines are driven by a single steam-engine or by a pair which are tandem-compound; with duplex engines, the steam is usually cross-compounded. Gas-driven engines so far are all horizontal.

3. HORIZONTAL BLOWING ENGINES are always direct-connected. Fig. 673 represents a single direct-connected horizontal engine with flywheel, in which *a* is the steam-cylinder; *b*, air-cylinder; *c*, blast-pipe; *d*, flywheel; *f*, broad guides.[1] All single engines have the disadvantage that when the steam-pressure is at a maximum, the blast-pressure is at a minimum and *vice versa* This is shown by the indicator cards given in Figs. 674 and 675. These diffi-



FIGS. 671 and 672.—Fan-shaped gridiron-valve.

culties are in part overcome by extra heavy flywheels; they are done away with almost wholly by duplex engines with cranks at 90 degrees. Fig. 676 shows such a direct-connected horizontal engine with flywheel having duplex steam-cylinders and connected air-cylinders.[2] The indicator-cards, Figs. 677 to 680, show that when the power of steam-cylinder *I* is weakening, that of steam-cylinder *II* is strongest and does the increasing work of air-cylinder *I*; in the same way steam-cylinder *I* supplements steam-cylinder *II* when the power of the latter is weakening.

Fig. 681 represents a horizontal blowing engine driven by a two-cycle Oechelhaeuser gas-engine, in which P_1 and P_2 are the pistons; *d* and d_1,

[1] Drawing: Wöhlert, *Verh. Verein. Beförd. Gewerb.*, 1867, XLVI, 52.

[2] Drawings: Kennedy, *Tr. A. I. M. E.*, 1893, XXII, 537; Krompach, *Stahl u. Eisen*, 1898, XVIII, 929; Mesta Machinery Co., *Iron Trade Rev.*, Aug. 11, 1904; Tod, *Iron Age*, 1906, LXXVIII, 338; Nordberg, *op. cit.*, 1907, LXXXIII, 568; Slick, *op. cit.*, 1909, LXXXIV, 1773.

feeders; S, air-cylinder; V, flywheel; D, dynamo for starting; C, pump for compressing gas-air mixture; T, cross-head.

The piston-rod of P_2 acts directly upon the cross-head; that of P_1 through a three-throw crank the shaft of which is also connected with the flywheel and the compression-pump.[1]

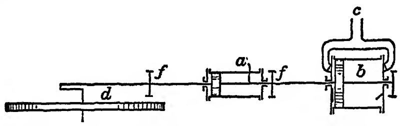


FIG. 673.—Single horizontal blowing engine.

Steam cylinder. 36×48 in. Blowing cylinder. 84×48 in.

FIGS. 674 and 675.—Indicator cards, single blowing engine.

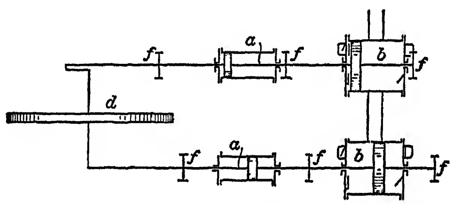
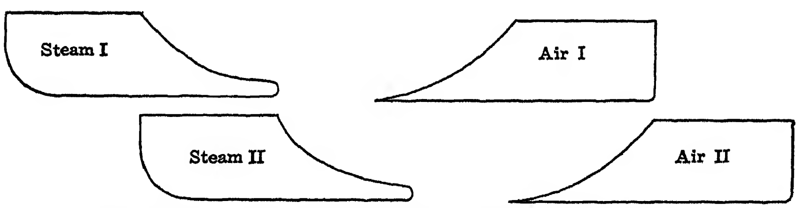


FIG. 676.—Double horizontal blowing engine.



FIGS. 677 to 680.—Indicator cards duplex blowing engine.

Fig. 682 gives a sketch of a Riedler-Stumpf duplex crank-driven blowing engine with a flywheel having a four-cycle gas-engine as motive power. A_1 and A_2 are the air-cylinders; G_1 and G_2, working-cylinders, V, flywheel.[2]

[1] Drawings: Oechelhaeuser-Junkers, *Iron Age*, Dec. 6, 1900, p. 8; Greiner, *J. I. and St. I.*, 1900, I, 109; Hubert, *Rev. Un.*, 1900, L, 156; Lürmann, *Stahl u. Eisen*, 1901, XXI, 489; Lackawanna Steel Co., *Iron Age*, Jan. 7, 1904, p. 49; Riedler, *Zt. Verein. deutsch. Ing.*, 1905, XLIX, 273; *Rev. Mét.*, 1905, II, 368; *Iron Age*, 1905, LXXV, 1980, 2050; LXXVI, 80, 160; Strack, *Stahl u. Eisen*, 1904, XXIV, 1296; *Iron Age*, 1905, LXXV, 242; Meyer, *Stahl u. Eisen*, 1900, XX, 385; 1905, XXV, 67, 133; Hubert, *Tr. A. I. M. E.*, 1906, XXXVII, 647; *J. I. and St. I.*, 1906, III, 16; Reinhardt, *Tr. A. I. M. E.*, 1906, XXXVII, 669; *J. I. and St. I.*, 1906, III, 36.

[2] Drawings: Lürmann, *Stahl u. Eisen*, 1899, XIX, 763; Riedler, *Zt. Verein deutsch. Ing.*, 1905, XLIX, 273; *Rev. Mét.*, 1905, II, 368; *Iron Age*, 1905, LXXV, 1980, 2050; LXXVI, 80, 160; Reinhardt, and *J. I. St. I.*, 1906, III, 36; *Tr. A. I. M. E.*, 1906, XXXVII, 669; Simmersbach, *Stahl u. Eisen* 1006, XXVI, 1311; Drawe, *op. cit.*, 1910, XXX, 246, 290; Hellmann, *op. cit.*, 1911, XXXI, 1292.

Horizontal blowing engines have the advantages of simplicity of construction, of accessibility and moderate foundations; the disadvantages are shortness of stroke and wear of pistons, piston-rods and cylinders on the lower sides. Up to 1873 they were the leading engines for iron blast-furnaces; since then they have been largely replaced by vertical engines.

With the advent of gas-driven blowing engines the horizontal type is again making headway. For steel-converters horizontal engines are used almost exclusively.

In this country the leading user of the horizontal steam-driven engine for blast-furnace work has been the Bethlehem Steel Co., South Bethlehem, Pa. Most of its engines are cross-compounded and the air-cylinders are connected; further, the weight of the piston is carried by steam-pressure applied

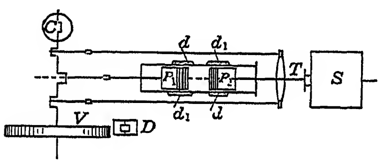


FIG. 681.—Single horizontal blowing-engine driven by two-cycle Oechelhauiser gas-engine.

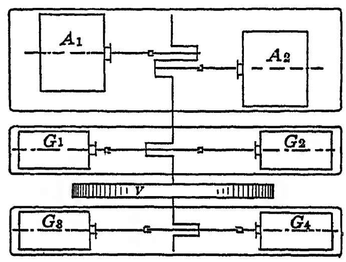


FIG. 682.—Double horizontal blowing-engine driven by four-cycle Riedler-Stump gas-engine.

in chambers on the lower side, the steam being admitted through the hollow piston-rod.[1] A recent innovation with duplex horizontal as well as with vertical engines is to have the steam-cylinders cross-compounded and the air-cylinders disconnected. This arrangement is advocated by the Southwark Foundry & Machine Co., Philadelphia, Pa. This company has developed a quarter-crank arrangement of steam- and air-cylinders "whereby the maximum pressure upon the steam-piston before cut-off is balanced by resistance upon the air-piston" which makes the engine run smoothly and without noise.

4. VERTICAL BLOWING ENGINES are direct-connected or beam-engines. Direct-connected engines are of two types.

The long-crosshead type,[2] shown in Fig. 683, has the air-cylinder, *b*, above the cross-head, *c*, which is between steam-cylinder, *a*, and the air-cylinder; two flywheels each with a wrist-pin in the hub or in one arm. This style is largely used in this country, as it is cheap, takes up little room and is easily

[1] Kennedy, *Tr. A. I. M. E.*, 1893, XXII, 537.

[2] Drawings: Ashland, *Iron Age*, 1889, XLIII, 691; Witherow, *op. cit.*, 1890, XLVI, 1078; Gordon-Strobel-Lareau, *op. cit.*, 1891, XLVII, 374; Dickson, *op. cit.*, 1891, XLVII, 319; Columbus, *op. cit.*, 1891, XLVIII, 441; Macbeth, *Iron Trade Rev.*, March 5, 1903, p. 96; Gordon, *Iron Age*, July 13, 1897, p. 7; Mesta Machine Co., *op. cit.*, Dec. 22, 1904, p. 7, and Dec. 21, 1905, p. 25.

accessible; its disadvantages are that the cross-head is likely to break, and that putting the wrist-pins in the wheels tends to set up vibrations in them.[1]

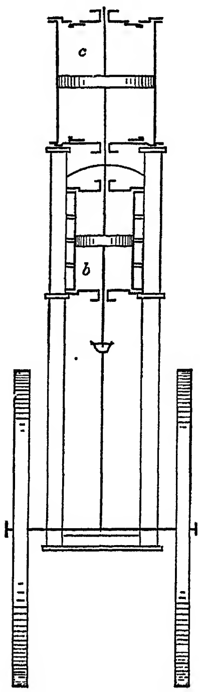


FIG. 683.—Vertical blowing engine, long-cross head type.

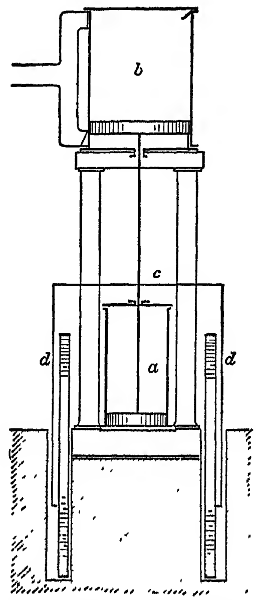
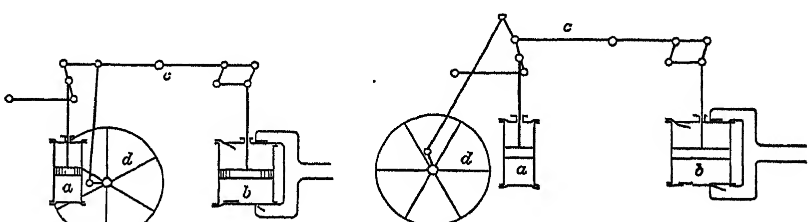


FIG. 684.—Vertical blowing engine, steeple type.

The second type, that of the steeple blowing engine,[2] is shown in Fig. 684. The air-cylinder, *c*, above, the steam-cylinder, *b*, below, and the cranks at the bottom; the engines are coupled to cranks on the ends of the shafts



FIGS. 685 and 686.—Vertical blowing engines, beam type.

placed at 90 degrees to one another; the shaft carries the two flywheels, *d*. The engines avoid the disadvantage of having wrist-pins in the flywheels and

[1] Kennedy, *Tr. A. I. M. E.*, 1893, XXII, 537.

[2] Allis, *Iron Age*, Sept. 16, 1897, p. 11; Youngstown, *op. cit.*, Dec. 21, 1899, p. 9; Buckeye, *op. cit.*, June 7, 1900, p. 10; Colo. Fuel & Iron Co., *op. cit.*, Aug. 14, 1902, p. 1; Westinghouse, *op. cit.*, Dec. 3, 1903, p. 4; Reynolds, *Eng. Min. J.*, 1903, LXXVI, 280.

dispense with the long cross-head; they give a uniform pressure of blast and are convenient for starting.[1] Vertical direct-connected blowing engines take up little room, are accessible, have a uniform wear of piston and piston-rod, and permit a long stroke; but they require a strong foundation.

Beam-engines[2] are practically obsolete in this country. While they share many of the advantages of the direct-connected engines they have the disadvantages of indirect transmission of power, of slow speed, of occupying much room, of being inaccessible and expensive. Two types of single engines are represented in Figs. 685–686, in which a = steam-cylinder, b = air-cylinder, c = beam, d = flywheel.

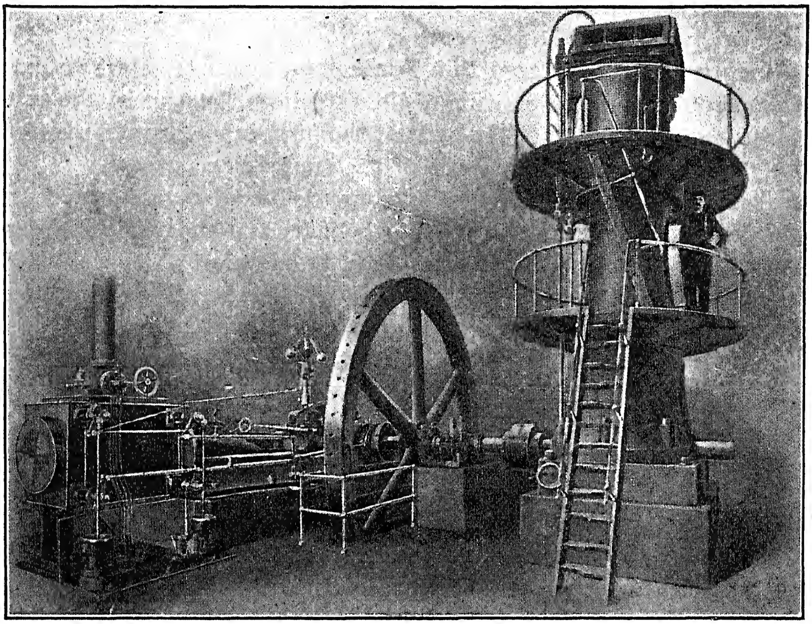


FIG. 687.—Southwark vertical-horizontal blowing engine, compound disconnected.

5. VERTICAL HORIZONTAL BLOWING ENGINES[3] reduce the great height necessary for a modern steeple blowing engine and increase the accessibility. A horizontal steam- (or gas-) engine has been combined with a vertical blowing engine of the steeple type as shown in Fig. 687. This arrangement combines the advantages of the horizontal and vertical types of blowing engines, and is growing in favor with iron blast-furnace men.

318. Capacity and Efficiency of Blowing Engines.—The volume or quantity of blast a blowing engine can deliver per min. is governed by the net piston-area

[1] Kennedy, *loc. cit.*

[2] Drawings: Maryland Steel Co., *Iron Age*, 1891, XLVII, 968.
Pribram, *Oest. Zt. Berg. Hüttenw.*, 1897, XLV, 517, 553.

[3] Drawings: Weimer, *Iron Age*, March 21, 1901, p. 8.
Koerting, *op. cit.*, Jan. 7, 1904, p. 55.

(the total area minus the piston-rod area) A, the length of stroke, l, the r.p.m. n, the pressure of the atmosphere, p, and the pressure of the blast, p'. The theoretical quantity of air drawn in is $Q=A\times l\times 2n$, and the theoretical quantity of blast delivered $Q_1=Q\frac{p}{p_1}$. On account of the clearance, the slight vacuum existing in the cylinder upon reversal of the stroke and the ever-present leakage, the actual air-capacity is smaller than the theoretical, which is generally accepted as about 75 per cent. With the slow velocity and the great length of stroke of a modern blast-furnace blowing engine the efficiency may reach 90 per cent.; with the quicker moving Bessemer blowing engine and its shorter length of stroke, the efficiency may fall to 60 per cent. The efficiency of the engine, that is, the ratio of the i.h.p. of the air-cylinder to that of the steam-cylinder is usually taken at 90 per cent.

319. **Amount of Air Delivered to a Furnace.**[1]—This can be arrived at: (1) from the displacement of the blowing engine; (2) from the area of the tuyères, the temperature and pressure of the blast at the furnace; (3) from measurement with Pitot tube; (4) from the analysis of the tunnel-head gases.

It has been shown that the volume of air delivered to a furnace can be ascertained from the dimensions of the air-cylinder and the number of strokes of its piston; also that the air-efficiency may be assumed to be 75 per cent., but this figure varies as much as 10 per cent.[2] The method therefore gives only approximate results, but these are of the greatest value in the practical running of a furnace. It was J. Kennedy who in 1871 started the working-principle of running an iron blast-furnace by the revolutions of the blowing engine, provided of course that each furnace has its own engine. This principle has become the universal practice in this country, and all blowing engines have their revolution-counters.

The product of velocity of flow and tuyère-area gives closer results than the foregoing method, provided the counter-pressure of the gases in the furnace, the barometric pressure of the air, and the temperature of the blast (elevated under ordinary compression of the air) are duly considered. Von Hauer[3] gives the following formula for calculating in cbm. the quantity of blast M_o delivered per min. at 0° C., and the standard atmospheric pressure of 760 mm.:

$$M_o=21{,}110\ \lambda d^2\sqrt{\frac{b+h_2}{1+\alpha t_1}}\ (h_1-h_2)\ \text{cbm.},$$

in which d is diam. of tuyère in mm.; b, pressure of atmosphere; h_1, that of blast and h_2 that of gases in furnace, all in mm.; α, coefficient of air for 1° C., 0.00366; t_1 temperature of air in blast-pipe, deg. C., and λ a factor $=1-0.03\times\frac{h_1-h_2}{b+h_2}$. In order to determine the quantity of blast

[1] For iron blast-furnace: Brisker, C., "Berechnung und Untersuchung des Eisenhochofens," Knapp, Halle, 1909, p. 58.

Hurck, *Stahl u. Eisen*, 1910, XXX, 500.

[2] Uehling, *J. Franklin Instit.*, 1905, CLIX, 125.

[3] Die Hüttenwesens-Machinen, 1876, p. 19.

delivered under any working conditions, the above formula has to be multiplied by $\frac{(1+\alpha t_s)\beta}{b}$, which gives the following expression

$$Ms = 21{,}1110 \times \frac{(1+\alpha t_s)\beta}{b} \times \lambda\ d^2 \sqrt{\frac{b+h_2}{1+\alpha t_1}}(h_1-h_2) \text{ cbm.}$$

in which t_s is the temperature of atmosphere and $\beta = 760$ mm. The same author[1] furnishes tables for calculating the values of $b+h_2$ and λ for different temperatures. He further simplifies the above formula to

$$M = 18{,}740\ d^2\sqrt{h_1 - h_2}$$

for the assumption that $t_s = t_1 = 10°$ C., $b = \beta$, $\lambda = 1$, and that the increase of h_2 over b is ignored. For this formula he supplies[2] tables covering tuyère-diameters 20 to 160 mm., and values of $h_1 - h_2$ ranging from 12 to 300 mm. A third set of tables[3] gives the values for bessemer blowing engines.

Measurement by means of the Pitot tube[4] was devised by Pitot in 1732 for ascertaining the velocity of water. In its simplest form it consists of a tube bent at right angles and open at the ends; the longer leg is held vertically and the shorter is plunged in the water with the mouth pointing squarely to the current. The height of the rising water is proportional to the velocity of the stream and furnishes thus a means of measurement. In 1856 Darcy added a straight tube to the bent one in order to measure the pressure-head in addition to the velocity-head. The first application of the Pitot tube to the flow of gases was probably made by Robinson in 1873. The tube has undergone many changes in the details of construction and arrangement.

The form shown in Figs. 688 to 692 and described by Jager and Westly, is essentially the same as the standard of the Boston and Montana and the Anaconda smelting companies. The static-pressure tube, A, and the velocity-head tube, B, are joined as shown in C. The lower end of tube A, carries a horizontal nipple, closed at the front by a plug and provided on either side with a slot 1 1/4 by 1/32 in. By this arrangement the velocity of the gas has no aspiratory effect. Tube B is bent at right angles and measures both the static- and velocity-pressures when it is turned toward the gas-current. By connecting the ends of the two tubes to those of an U-shaped manometer, the static pressure is canceled, while the manometer read direct shows only the velocity-pressure.

The manometer, Figs. 691 and 692, consists of an inverted glass bottle filled

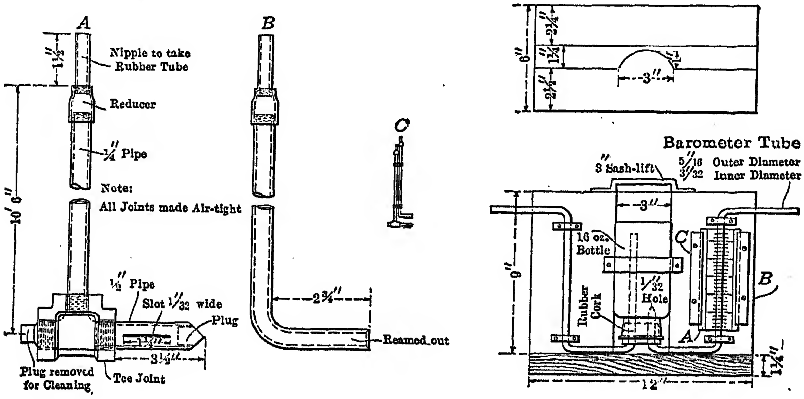
[1] *Op. cit.*, p. 600.

[2] *Op. cit.*, p. 602.

[3] *Op. cit.*, p. 613.

[4] Robinson, *Van Nostrand Mag.*, 1886, XXXV, 91; *Geol. Survey of Ohio*, 1888, VI, 548; *Tr. Am. Soc. Mech. Eng.*, 1904, XXV, 208; Káš, *Oest. Zt. Berg. Hüttenw.*, 1888, XXXVI, 591.
Rateau, *Ann. Min.*, 1898, XIII, 331.
Taylor, *Eng. News*, 1904, LII, 387.
Burnham, *op. cit.*, 1905, LIV, 660.
Vambera-Schraml, *Oest. Jahrb.*, 1904, LIV, 1.
Jager-Westly, *Eng. Min. J.*, 1909, LXXXVIII, 468.
Smith, *op. cit.*, 1910, XC, 1197 (clamp for tube.
McQuigg, *op. cit.*, 1913, XCV, 649,768 (Richards).
Cloud, *op. cit.*, 1913, XCV, 961.

in part with grain alcohol and closed with a stopper through which pass the velocity-tube extending above, and the static tube ending beneath the level of the liquid. In order to do away with oscillations, which prevent accurate readings on the scale A, the static tube is drawn to a point. The scale, cross-section paper divided into 1/10 in. and subdivided again into halves, permits estimating 1/100 in.; it is held down by the metal straps B and C, and can be moved up and



FIGS. 688 to 692.—Pitot tube and manometer.

down to adjust the zero point. As the bottle is 3 in. diam., and the tube 0.0938 in., a rise of 1 in. in the tube will correspond to a fall in the bottle of only 0.00098 in., a negligible quantity.

In making a test, it is necessary to make a number of observations in the cross-section of a flue or pipe in order to obtain a true average; a long straight part of the flue should be chosen to take observations, as otherwise eddy-currents will be met with. In averaging the observations, any negative readings due to eddy-currents have to be thrown out.

TABLE 244.—CORRECTION FOR POSITION OF TIP WITH PITOT TUBE

Fraction of distance of tip from center to wall of flue	Correction to be allowed per cent.	Fraction of distance of tip from center to wall of flue	Correction to be allowed per cent.
Center 0.0	Deduct 9.2	0.55	Deduct 5.2
0.1	Deduct 9.1	0.60	Deduct 4.3
0.2	Deduct 9.0	0.65	Deduct 2.9
0.25	Deduct 8.7	0.70	Deduct 1.5
0.30	Deduct 8.3	0.75	Add 0.8
0.35	Deduct 7.8	0.80	Add 3.1
0.40	Deduct 7.3	0.85	Add 6.4
0.45	Deduct 6.6	0.90	Add 9.8
0.50	Deduct 6.0	Wall 1.00	....

As the flow of gas in a flue is quicker at the center than at the side, a correction must be applied for the position of the tip in the flue. Table 244 by Robinson[1] gives the corrections to be made for a circular flue. Thus, when the tip is held in the center, a deduction of 9.2 per cent. must be made from the calculated result.

Besides the readings with the tube, it is necessary to take the analysis and temperature of the gas, the pressure of the atmosphere and the specific gravity of the alcohol.

Let, *e.g.*, the manometer-readings average 1/4 in. =1/48 ft. alcohol; the gas-analysis be CO_2 9.8, N 79, O 10.5, SO_2 0.7 per cent. by vol.; the gas temperature 285 deg. C., the barometer-reading 27.95 in. =710 mm.; and the specific gravity of the alcohol 0.81. First, the volume will have to be changed into weight as shown in the following table. At 0 deg. C. and 760 mm. pressure, each

Cas	Per unit. Volume	Multiplied by density referred to H=1	Gives sp. gr.
CO_2	0 098	22	2.156
N	0.790	14	11.060
O	0.105	16	1.680
SO_2	0.007	32	0.224
	1.000		15.120

gram molecule of gas occupies a space of 22.3 l. As the molecule of H contains 2 atoms, we have in 22.3 l., $2\times15.12=30.24$ g. gas. At 285° C. or $273+285=558°$ C. absolute, and at 710 mm. pressure, the original 22.3 l. change to $\frac{22.3\times558\times760}{273\times710}=49$ l.; hence 49 l. hot gas weigh 30.24, or 1 l. =0.617 g. As 1 l. alcohol weighs 810 g. and is 1313 times as heavy as the gas, the height of column of 1/48 ft. alcohol$=\frac{1313}{48}=27.34$ ft. of column of gas. Now the velocity formula $V=\sqrt{2gh}$ gives $\sqrt{2\times32.16\times27.34}=41.93$ ft. per sec.

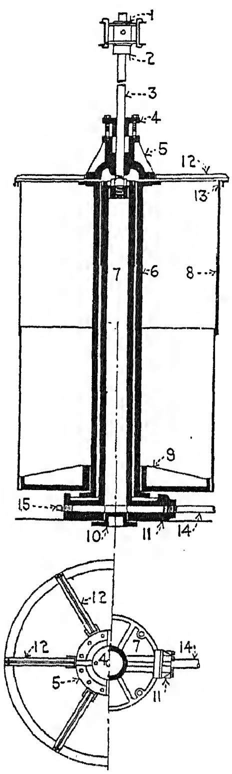
In calculating the amount of blast entering a furnace from the analysis of the tunnel-head gas, the simplest way is to base it upon the weight of charge, say of 100 kg. In a lead blast-furnace, *e.g.*, the analysis of the tunnel-head gas gave CO_2 19.4, CO 9.5, O none, N 71.1 per cent. by vol. As 1 cbm. CO_2 or CO contains at 0° C. and 760 mm. pressure 0.54 kg. C, the weight of the C in a cbm. of tunnel-head gas is equal the amounts of CO_2 and CO multiplied by 0.54. Hence C as $CO_2=0.194\times0.54$ and C as $CO=0.095\times0.54$; from which, total C in 1 cbm. $=0.289\times0.54=0.1561$ kg. The volume of gas produced per 100 kg. charge is now obtained by dividing the total weight of C gasified by the weight of the C in 1 cbm. gas. The coke required to smelt 100 kg. contained 15.63 kg. C., the CO_2 of the cerussite from the lead ore furnished 0.48 kg. C, and the CO_2

[1] *Loc. cit.*

of the limestone used as flux 0.72 kg. C; this gives a total of 16.83 kg. C. Therefore 16.83 : 0.1561 = 107.8 cbm. gas at 0° C. and 760 mm. pressure, which according to analysis contains 107.8×0.711 = 76.65 cbm. N, and this amount of N corresponds to 76.65 : 0.792 = 96.78 cbm. = 125.1 kg. of air.

320. Blast Accessories.—The leading apparatus to be considered briefly are the regulators, equalizers, or accumulators, the blast-mains with their gates, and the tuyères.

EQUALIZERS.—The use of an equalizer is to reduce to a minimum the pulsations of the blowing engine in order that the furnace may receive a constant current of air. Formerly a large spherical steel vessel was used for this purpose in connection with blast-furnaces; it has been replaced in most instances by a cylindrical vessel, the cubical content of which ought to be, with a single engine, at least 20 times the capacity of the air-cylinder. With duplex engines the capacity can be one-half that required for a single engine, and with triplex engines it can be dispensed with entirely. With hot-blast, the brick stove takes the place of the equalizer.



FIGS. 693 and 694. Accumulator.

In converting pig iron or matte, the converter must be supplied at intervals with the required volume of air at a given pressure. The blowing engine running more or less continuously delivers the compressed air to an accumulator. Figs. 693 and 694 represent an apparatus 14×120 in. made by the Philadelphia Engineering Works: stationary ram (7), 14 in. in diam., is securely bolted to the foundation; moving cylinder (6), 17 1/2 in. in diam., slides on guide-rod (3), 3 1/2 in. in diam., and is guided by means of stuffing-box (5); rod (3) is held at the bottom by ram (7) and at the top by bearing (1) which is secured to the roof truss; the weight-box (8), 1/16-in. plate-iron and 71 in. in diam., rests upon flange (9) of cylinder (6). The apparatus can be weighted up for a pressure reaching 500 lb. per sq. in.

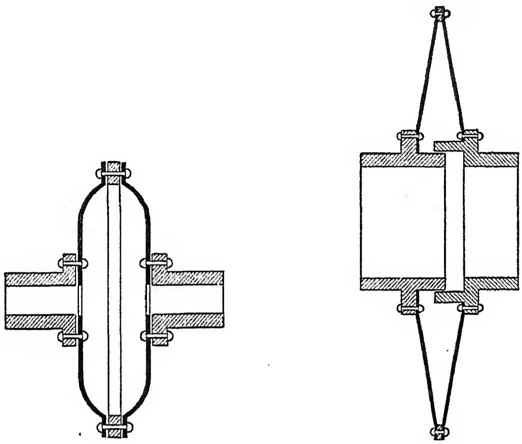
Hydraulic accumulators are obsolete with blowing engines.

BLAST PIPES.—The blast-pipe conducting the blast to the furnace is usually of steel plate; with preheated blast it is lined with fire-brick. Its area, A, is proportional to the quantity of the blast, Q, and indirectly to the velocity, v, or $A = \frac{Q}{v}$; hence the diameter $d = \sqrt{\frac{Q \times 4}{v \times \pi}} = 1.131\sqrt{\frac{Q}{v}}$. With high-pressure blast, v should not be over 70 ft. per sec.; with low-pressure not over 35 ft. Von Hauer[1] gives the formula

$$D = \sqrt{0.0212 \times \frac{(1+\alpha t_1)\beta}{b+h_1} \times \frac{M_0}{U}} \text{ meters.}$$

[1] *Op. cit.*, p. 53.

The terms of this formula have been given on p. 788; $U = 10$ to 15 meters. In order to allow for expansion and contraction (coefficient for iron $= 0.00111$ for every $100°$ C.,) some form of stuffing-box or expansion-joint as shown in Figs. 695 and 696 is placed at intervals in the blast-main. The main as well as the



FIGS. 695 and 696.—Expansion joint.

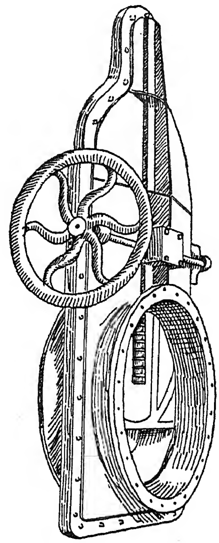


FIG. 697.—Blast-gate.

branches are usually supplied with sliding-gates to cut off the blast entirely, to throttle or regulate[1] it, or to let some blast escape into the open. They are usually of cast-iron; large sizes are always operated by rack and pinion, Fig. 697; small sizes by hand, Fig. 698; frequently they are of the lever-pattern Fig. 699.

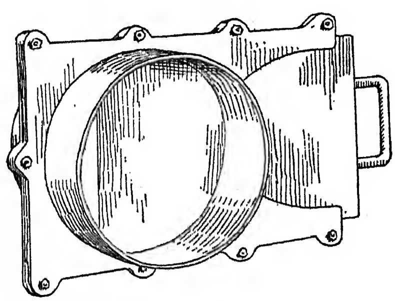


FIG. 698.—Blast-gate.

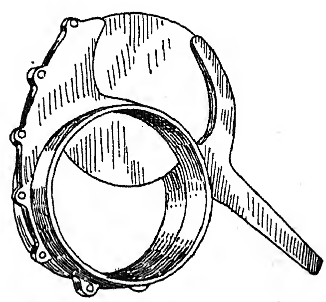


FIG. 699.—Blast-gate.

TUYÈRES.—Since the forms of tuyères vary with different furnaces, details are better studied in connection with the furnaces. The nozzle of a tuyère is always conical, as a high velocity is thereby obtained; according to Weisbach[2]

[1] Cabot-Vaughen, Blast-regulator, *Iron Age*, Dec. 4, 1902, p. 27.

[2] Weisbach, J., and Coxe E. B., "Theoretical Mechanics," Van Nostrand, New York, 1872, p. 949.

the coefficient of efflux for a conical pipe reaches the maximum of 0.92 with an angle of convergence of about 6°. With blast-furnaces the nozzles are cast-iron, bronze or brass, and they may be water-jacketed, water-sprayed or simply air-cooled. The number and size of tuyères depend upon the quantity of blast to be delivered into the furnace. The number is governed in part also by the intervening space necessary for the stability of the furnace. Ledebur[1] gives for an iron blast-furnace, $n=\pi\times$ diam. of tuyère-section in meters. The diameter is calculated by Von Hauer[2] by the formula $d=\sqrt{\frac{M_0}{21110}}\sqrt{\frac{1+\alpha t_1}{(b+h_2)(h_1-h_2)}}$ meters. The meaning of the letters in the formula is given on p. 788.

PREHEATING AND DRYING OF AIR

321. Preheating Fuel and Air in General.—In some pyrometallurgical processes the fuel is preheated before it comes in contact with air as, *e.g.*, the coke during its descent in a blast-furnace; in others the air alone is preheated as is the case with the open-hearth steel furnace fired with natural gas; again both fuel and air are preheated as in the hot-blast iron blast-furnace or the open-hearth steel furnace fired with producer-gas. The result of this preheating is to add the sensible heat of the fuel or the air or of both to the heat generated by combustion, and the effect is a raising of temperature and a saving of fuel. Only in exceptional cases will it pay to burn extraneous fuel for the sake of preheating; as a rule the heat of the furnace otherwise lost by conduction and radiation, or that contained in the waste gases or slags is utilized for this purpose, or the tunnel-head gases themselves have a fuel-value which can be further utilized. In preheating fuel, only gaseous fuel need be considered, as the artificial drying and warming of natural solid fuels, such as wood, peat and lignite have become obsolete in metallurgical practice, and the preheating of carbonized fuel in the blast-furnace is inherent in the process. The warming of liquid fuel to be burnt in a furnace has already been discussed in §§ 138 and 153. Of the gaseous fuels the leading one is producer-gas; gases rich in C_xH_y are decomposed upon heating, and water gas is used only in small heating furnaces with which preheating is out of question.

Heat is transmitted[3] either by conduction or radiation; it is conducted when the transmission is in the mass of a body; it is radiated when the transmission is through an intervening medium without affecting the temperature of the latter.

In heat conduction[4] a distinction has to be made between that taking place in a single body, and between two bodies in close contact. For a single body the

[1] "Handbuch der Eisenhüttenkunde," 4 ed., 1903, p. 426.

[2] *Op. cit.*, p. 25.

[3] Richards, J. W., Metallurgical Calculations, McGraw-Hill Book Co., New York, 1906, I, p. 172.

[4] Hering, *Met. Chem. Eng.*, 1908, VI, 495; 1909, VII, 11; 1910, VIII, 627; 1911, IX, 8, 175, 568, 625.

thermal conductivity, or the quantity of heat which passes through a plate the two sides of which are kept at constant difference of temperature, is proportional to the area, A, of the plate, the time of flow, t, the difference in temperature between the two faces of the plate, $S_1 - S$, and a constant, K, representing the rate of flow, which differs with the substance; it is inversely proportional to the thickness, d, of the plate; or $Q = \frac{A \times t \times (S_1 - S)}{d} \times K$; $K = \frac{Q \times d}{A \times t \times (S_1 - S)}$. This coefficient of thermal (or calorimetric) conductivity, K, in C.G.S.[1] units is the number of gram cal. which passes in 1 sec. through the opposing faces of a cube of substance 1 cm. in thickness which are kept at a constant difference of 1° C. Thus K for soft steel is 0.11; this means that a cube of soft steel with sides of 1 cm. which have a difference of temperature at the two surfaces of 1° C., will allow 0.11 gram cal. to flow through it per sec. In the same way a sheet of soft steel 1/10 the thickness and 100 times the area will allow $0.11 \times 10 \times 100 = 110$ gram cal. to flow through per sec. The thermal resistance (restivity) is the reciprocal of thermal conductance (conductivity).

TABLE 245.—VALUES OF K IN C.G.S., CALORIES[2]

Silver, at 0°	1.10	Ice	0.00500
Copper, 0–30°	0.92	Glass, 10–15°	0.00150
Copper, commercial	0.82	Water	0.00120
Copper, phosphorized	0.72	Quartz sand, 18–98°	0 00060
Aluminum, 0°	0.34	Fire-brick dust, 20–98°	0.00028
Aluminum, 100°	0 36	Retort-graphite dust, 20–100°	0 00040
Zinc 15°	0.30	Magnesia-brick dust, 20–100°	0.00050
Brass, yellow 0°	0.20	Magnesia, calcined, Grecian, granular, 20–100°	0.00045
Brass, yellow 100°	0.25	Magnesia, calcined, Styrian, granular, 20–100°	0.00034
Brass, red, 0°	0 25	Magnesia calcined, light, porous, 20–100°	0.00016
Brass, red, 100°	0 28	Infusorial earth[3] (Kieselguhr) 17–98°	0.00013
Iron, wrought, 0°	0 21	Infusorial earth, 0–650°	0.00038
Iron, wrought, 100°	0 16	Brick-dust, ordinary coarse, 0–100°	0 00039
Iron, wrought, 200°	0.14	Charcoal, powdered, 0–100°	0.00022
Iron, steel, soft	0.11	Coke, powdered, 0–100°	0.00044
Iron, steel, hard	0.06	Carbon, gas-retort, solid, 0–100°	0.01477
Lead, 0°	0.084	Cement, 0–700°	0.00017
Lead, 100°	0 076	Alumina-brick, 0–700°	0.00204
Antimony, 0°	0.044	Magnesia-brick, 0–1300°	0.00620
Mercury, 0°	0 015	Fire-brick, 0–1300°	0.00310
Mercury, 50°	0.019	Fire-brick, 0–500°	0.00140
Mercury, 100°	0.024		
Bismuth, 0°	0.018		

Table 246 by Queneau[4] for refractory brick is compiled from the experimental results of Wolgodine[5] for temperatures up to 1,000° C. and in a few instances 1,200° C. The latest table is that compiled by Hering.[6]

[1] Hering, Thermal ohm and mho, *Met. Chem. Eng.*, 1912, IX, 3.
[2] Richards, *op. cit.* pp. 175, 184.
[3] Steger, *Stahl u. Eisen*, 1907, XXVII, 1697.
[4] *Electrochem. Met. Ind.*, 1909, VII, 383.
[5] *Rev. Mét.*, 1909, VI, 765, transl. by Queneau, *Electrochem. Met. Ind.*, 1909, VII, 383, 433.
[6] *Met. Chem. Eng.*, 1911, IX, 652.

TABLE 246.—THERMAL CONDUCTIVITY OF REFRACTORY MATERIALS

Material	Conductivity		
	Gr. cal. sec., per sq. cm., per cm. per 1° C. diff.	Kg. cal. hr., per sq. cm., per cm., per 1° C. diff.	Relative per cent.
Graphite brick	0 025	9 0	100
Carborundum brick	0 0231	8.32	92.4
Magnesia brick	0 0071	2.54	28 4
Chromite brick	0 0057	2 05	22 8
Fire-brick	0 0042	1 50	16.7
Checker brick	0 0039	1 42	15 8
Gas-retort brick	0 0038	1.36	15 2
Building brick	0.0035	1.26	14 0
Bauxite brick	0 0033	1 19	13.2
Glass pot	0.0027	0 96	12 4
Terra-cotta	0.0023	0 84	9.3
Silica brick	0 0020	0 71	7 8
Kieselguhr brick	0 0018	0 64	7 1

For two bodies, like solid and liquid or solid and gas, the heat-tansfer from one body to the other varies with the character of the solid and the liquid; the solid shows greater variations than the liquid; they are further influenced by the renewal of exposed surfaces due to the flow of the liquid or gas, *i.e.*, the velocity of its circulation. Experiments have shown that the coefficient of thermal conductivity for air or similar gases and metal, $K=0.000028(2+\sqrt{v})$ gram cal., with v, measured in cm., and for hot water and metal, $K=0.000028(300+180\sqrt{v})$.

A hot body will radiate heat in all directions. The amount of heat lost by radiation depends upon the nature of the hot body and is proportional to the difference between the fourth powers of the absolute temperatures of the hot body and its surroundings.

It has been determined by Peclet[1] in gram. cal. per sq. cm. of surface for surroundings at 0° C. and hot bodies at 100° C. Some of his figures are as given in Table 247.[2]

TABLE 247.—LOSS OF HEAT BY RADIATION AT LOW TEMPERATURES

Polished brass	0.00108	Russia sheet iron	0.01410
Copper	0.00068	New cast-iron	0.01332
Polished sheet iron	0.00189	Oxidized iron	0.01410
Leaded sheet iron	0.00273	Glass	0.01222
Ordinary sheet iron	0.01164	Building stone	0.01500

For higher temperatures, taking Peclet's figures as units, the following factors will give the desired data:[3]

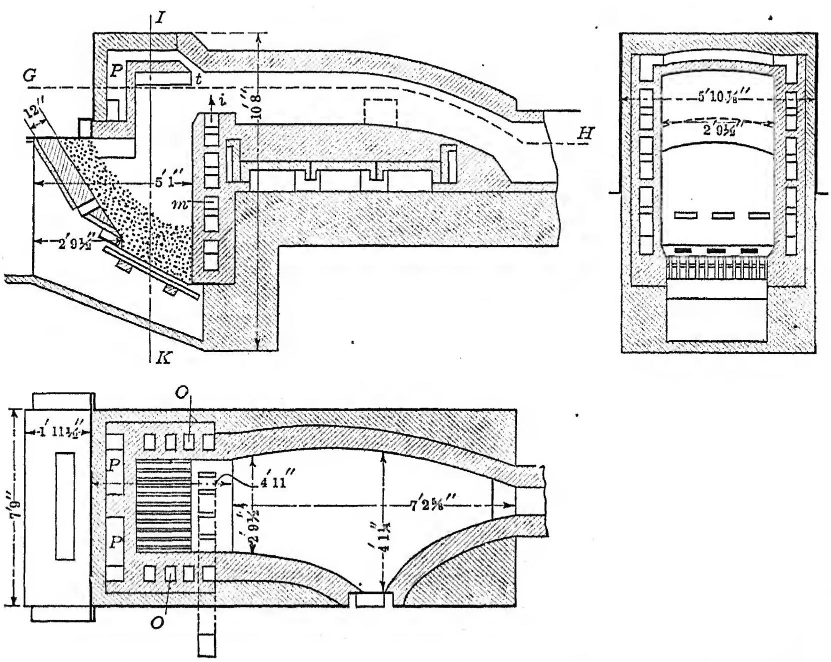
[1] Péclet, E., "Traité de la Chaleur," Masson, Paris, 1878, Vol. 1, p. 519.
[2] Richards, *op. cit.*, pp. 185 and 186.
[3] *Loc. cit.*

TABLE 248.—LOSS OF HEAT BY RADIATION AT HIGH TEMPERATURES

100 to 0°	1.0	600 to 0°	26.0
150 to 0°	2.0	700 to 0°	35.0
200 to 0°	3.3	800 to 0°	45.3
300 to 0°	7.0	900 to 0°	57.0
400 to 0°	12.0	1,000 to 0°	70.0
500 to 0°	18.3		

In calculating the loss of heat from a furnace wall, first the amount lost by transference through contact of wall and air will have to be ascertained, and secondly the amount lost by radiation proper.

The two systems of heat-transmission in furnaces are carried out in different ways. They may be conveniently classed under three heads: (1) The contact radiation system, action continuous; (2) the counter-current (recuperator) system, action continuous, and (3) the Siemens (regenerator) system, action intermittent. In each a distinction is to be made between low and high pressures representing the application to the reverberatory and the blast-furnace.



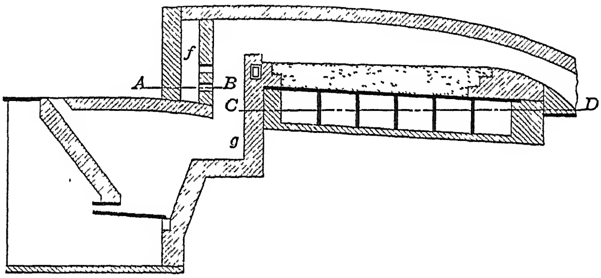
FIGS. 700 to 702.—Boëtius gas-producer and heating furnace.

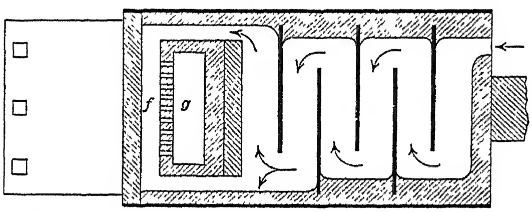
322. The Contact-radiation System.—Part of the heat generated in a furnace is transmitted through the walls by conduction and lost by radiation at the surface. The amount thus lost depends upon the thickness and conducting power of the furnace material, the difference in temperature inside and outside, the surface and its radiating capacity, and the time.

In reverberatory smelting-furnaces it is not uncommon to make the fire-

bridge hollow or to build flues beneath the hearth in order that air drawn through may keep these parts cool and thus protect them against slagging or melting. In many cases the air thus preheated is conducted underneath the grate or conducted through the roof above the fire-bridge, and the heat withdrawn from the furnace thus restored to it.

The Boëtius gas-producer,[1] Figs. 700 to 702, patented in 1865, represents a type of furnace in which air-passages surround the lining of the fire-box. Air circulates through the horizontal flues, *m*, and enters the furnace through the fire-bridge, *i*, while the air that travels to and fro through the flues, *o*, enters the flues, *p*, and is admitted above the fire-bridge through ports, *t*, in the roof. This form of semi gas-firing is used quite extensively on the European continent in puddling-, zinc-[2] and glass-furnaces.[3]





FIGS. 703 and 704.—Bicheroux heating furnace.

In the Bicheroux furnace,[4] Figs. 703 and 704, air circulates underneath the hearth, enters a vertical chamber, *f*, on top of the producer and enters the rising flue, *g*, through a number of ports.

These furnaces are usually run with natural draft; however, as the air for a modern gas-producer is frequently supplied by an injector, air under moderate pressure may be used here, but only in exceptional cases.

The following may serve as an example of a calculation for the cooling of the flue in the fire-bridge of a reverberatory furnace: the cooling flue is 20 cm. wide, 30 cm. deep, and 2 m. long; the walls are 25 cm. thick; the temperature on the

[1] *Berg. Hüttenm. Z.*, 1869, XXVIII, 452; *Rev. Un.*, 1877, I, 202.

[2] Ingalls, W. R., "Metallurgy of Zinc and Cadmium," Engineering and Mining Journal, 1903, pp. 23, 271, 288, 304, 370, 417, 446, 448.

[3] Ramdohr, "Die Gasfeuerung," Halle, 1877, Part II, Plate II.

[4] *Rev. Un.*, 1874, XXXVI, 139, 1877, I, 196; *Berg. Hüttenm. Z.*, 1874, XXXIII, 434; 1877, XXXVI, 233; 1878, XXXVII, Plate V, Figs. 18 and 19.

Ledebur, A., "Die Gasfeuerungen für Metallurgische Zwecke," Felix, Leipsic, 1891, p. 112.

fire-box side is 1,400° C., over the top 1,200°, and on the hearth-side 1,100° C. How much air at 20° C. must be blown through the flue to reduce the temperature of the walls to 200° C. ?

The conductance of 1 cm. thickness of fire-brick is 0.0014 g. cal., that of 25 cm. = 0.000056 g. cal. The area of the fire-box side = $200 \times 30 = 6{,}000$ sq. cm.; the difference in temperature = $1{,}400 - 200 = 1{,}200°$ C.; hence the heat to be removed = $1{,}200 \times 6{,}000 \times 0.000056 = 403.2$ g. cal. per sec. The area on the hearth-side = $200 \times 30 = 6{,}000$ sq. cm., the difference in temperature = $1{,}100 - 200 = 900°$ C., hence the heat to be removed = $900 \times 6{,}000 \times 0.000056 = 302.4$ g. cal. per sec. The area on the top = $200 \times 20 = 4{,}000$ sq. cm., the difference in temperature = $1{,}200 - 200 = 1{,}000°$ C., hence the heat to be removed = $1{,}000 \times 6{,}000 \times 0.000056 = 336$ g. cal. per sec. The total heat to be removed = 1,041.6 g. cal. per sec. The conductance of fire-brick to air = $0.000028\ (2+\sqrt{v})$ C.G.S. The total surface of the inside flue = 16,000 sq. cm. Assuming 20° C. to be the mean temperature of the air, the total difference in temperature inside the flue and outside is $200 - 20 = 180°$ C., hence the heat to be removed $16{,}000 \times 180 \times 0.000028\ (2+\sqrt{v}) = 1{,}041.6$ g. cal. per sec.; $2+\sqrt{v} = \frac{1041.6}{16 \times 18 \times 0.28} = 12.9$; $\sqrt{v} = 10.9$; $v = 116.6$ cm. per sec. The cross-section of the flue = $20 \times 30 = 600$ sq. cm.; hence $600 \times 118.8 = 71{,}286$ cbm. = 0.713 cbm. per sec., approximately.

In blast-furnaces, the smelting-zone is sometimes cooled by an air-jacket, that is, it is enclosed by steel or copper boxes through which air is forced by means of a pressure-blower and then delivered to the bustle-pipe to pass as pre-heated air through the tuyères into the furnace. Enclosing the smelting-zone of an Arizona copper blast-furnace[1] with copper plates stopped the melting of the sides, as the conducting- and radiating-power of copper is greater than that of fire-brick.

The following calculation gives the amount of air which is required by an air-jacket in a blast-furnace.

The tuyère-section is 1×1.3 m.; the height of the copper jacket 1 m.; the air-space 15 cm.; the brick walls inside the jacket are 15 cm. thick. The temperature inside the furnace is 1,200° C., the desired temperature of the outside wall 200°; the mean temperature of the cooling-air 25° C.

The conduction of 1 cm. thickness of fire-brick is 0.0014 g. cal., that of 15 cm. = 0.0001 g. cal. The mean area of the inside and outside of the brick wall is $520 \times 100 = 52{,}000$ sq. cm. The difference in temperature between the inside and outside brick wall = $1{,}200 - 200 = 1{,}000°$ C.; hence the heat to be removed by air = $1{,}000 \times 0.0001 \times 52{,}000 = 5{,}200$ g. cal. per sec. The conduction of fire-brick to air = $0.000028(2+\sqrt{v})$ C.G.S. The area of the fire-brick surface = $2 \times (130+160) \times 100 = 58{,}000$ sq. cm., and the difference in temperature = $200 - 25 = 175°$ C.; hence the heat to be removed is $58{,}000 \times 0.000028(2+\sqrt{v}) \times 175 = 5{,}200$ g. cal. per sec.; this gives $v = 265.7$ cm. per sec. The cross-section of the four cooling-jackets = $4 \times 160 \times 15$ cm. = 9,600 sq. cm. = 0.96 sq. m., and the volume of air = $0.96 \times 2.657 = 2.5$ cbm. per sec.

[1] Wendt, *Tr. A. I. M. E.*, 1886–87, xv, 43.

323. The Counter-current (Recuperator) System.—In the preceding system, heat from the fire-box, the hearth or smelting-zone has been transferred to another part of the furnace or carried away. In the counter-current system, the heat of the products of combustion of a reverberatory or blast-furnace, or in an iron blast-furnace the heat generated by the burning of the hot tunnel head gases, is utilized in part by having the gases travel around a set of flues through which the air passes in the opposite direction, Fig. 705. With the reverberatory furnace, only the air is usually thus pre-heated, as the producer gas generally goes straight from the producer with its primary heat to the furnace placed close to it; however, sometimes the gas also is pre-heated. With the blast-furnace, of course, only the air is pre-heated.

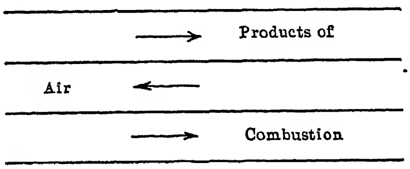


Fig. 705.—Principle of recuperation.

The counter-current principle of heating is of great importance and can be made as complete as is desired by increasing the length of flue and thereby the duration of contact. The apparatus for furnishing hot air to reverberatory and hot blast to blast-furnaces differ considerably from one another as regards form and building material.

In a reverberatory furnace the recuperator consists of a series of circular or rectangular flues, usually vertical and not over 6 in. wide, which are built of fire-clay tiles, rarely of brick, with rabbet or broken joints in order to insure an air- and gas-tight fit. Iron pipes used in the earlier constructions have been entirely abandoned on account of their scaling when heated to above 400 ° C.[1] Any leakage in such a heating-flue would allow air to escape into the gas-flue or *vice versa*, the direction of flow depending upon which of the two had the greater pressure.

Beside the danger of leakage, the system has the disadvantage that the heat has to be conducted through a material which is an inferior conductor of heat, and which has to be made at the same time thick enough, 1 to 1 1/4 in., to be mechanically strong. For the satisfactory absorption of heat the flues can be numerous and short or few and long; the former plan is preferable as it makes exchanges possible, and permits arrangements for easy accessibility for the removal of dust and accretions. One of the earliest furnaces is the one erected by Ponsard[2] to serve a puddling furnace; the inaccessibility and cracking of the tiles interfered with its working satisfactorily. Other forms are those of Schwarzkopf,[3] Schmidthammer,[4] Pietzka,[5] Blezinger,[6] Schmatolla,[7] Lencau-

[1] Schmidhammer, *Stahl u. Eisen*, 1893, XIII, 798.

[2] *Berg. Hüttenm. Z.*, 1873, XXXII, 197; *Bull. Soc. Ind. Min.*, 1872, I, 747; *Iron Age*, 1876, XII, 342; *Rev. Un.*, 1876, XXXIX, 131.

[3] *Op. cit.*, 1896, XVI, 688.

[4] *Op. cit.*, 1893, XIII, 796.

[5] *Stahl u. Eisen*, 1889, IX, 562.

[6] *Op. cit.*, 1893, XIII, 466.

[7] *London Min. J.*, 1909, LXXXV, 167.

chez,[1] Nehse,[2] Converse-De Saulles;[2] further the coking ovens of Coppée (Figs. 154–158), Simon-Carvès (Fig. 173), Semet-Solvay (Figs. 174–176), and others. The fuel-economizers of steam-boilers are counter-current recuperators in which water circulates through iron pipes instead of air.

The calculation of the flue area of a recuperator may be shown in connection with the Converse-De Saulles zinc distilling furnace, Fig. 706, in operation at the works of the New Jersey Zinc Co., Palmerton, Pa. In the figure the products of combustion make one zig-zag pass in the recuperators on either side of the furnace through 15 horizontal flues. The number and form of flues have been changed in the latest furnaces, and the gases make three passes instead of two. There are 21 flues (each 12×23 cm.), in a pass; the walls are 5 cm. thick; the waste gas enters the recuperator at 1,100° C. and leaves it at 550° C., at which temperature its volume is 100 cbm. per min. The volume of air required, measured at standard conditions, is 21 cbm. per min.; it is blown in under a pressure of 29 g. per sq. cm. The draft of the stack shows a depression of 1.5 mm. Hg. The problem is, how long must the air- and gas-flues be to reduce the temperature of the waste gas from 1,100 to 550° C., and raise that of the air from 20 to 700° C.

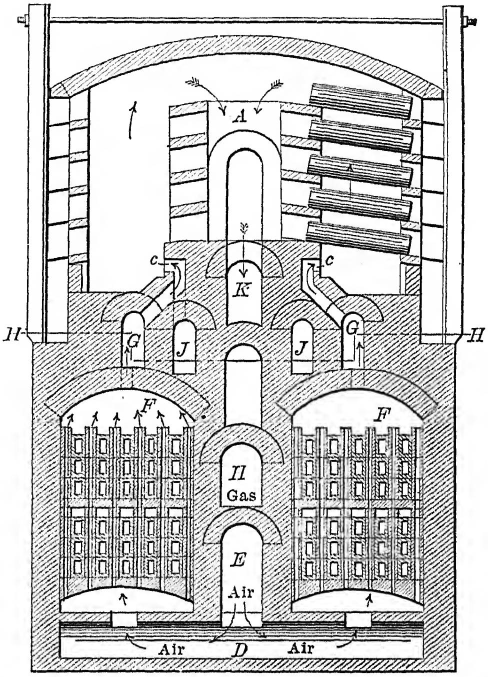


Fig. 706.—Converse—De Saulles recuperator zinc smelting furnace.

A, Vertical flue for products of combustion; *D* and *E*, cold-air flues; *F*, recuperative chambers; *G*, hot-air canals; *H* and *J*, gas-flues; *K*, horizontal canal for products of combustion; *c*, air-ports.

(A) Velocity of Gases.—The cross-section of a gas-flue is 12×23 = 276 sq. cm.; the 21 flues = 21×276 = 5,796 sq. cm. The volume of the waste gas entering the stack per min. is 100 cbm.; its velocity on leaving the flues $= \frac{100,000,000}{5796} = 17,250$ cm. per min. $= 287.5$ cm. per sec. The velocity of the waste gas entering the flues is to the velocity leaving the flues as are their absolute temperatures. $x : 287.5 = (1,100+273) : (550+273)$, $\therefore x = 480$ cm. per sec. entering the flue. The mean velocity is $\frac{287.5+480}{2} = 384$ cm. per sec.

[1] Le Verrier, "Procédés de Chauffage," Gauthier-Villars, Paris, 1902, p. 238.

[2] Ingalls, W. R., "Metallurgy of Zinc and Cadmium," Engineering and Mining Journal, New York, 1903.

(B) VELOCITY OF AIR.—The cross-section of an air-flue is $\frac{\pi D^2}{4}=0.7854 \times 7.6^2=45.36$ sq. cm.; that of the 360 air-flues$=360\times 45.36=16,330$ sq. cm. The volume of air entering the flues per min. is 21 cbm. at standard conditions; at 20° C. and 29 g. pressure this is 21.95 cbm. The velocity of the air entering the flue is $\frac{21,950,000}{16,330}=1,344$ cm. per min.$=22.4$ cm. per sec. The velocity of the air leaving the flues is to the velocity entering as are their absolute temperatures, $x:22.4=(700+273):(20+273)$; $\therefore x=74.4$ cm. per sec. The mean velocity is $\frac{22.4+74.4}{2}=48.4$ cm. per sec.

(C) The coefficient of transfer from gas to fire-brick is $0.000028(2+\sqrt{v})$ $=0.000028(2+\sqrt{384})=0.0006048$.

(D) The coefficient of transfer from fire-brick to air is $0.000028(2+\sqrt{v})$ $=0.000028(2+\sqrt{48.4})=0.0002509$.

(E) The coefficient of conduction of fire-brick is 0.0021 per cm. thickness, or for 9 cm.[1] it is 0.00023.

(F) Heat absorbed by the air in being heated from 20° to 700° C. The mean specific heat of air from 20 to 70° is 0.322 per cbm.; for the 21 cbm. it is $21\times 0.322\times(700-20)=4,598$ kg. cal. per min.$=76.65$ kg. cal. per sec. $=$ 76,650 g. cal. per sec.

(G) CONDUCTANCE OF THE SYSTEM.—This is the sum of

$$\frac{1}{\frac{1}{C}+\frac{1}{D}+\frac{1}{E}}=\frac{1}{\frac{1}{0.0006048}+\frac{1}{0.0002509}+\frac{1}{0.000023}}=\frac{1}{1653+3986+4348}=\frac{1}{9987}=0.0001$$

for 1° C. difference in temperature for 1 sq. cm. surface.

The mean difference in temperature of gas and air is $\frac{\text{gas entering}+\text{gas leaving}}{2}-\frac{\text{air entering}+\text{air leaving}}{2}=\frac{1100+550}{2}-\frac{20+700}{2}=465°$ C.

Let x = desired surface, then $x\times 465\times 0.0001=76,650$ g. cal. to be transferred; $x=1,649,000$ sq. cm. the mean between the total surface of the air- and the gas-flues. Assuming that, for standard conditions of gas and air, the effective heating-surfaces of the gas- and air-flues are in the ratio of 3 : 2; the effective surface of the gas-flues will be 1,979,000 sq. cm., that of the air-flues 1,319,000 sq. cm. The mean between the total surfaces of the air- and gas-flue was found to be 1,649,000 sq. cm. The total surface is $2\times 1,649,000=3,298,000$ sq. cm.; this is to be apportioned in the ratio of 3 : 2 and gives 1,979,000 and 1,319,000 sq. cm.

(H) LENGTH OF GAS-FLUES.—The calculation of the above surface areas is based upon the assumption of a uniform flow of heat through all parts of the

[1] The minimum thickness of fire-brick between air- and gas-flues is 7.7 cm.; in order to obtain an approximate value, 1/3 the radius of the air-flues has been added, which gives the value 9 cm. It is understood that the fact that heat coming from the ends of the gas-flues travels a longer distance has been neglected.

surface, but it is evident that the rate of flow through the tops and bottoms of the flues is much less than that through the sides.

In order to be on the safe side, it will be assumed that all the heat passes through the sides only. For each cm. in length a gas-flue has $2\times 23=46$ sq. cm. effective heating surface, or 21 flues for three passes have 2,898 sq. cm. The desired length will be $\frac{1979000}{2898}=683$ cm. $=6.83$ m.

(I) LENGTH OF AIR-FLUES.—As it is impossible to calculate the rate of transfer through each part of the circumference of an air-tube, it will be necessary to assume also here an effective surface of transfer and thus be in line to use the figure of total surface (1,319,000) calculated under (*G*). In order to be on the safe side, it will be assumed that this surface is twice the diameter of a flue, *i.e.*, for each cm. in length, a tube has $2\times 7.6=15.2$ sq. cm. effective surface, or 360 flues have $360\times 15.2=5{,}472$ sq. cm. The length of each flue is therefore $\frac{1319000}{5472}=240$ cm. $=2.4$ m.

HEATING BLAST BY COUNTER-CURRENTS (PIPE-STOVES).—Heating the air for an iron blast-furnace according to the counter-current principle is carried on in pipe-stoves,[1] that is, a series of externally-heated U-shaped cast-iron pipes through which air is forced by the blowing-engine on its way to the furnace and heated to a maximum of 500° C. In furnaces with open-top apparatus such as the Giroux hot-blast top,[2] the Kiddie hot-blast system[3] and others have been employed.

J. B. Neilson[4] of Glasgow, Scotland, in 1828 patented a blast-heating apparatus; in 1832 his first apparatus was introduced at the Garthsherry iron works; he later improved it, but always used extraneous fuel. In 1832 Faber du Faur[5] utilized the gases from the iron blast-furnace for heating blast and thus laid the foundation of modern practice.

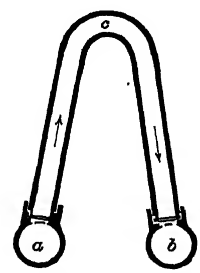


FIG. 707.—U-shaped blast-heating pipe.

In the earlier forms of stoves, the entire blast passed into the so-called tubular oven through a single zig-zag pipe. This was soon abandoned, on account of the leaky joints caused by the irregular expansion and contraction, and replaced by a series of smaller U-shaped pipes, *c*, Fig. 707, with ends fitting into sockets of two mains *a* and *b*. This arrangement forms the basis of all early pipe-stoves. The pipes, originally circular, are preferably made oval, as this form offers for a given area a larger heating surface; the larger the surface for a given quantity of air, the slower can be the velocity of the current, and slow speed means large heating-effect. However, a large surface goes with a large sectional area, and large pipes are difficult to cast. In order to reduce the num-

[1] Wedding, H., "Handbuch der Eisenhüttenkunde," Vieweg, Brunswick, 1906, Vol. III, pp. 54–95.

[2] *Min. Sc. Press*, 1906, LXXXXIII, 793; *Electrochem. Met. Ind.*, 1906, IV, 420.

[3] *Eng. Min. J.*, 1906, LXXXII, 598 (Jacobs).

[4] Sexton, *West. Scotland Iron and Steel Inst.*, through Ledebur, *Stahl u. Eisen*, 1895, XV, 509.

[5] Ledebur, *Stahl u. Eisen*, 1904, XXIV, 562.

ber of bends, pipes have been provided with a partition-wall, as shown in Figs. 708 and 709, which forces the air to make two passages in a pipe before it enters the next adjoining. The thickness of wall of a pipe has a range of 5/8 to 1 in.; the gray iron best suited should not contain[1] elements in excess of the following: P 0.5 per cent., C 3–3.5 per cent., Si 0.7–1.0 per cent., Mn 0.5 per cent. After

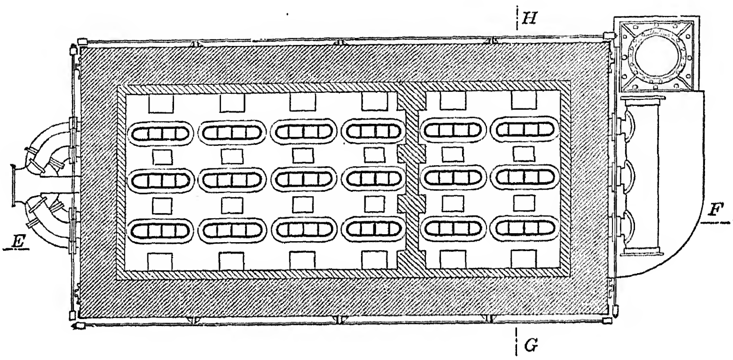


FIG. 708.—Cleveland pipe-stove.

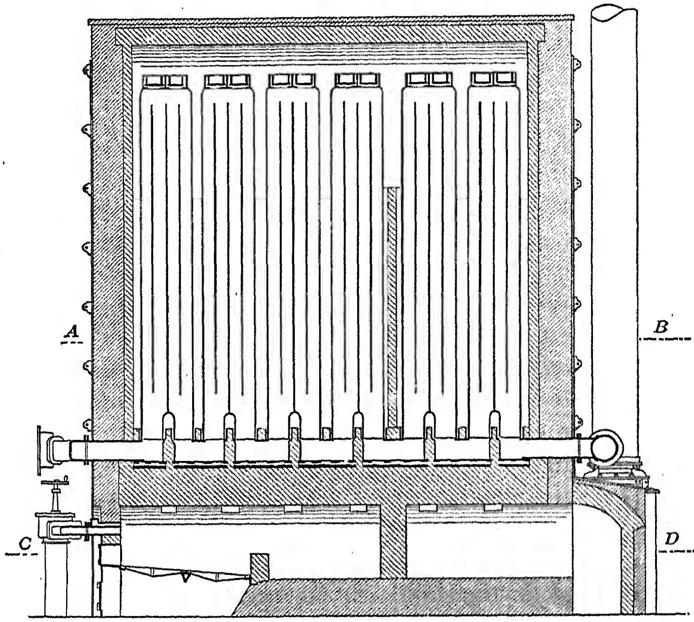


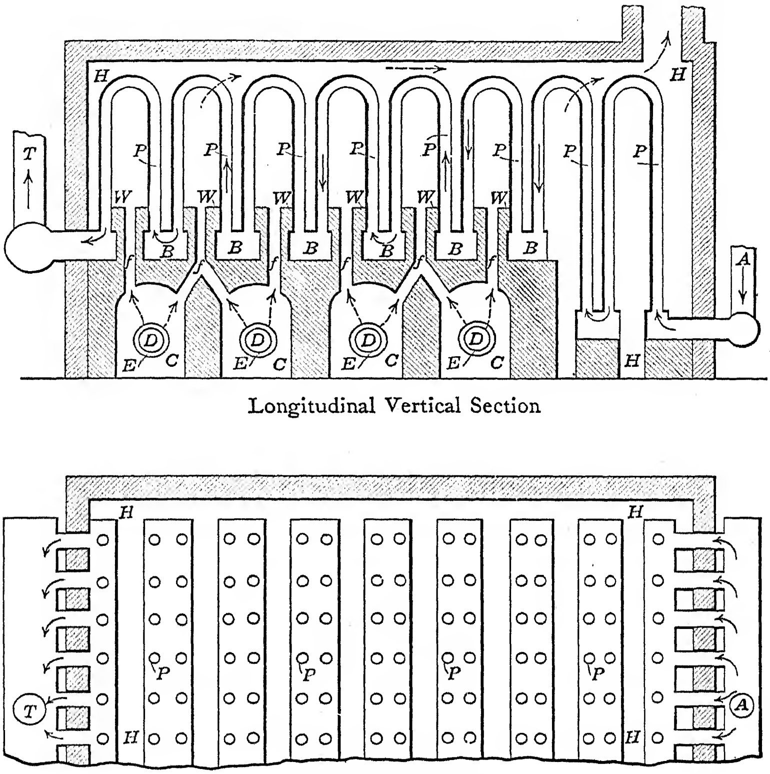
FIG. 709.—Cleveland pipe-stove.

being in use for some time, pipes become oxidized on the inner sides and do not transfer the heat as well as at first; they are also likely to become leaky. With large amounts of air to be heated, it is not advisable to have correspondingly large stoves, as the pipes are likely to be heated unevenly; the remedy lies in

[1] Ledebur, A., "Handbuch der Eisenhüttenkunde," Felix, Leipsic, 1903, p. 490.

increasing the number of stoves. In order to raise the temperature of 1 cu. ft. air per min. from say 15 to 450° C., not less than 0.6 sq. ft. of heating surface is required, supposing the velocity of the air-current not to exceed 50 ft. per sec.

Pipe-stoves have lost their former importance; they are still in use with low-temperature charcoal-furnaces, and some anthracite furnaces; they have maintained their original prominence with iron blast-furnaces smelting ore rich in



Horizontal Section

A - Cold-Air Main
B - Bed-Pipe
C - Combustion Chambers
f - Flues
P - Ox-Bow Iron Pipe
W - Brick Wall
T - Hot-Air Main
H - Chamber Heating
I - Chimney

FIGS. 710 and 711.—Ford pipe-stove.

zinc, as the fumes quickly clog the passages of brick stoves. According to the position of the pipes, stoves are classed as having standing, lying and suspended pipes.

The Ford Pipe-stove, Figs. 710 and 711, is an old form of stove formerly used quite extensively with the Lehigh Valley anthracite furnaces. Cold air from the blast-main, *A*, enters a branching battery of inverted U-shaped pipes at the right end of a heated brick chamber, *H*, and after passing through the pipes as indicated by the full-drawn arrows, the hot-blast is gathered into the hot-

blast main, T, at the opposite end. The tunnel-head gas enters the combustion chambers, C, through burners, D, and the air through the annular spaces, E; the gases burn, the products of combustion ascend in flues f, pass into the heating-chamber H, travel from left to right, and leave by the chimney I.

The Cleveland Pipe-stove, Figs. 708 and 709, is represented by the stove at

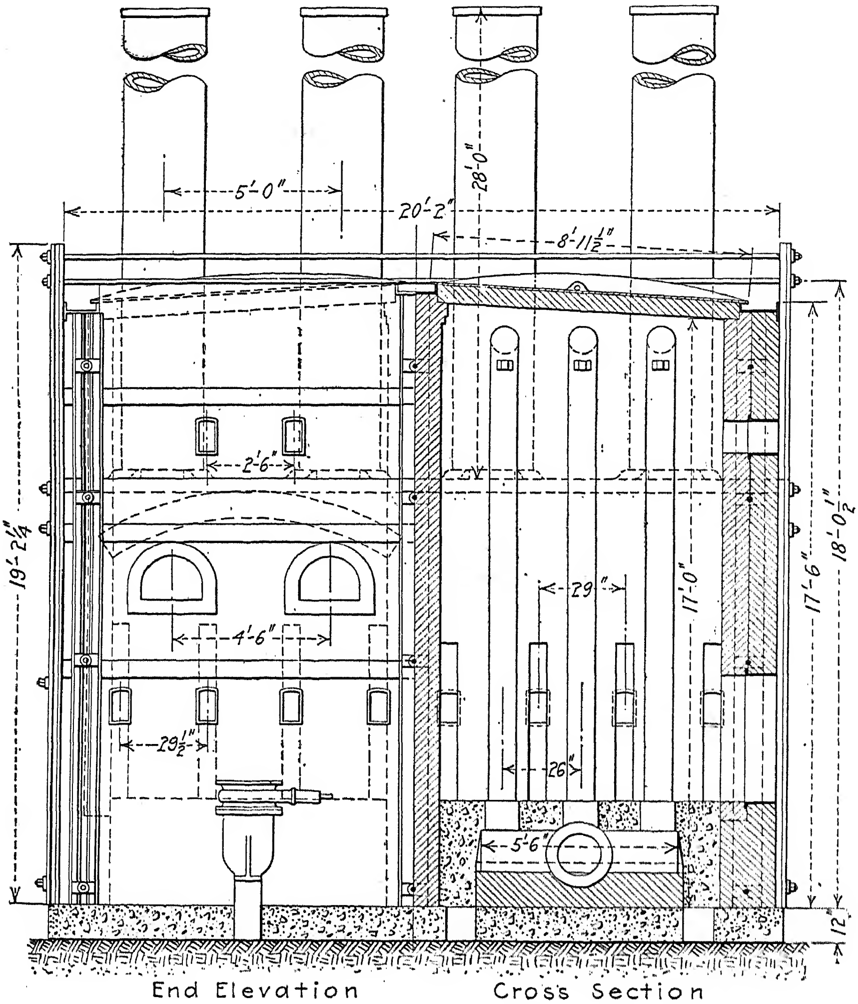


FIG. 712.—Durham pipe-stove.

Gleiwitz, Silesia.[1] There are three rows of vertical pipes; each pipe has one partition-wall and two strengthening-ribs; the open top is closed by a collar made air-tight with a lute. A pipe is 18 ft. 6 in. long, 2 ft. 2 in. wide, and 9 1/2 in. deep inside; the inner cross-sectional area is 1.57 sq. ft., the outer heating-

[1] Wiebmer. *Zt. Berg. Hütten. Sal. Wesen i. Pr.*, 1882, xxx, 178.

surface 97 sq. ft.; the stove has 18 pipes with a content of 1,260 cu. ft., and offers a heating-surface of 2,655 sq. ft. Cold blast arrives through the vertical pipe at the right, is split into three branches, and zig-zags up and down from right to left through the pipes traveling a distance of 206 ft.; at the left the branches combine and form the hot-blast main. The heating-gas enters at the left above the grate, carrying glowing coal, and is ignited. The flame enters the larger heating chamber through 16 ports, rises, passes over the division-wall, descends in the smaller chamber, leaves this through 8 ports, and enters the flue that leads to the stack. Three stoves can heat 5,600 cu. ft. air per min. to 600° C., but at this temperature the pipes begin to soften, twist and crack; usually the temperature is held at 400 to 500° C., but occasionally it is raised to 520°. The Haas stove[1] is similar in its leading features.

The Durham Stove,[2] a third form, is the one designed by E. Cooper for the Durham blast-furnaces at Riegelsville, Pa.; it has become the standard for New Jersey and Eastern Pennsylvania. One of its latest forms, as constructed by the Hartman Co. of Philadelphia, is shown in Figs. 712 to 714. This represents a 30-pipe double stove. Each stove has 3 rows of pipes and 10 pipes in a row. Sometimes a stove has 4 rows of pipes, but never more than 10 in a row. The characteristic of the Durham stove is that the burning-gas and cold-blast enter at the same end of the stove and travel through it in the same direction, so that there is no counter-current system. The advantage claimed for the procedure is that the pipes are protected from destruction where the flame is hottest, but this is done at the expense of an imperfect utilization of the heat developed. The stove is an oblong block 35 ft. 3 3/4 in. by 20 ft. 2 in., and 18 ft. 8 1/4 in. high; the strongly ironed vertical walls are red-brick on the outside, fire-brick on the inside; the straight sloping roof is built of guttered iron plates, 9 in. wide, that are lined with refractory tiling 6 in. thick; the chimneys are of 3/16-in. iron. In one end-wall are the ports, 21×15 1/2 in., for four 14-in. jet-burners; the gas is ignited in a combustion-chamber, 8 ft. 2 in. × 2 ft. 6 in., from which the flame passes into two heating-chambers, each 24 ft. 10 in. × 7 ft. 10 in. and 14 ft. high, through four slots, 5 ft. × 6 in., placed opposite the open spaces between the pipes, and between the pipes and walls. Beneath the gas-ports are small cleaning-doors, 6×11 in., which furnish access from the outside, while the stove is in operation, and thus provide a means for insuring a uniform distribution of flame. This travels through the chamber in four streams, leaves it through four slots, 4 ft. 6 in.×6 in., terminating in a main flue, 8 ft. 2 in.×1 ft. 6 in., that carries four iron chimneys, 28 3/4 in. in outside diam. There are two cold-blast mains, one for each section. A main, 15 in. in diam., delivers the air to a distributing bed-pipe which feeds three rows of heating-pipes; the last pipe of each row ends in a receiving bed-pipe which is connected with the hot-blast main. The heating-pipes, 14 ft. long with 9-in. outside and 6 7/8-in. inside diameters, are placed 13 5/8 in. between centers. The bed-pipes, which are short and free to move, are embedded in the clay filling. The 60 pipes of the

[1] *Eng. Min. J.*, 1904, LXXVIII, 1028.

[2] Fackenthal, *Tr. A. I. M. E.*, 1885–86, XIV, 130.

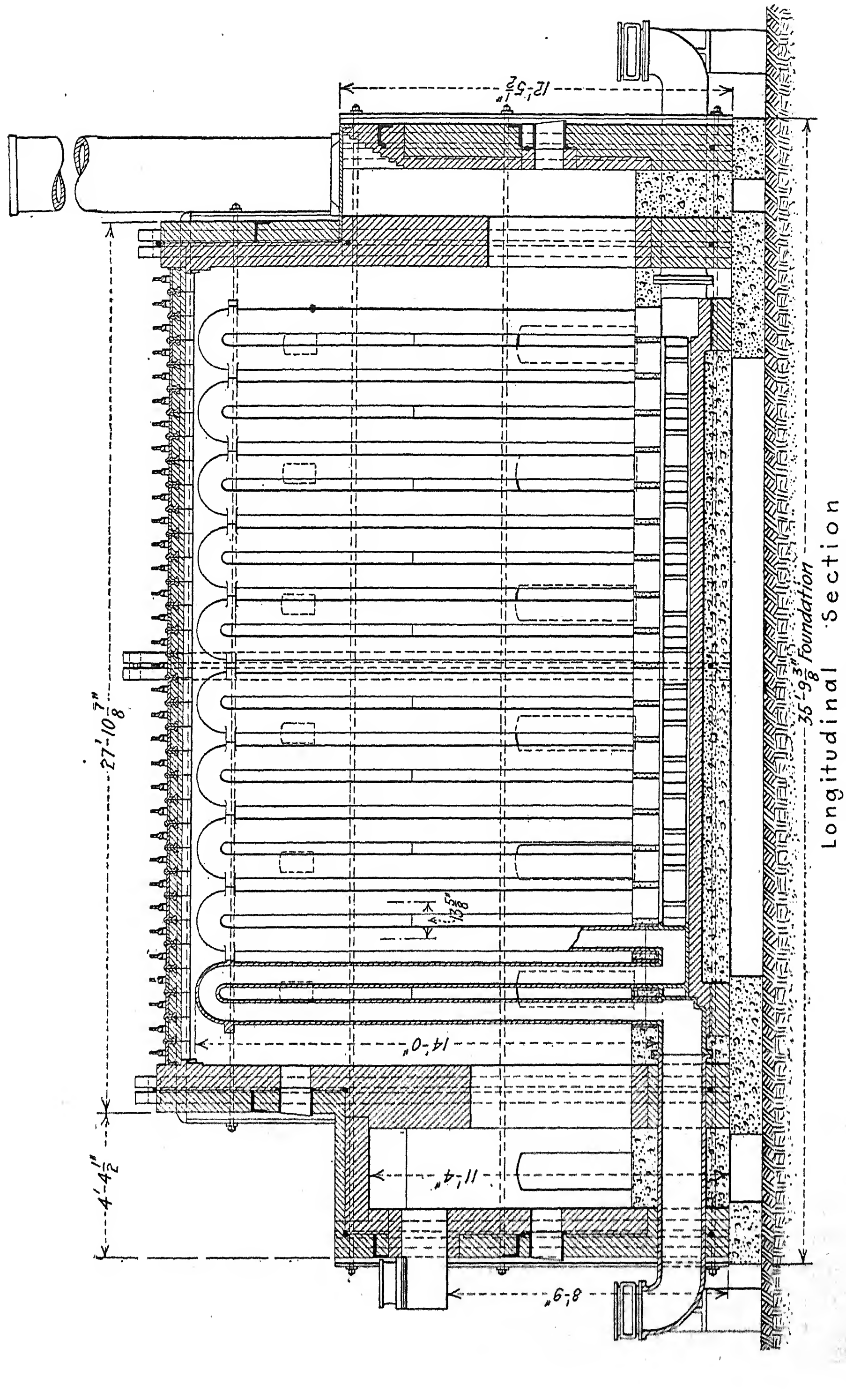


FIG. 712.—Durham pipe-stove.

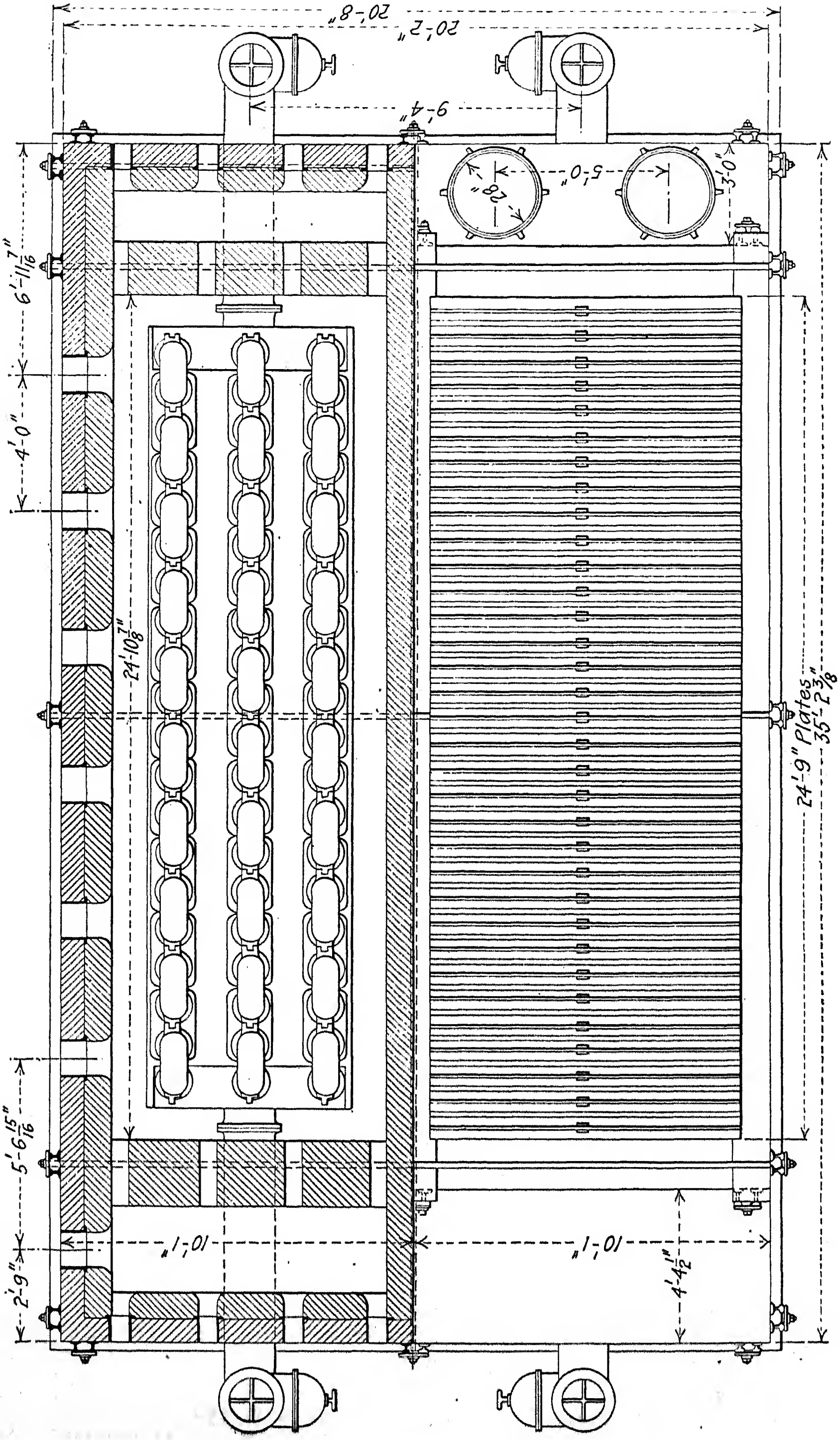


FIG. 714.—Durham pipe-stove.

double stove have a heating-surface of 3,900 sq. ft., or 65 sq. ft. for each pipe. It is safe to say that 1 sq. ft. heating-surface is sufficient for 2 cu. ft. air (piston-displacement) to be maintained at 500 to 540° C.

Vertical stoves having from 6 to 8 superimposed rows of horizontal zig-zag pipes, with or without partition-walls, are called Langen, Westphalian, Lotharingian. They were in operation from about 1850 to 1870; the cold-blast entered at the top, and the burning-gas at the bottom of the heating-chamber. These stoves have the advantage that the counter-current system of heating is more pronounced and hence more efficient than with stoves having vertical pipes; the main disadvantage is that, on account of a tendency to break, the pipes cannot be longer than 10 ft.; it is therefore necessary to have a large number of pipes which results in increased cost of plant and greater consumption of power because of the friction caused by the large number of turns the blast has to make.

Vertical stoves with suspended pipes[1] have been constructed with the expectation that the pipes being able to expand and contract freely would not crack at the bends. It was found, however, that the pipe subjected to tensile strains quickly enlarged any slight crack that had formed. This type of stove has, therefore, been almost wholly abandoned; it shares with that having horizontal pipes the disadvantage that the curved connecting bends lie outside of the heating-chamber.

It has been proposed, to utilize the waste heat of slags from copper blast-furnaces for preheating the blast. Of the different apparatus, those of Bretherton,[2] and Lang,[3] have gained some prominence (see also §213).

The following example gives in outline a calculation of a pipe-stove. The cast-iron pipes are 3 m. long, 25 cm. in inside diam., and 2.5 cm. thick. There are to be heated per sec. 8.8 cbm. of air (standard conditions), from 20 to 450° C.; the blast pressure is 0.7 kg. per sq. cm. (515 mm. above normal). The temperature of the stove is 900° C.; the velocity of the air 6 m. per sec. cold, and 15 m. per sec. hot; the velocity of the gases 2 m. per sec. How many pipes are to be placed in series and how many in a row?

(A) Pipes in Series.—The cross-section of a pipe $=\frac{\pi D^2}{4}=0.7854\times 625=491$ sq. cm. $=0.0491$ sq. m. The volume of air passing through a row = area × maximum velocity $=0.0491\times 15=0.7365$ cbm. per sec. The total volume air per sec. $=8.8$ cbm. at 0° C. and 760 mm. pressure; this $=\frac{8.8\times(450+273)\times 760}{273\times(760+515)}$ $=13.86$ cbm. per sec. at 450° C. and 1,275 mm. pressure. The number of pipes in series will be $\frac{13.86}{0.7365}=19$.

(B) Pipes in a Row.—The number of pipes to be placed in a row is governed

[1] Birkinbine, *Tr. A. I. M. E.*, 1875–76, IV, 208.
Wendt, *op. cit.*, 1886–87, XV, 78; *Circular*, Colorado Iron Works, Denver, Colo.

[2] *Eng. Min. J.*, 1899, LXVIII, 604, 698; 1900, LXIX, 614; LXX, 760; *Min. Sc. Press*, 1900, LXXXI, 572; 1912, CIV, 243.

[3] *Eng. Min. J.*, 1896, LXII, 79; 1900, LXX, 70, 334; 1909, LXXXVIII, 916.

by the rate at which the heat is absorbed by the air in a row. The air passing through each row $=\frac{8.8}{20}=0.463$ cbm. per sec. $=0.599$ kg per sec. With the mean specific heat $=0.244$, the total heat conveyed is $=0.599\times0.244\times(450-20)=62.85$ kg. cal. per sec. The coefficient of transfer from gas to pipe (mean velocity 2 m. per sec.) is $0.000028\,(2+\sqrt{v})$C.G.S. $=0.000028(2+\sqrt{200})=0.0004519$ g. = cal. per sq. cm. $=0.004519$ kg. cal. per sq. m. The coefficient of transfer from pipe to air (mean velocity 10.5 m. per sec.) is $0.000028\,(2+\sqrt{v})=0.000028\,(2+\sqrt{1{,}050})=0.000963$ g. cal. per sq. cm. $=0.00963$ kg. cal. per sq. m. The coefficient of conduction for 2.5 cm. cast-iron is $0.14:2.5=0.056$ g. cal. per sq. cm. $=0.56$ kg. cal. per sq. m. The resistance of the system

$$=\frac{1}{\text{conductance}}=\frac{1}{\frac{1}{0.004519}+\frac{1}{0.00963}+\frac{1}{0.56}}=\frac{1}{221.3+103.7+1.78}=\frac{1}{326.8}$$

$=0.00306$ k. cal. per sq. m. of surface per deg. C.

The difference in temperature outside and inside of a pipe is at entrance $900-20=880°$ C, at exit $900-450=450°$ C.; the mean difference $=\frac{880\times450}{2}$ $=665°$ C.

Let x = surface required, then $x\times0.00306\times665=62.85$ kg. cal.; $x=30.88$ sq. m. The surface area of each pipe equals inner surface times length $=\pi Dl=3.1416\times0.25\times3=2.36$ sq. m. The number of pipes in a row $=\frac{30.88}{2.36}=13$ pipes. As 19 pipes in series is inconvenient to heat in one furnace, there will be required 2 stoves, one with 10 pipes in series and 13 in a row, and the other with 10 and 13 respectively.

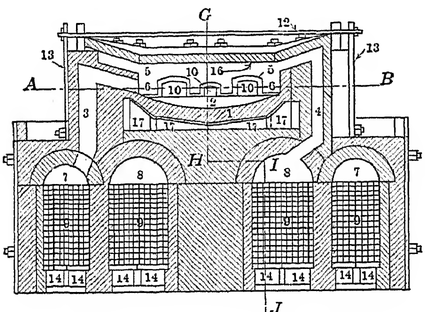
324. Siemens (Regenerator) System in General.—The characteristic of pre-heating in the regenerator system, invented by F. Siemens[1] in 1856, is that it is intermittent with two alternating operations: conducting hot waste gas through a chamber filled with loosely-set refractory material in order to heat this and cool the gas, and then conducting air or fuel-gas in the opposite direction through the same chambers that it may take up the heat stored in the refractory material and thus become pre-heated. The manner in which this principle of pre-heating is carried out with a reverberatory furnace differs greatly from that with the blast-furnace. With a reverberatory furnace the heat contained in the products of combustion is stored in the chambers; with a blast-furnace, the tunnel-head gas which has fuel-value is burnt in the chambers; with a reverberatory furnace fired with producer gas, both gas and secondary air are drawn or forced through the chambers; with the blast-furnace air alone is blown through them.

325. Siemens Regenerator System in the Reverberatory Furnace.—The regenerative system is used in most reverberatory furnaces in which high temperatures are to be obtained; it is used also in furnaces not demanding

[1] Editorial, *Stahl u. Eisen*, 1904, XXIV, 731.
Beck, *op. cit.*, 1906, XXVI, 1421.
Le Chatelier, *Rev. Mét.*, 1907, IV, 212.

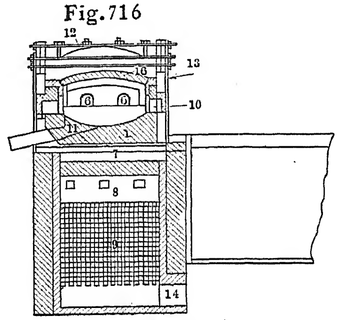
high temperatures on account of the satisfactory recovery of heat, and this in spite of the facts that producer gas has first to be cooled, involving a loss of 30 per cent. of its heat-value, to drop its tarry compounds which would clog the regenerators, that flue dust is liable to slag the refractory material, and that the cost of plant is very high. The manufacture of open-hearth steel

Fig. 715



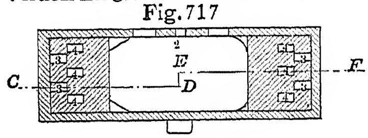
Vertical Longitudinal Section on *C D E F*

Fig. 716



Vertical Cross Section on *G H I J*

Fig. 717



Horizontal Section on *A B*

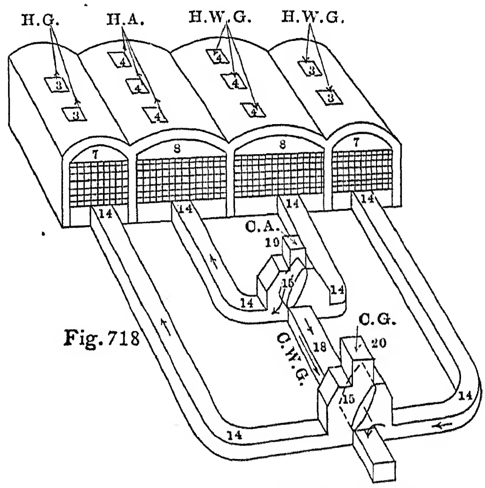
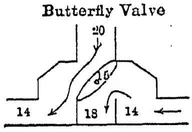


Fig. 718

Fig. 719

Butterfly Valve



FIGS. 715 to 717.—Siemen's open-hearth steel furnace. FIG. 718.—Perspective view of Siemen's regenerators. FIG. 719.—Butterfly-valve.

FIGS. 715 to 719.—1, Hearth-bottom; 2, hearth; 3, flues leading from gas-chamber to hearth; 4, flues leading from air-chamber to hearth; 5, air-ports; 6, gas-ports; 7, gas regenerator-chamber; 8, air regenerator chamber; 9, checker-work; 10, charging-doors; 11, tap-hole with spout; 12, tie-rods; 13, buckstaves; 14, flues to and from reversing-valves; 15, butterfly-valves; 16, roof; 17, bottom-plates; 18, flue to stack; 19, flue from cold-air inlet; and 20, flue from producer-gas inlet.

by Martin in 1865 became successful only by the use of regenerators; most steel-heating and -smelting furnaces have regenerators; some zinc and copper-smelting furnaces are similarly provided; the Otto-Hoffmann by-product coking-oven, Figs 159 and 160, is a representative of this type.

The general arrangement and mode of operating are shown in connection

with Figs. 715 to 717, based upon drawings by Campbell,[1] which represent the old standard type of open-hearth steel furnace. Fig. 718, by Richards[2] shows the travel of gas and air, and Fig. 719, the butterfly-valve.

If the furnace is in normal working order, cold gas (C.G.) and cold air (C.A.) enter through flues 20 and 19, respectively, above the butterfly-valves 15, travel through 14 and ascend in checkerwork, 9, of the gas, 7, and air, 8, regenerative chambers, take up heat stored there, leave the chambers as hot gas (H.G.) and hot air (H.A) through flues 3 and 4, uniting at their terminations 5 and 6, and burn. The flame passes over hearth 2 and leaves through flues 3 and 4 at the right. The hot waste gases (H.W.G.) descend through checkerwork 9, of regenerative chambers 7 and 8, at the right, leave them through 14, pass underneath the butterfly-valves 15, enter as cold waste gas (C.W.G.) flue 18 leading to the chimney.

After from 1/2 to 1 hr.[3] valves 15 are reversed, and gas and air now travel in the opposite direction, from right to left. With every reversal of the gas and air currents the temperature will be raised because the amount of heat stored in the checkerwork is being constantly increased, and because a nearer approach is made to perfect combustion with a smaller excess of air over the theoretical amount. The absorption of heat by the charge and the loss by radiation usually prevent overheating; it is, however, necessary to regulate the admission of gas and air in such a way that the temperature of the regenerators shall not exceed 1,000° C. Le Chatelier's measurements[4] in a steel open-hearth furnace gave the gas leaving the regenerator 5 min. after reversal of valves 1,200° and the air 1,070° C.

It has been shown (§311) that the draft of a chimney is at its maximum with an average gas temperature of 273° C., and that there is little difference in draft-power within the range of 200 and 300° C.; hence in a steel smelting furnace the checkerwork ought to be large enough to cool the gases from 1,500[5] to 1,600 down to 200° C.; the larger the regenerator the more slowly will it give up the heat stored in it, and the less frequent need be the reversing of the valves. Grüner[6] calculates that with producer gas a pair of regenerators should contain 50 kg. brickwork (specific heat 0.23–0.25) for each kg. bituminous coal gasified between two reversals of valves; generally 60 kg. are given with 1 hr. between two reversals. Now 60 kg. fire-brick (sp. gr. 1.8) have a surface of 0.033 cbm. and, as only half of the surfaces form flues, $2 \times 0.033 = 0.066$ cbm. per kg. coal, or 2,115 cu. ft. per short ton coal gasified per hr. will be required, or approximately 1 cu. ft. per lb. coal per hr. The cross-section of regenerators must be large to prevent the gases from rushing through. For every 2,000 lb. bituminous coal

[1] *Tr. A. I. M. E.*, 1893, XXII, 358.

[2] "Notes on Iron," Institute of Technology, Boston, 1895, p. 104.

[3] Juon: Graphical Records of Temperature Changes, *Stahl u. Eisen*, 1912, XXXII, 1774, 1869, 1873.

[4] Howe, *Eng. Min. J.*, 1890, L, 428.

[5] Le Chatelier's measurements (*loc. cit.*) gave 1,500° C. as the temperature of a steel smelting furnace at the moment of casting.

[6] Traité de Métallurgie, I, p. 383.

gasified per hr. a pair of regenerators should have a cross-section of 250 to 300 sq. ft.; with lignites 190 to 200 sq. ft. is ample. If 1 kg. of bituminous coal is burned with 20 to 25 per cent. excess air over the theoretical, it yields 11 cbm. of products of combustion at 0° C. or 48.3 cbm. at 1,200° C. With 100 kg. (220 lb.) coal per hr. and a temperature of 1,200° C. of products of combustion, 1.3 cbm. (46 cu. ft.) of gases will pass through 1.24 sq. m. (13 sq. ft.) of open spaces in the checkerwork at a rate of 3.28 ft. per sec. The cross-section of the gas-chamber is made smaller than that of the air-chamber, viz., 3 : 4 or 2 : 3, although the volume of air required is approximately the same as that of the producer gas. The reason for this is that the gas entering the regenerator is already warm while the air is cold.

Modern forms of open-hearth steel furnaces and details of construction belong to a discussion of the manufacture of open-hearth steel. Lebedeff-Pomeranzeff have constructed an open-hearth furnace for solid wood with regenerative chambers for heating the air.[1]

326. Siemens Regenerator System in the Blast-furnace.—The regenerative apparatus serving to pre-heat the air for the blast-furnace goes by the name of brick-stove.[2] It was introduced into iron smelting by Cowper in 1860.[3] The blast-temperature can be raised by it to 800° C.

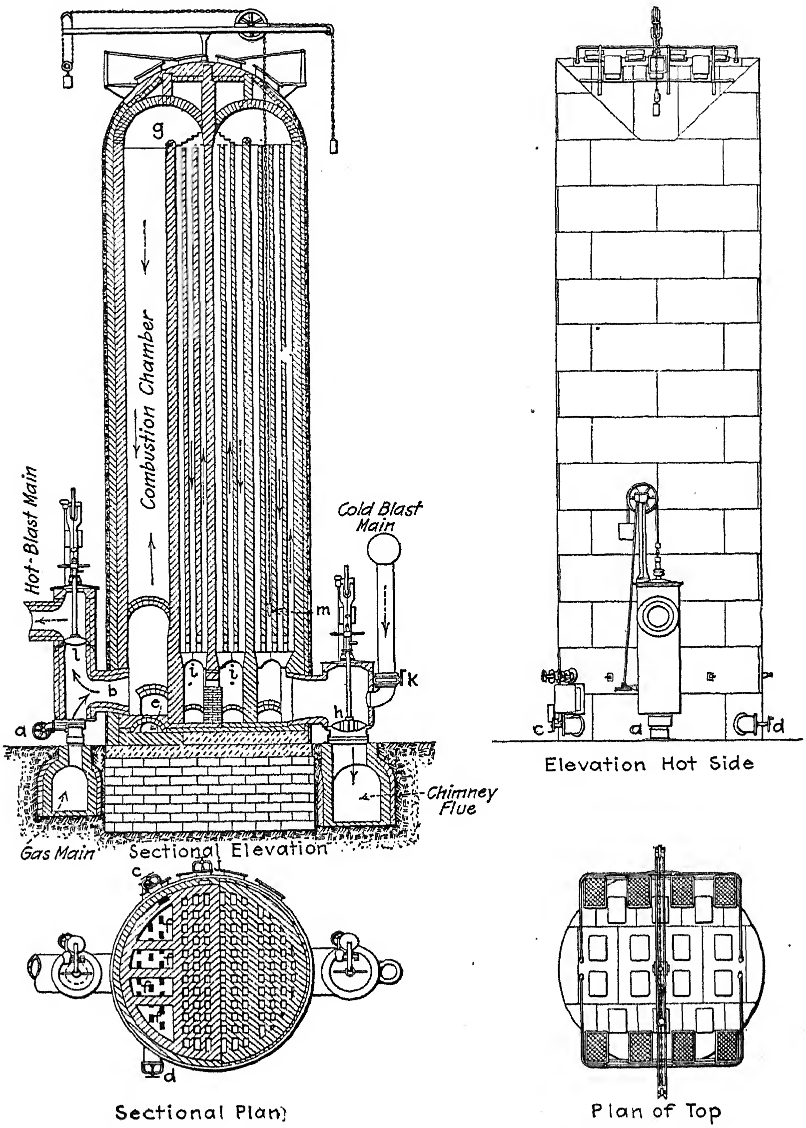
Hartman[4] states with its use in iron smelting, by replacing pipe-stoves heated to 480° C. with brick-stoves heated to 760° C., a saving of 15 per cent. of fuel and an increase of 20 per cent. in the production of iron has been effected.

A brick hot-blast stove is a cylinder of boiler iron with a dome-shaped top, filled with fire-brick of different shapes and forms having passages through which gases or air may pass freely. Between the iron shell and the brickwork there is an expansion space, 1 1/2 to 2 in. wide, filled with loose material. In iron-smelting brick-stoves are mounted in groups of 3 and even 5 for a furnace, Fig. 738. There are two periods in operating them, one of storing up heat and one of giving out heat. When two or three stoves are being heated by burning gas in them ("they are on gas"), one is giving out the stored heat to the cold-blast passing through it ("is on air"); gas and air travel alternately in opposite directions. From 5 to 6 sq.ft. of heating surface are necessary for 1 cu. ft. of air per min., the velocity of the cold air being 6 to 6 1/2 ft., that of the hot air 16 to 26 ft. per sec. Stoves are 18 to 24 ft. in diam. and about 60 ft. high; in the last few years the height has been increased to 80 and even 100 ft. The fire-brick[5] must be refractory, strong and porous; should they become glazed by heat alone or by the fuel dust (gas-ashes), which is rich in iron, they would practically cease to absorb any heat. The construction of a stove must be such as to permit a ready

[1] *Metallurgie*, 1910, VII, 332.
[2] Diehl, *Iron Age*, 1912, LXXXIX, 580; *Eng. Mag.*, 1912, XLIII, 265.
[3] Teichgräber, *Stahl u. Eisen*, 1902, XXII, 323.
[4] *J. Frankl. Inst.*, 1886, CXXI, 321.
[5] Lürmann, *Stahl u. Eisen*, 1901, XXV, 785.
Greaves-Walker, *Iron Trade Rev.*, Aug. 23, 1906, p. 17.
Harbison-Walker, *op. cit.*, 1909, XLIV, 111.

removal of these ashes, which are always carried along by the gases to a greater or less extent.

All brick-stoves may be considered as belonging to one of two types: the



FIGS. 720 to 723.—Whitwell hot-blast stove.

Whitwell, in which gas and air make more than two passes; and the Cowper, in which they make two passes only.

1. The Whitwell Stove.[1] The Whitwell stove has undergone many changes

[1] Gordon, *Tr. A. I. M. E.*, 1881, IX, 480.

in the details of construction since the first was erected in 1869. Figs. 720 to 723 represent a four-pass stove (diam., 17 ft.; height, 60 ft.; heating surface, 9,000 sq. ft.), constructed in 1881 by Witherow and Gordon of Pittsburgh, Pa. Gas or air makes four passes before it enters the flue or the hot-blast main. To the left of the stove is the combustion-chamber. The burning-gas coming from the gas-main passes through sliding valve, *a*, enters the combustion-chamber through a bell-mouthed opening *b*; here it meets air which has been drawn in through valves, *c* and *d*, into the vault, *e*, and thence through the vertical ports, *f*; the gas becomes thoroughly mixed with the air and burns quickly and any imperfect combustion in the chamber is finished in dome, *g*. The products of combustion zig-zag through the oblong flues of the checkerwork in the three chambers, as shown by the full-drawn arrows, and pass off into the chimney-flue through water-cooled puppet-valve, *h*. The chambers rest on brick arches leaving open spaces beneath which serve as dust-chambers; they have peep-holes, *i*, for watching the interior. The gas when burning expands; later it contracts while heat is being absorbed by the checkerwork; the flues in the third regenerator are therefore smaller than those in the first and second. Instead of burning all the gas in the combustion-chamber, only part of the air is admitted here and the rest through valve *j*, at the bottom of the first regenerator. This is called burning back and is accompanied by a higher temperature.

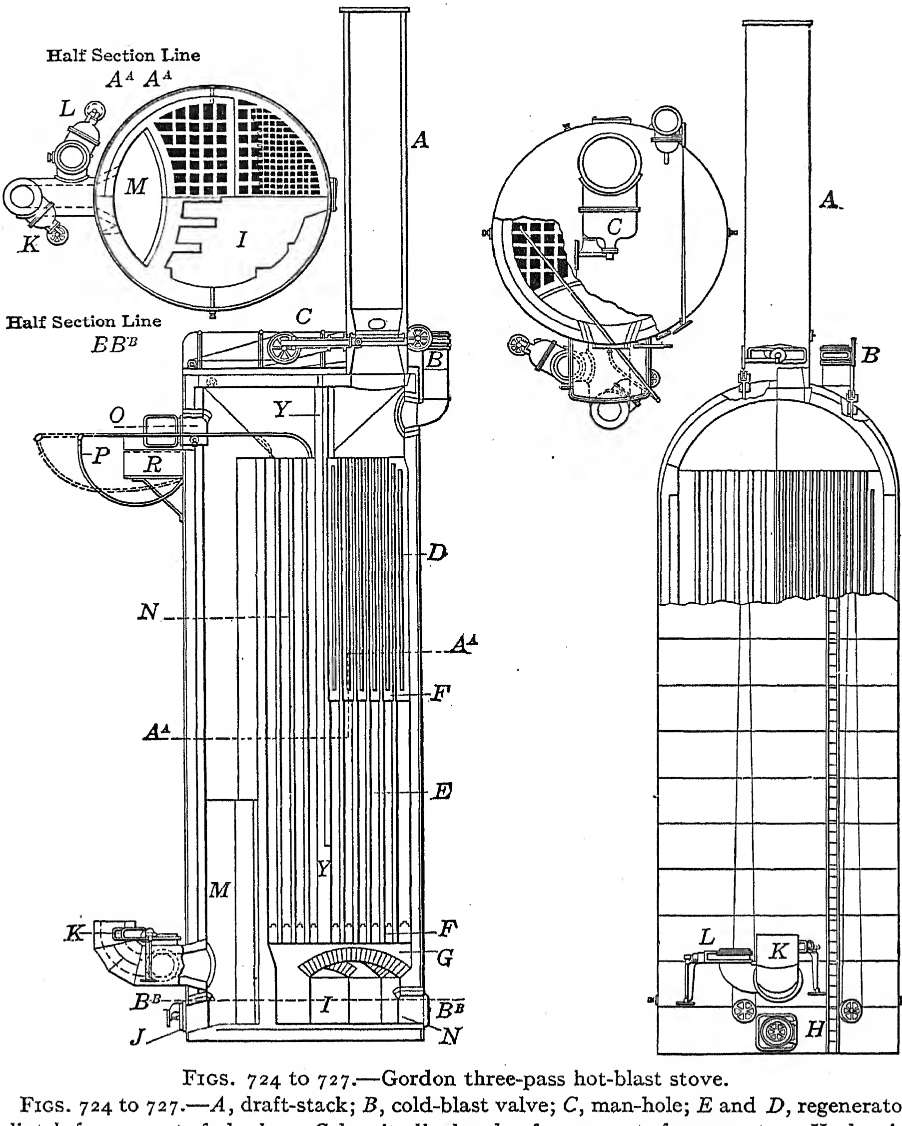
When a stove has been heated and is to be put on air, gas-valve *a*, air-valves *c* and *d*, and chimney-valve *h* are closed, and then gate-valve *k* in the cold-blast main and water-cooled puppet-valve *l* in the hot-blast main opened. The cold-blast going in the opposite direction of the products of combustion, as shown by the dotted arrows, ascends through the smaller flues in the 3d regenerator, then zig-zags through the 2d and 1st regenerators taking up more and more heat, finally descends in combustion-chamber, and passes off through flue *b* and hot-blast puppet-valve *l* into the hot-blast main.

The vertical section, Fig. 720, shows the cleaning-device, a suspended scraper, *m*, which is raised and lowered in the flues. The main advantage of the Whitwell stove over other forms is that the flues are easily cleaned; the disadvantages, however, are so great that very few, if any, stoves of this class are built at present. For a given cubical capacity of stove, the amount of brickwork is large, the heating surface small, and the friction great, requiring a high chimney and causing much back-pressure[1] upon the blowing-engine.

2. The Gordon Three-pass Stove.[2] Figs. 724 to 727 represent horizontal and vertical sections and in part plan and elevation of a 23×60 ft. stove with 28,000 sq. ft. heating surface. The gas is admitted through valve, *L*, into combustion-chamber, *M*; the products of combustion descend through regenerator, *N*, into chamber, *I*, ascend through regenerator flues, *E* and *D*, and pass off into the open through stack, *A*. In order to secure the same heating-surface as a 4-pass Whitwell stove, the number of flues, *D*, in the upper part of pass, *D*, has

[1] Grüner, *Ann. Mines*, 1872, II, 295.
[2] Strobel, *Tr. A. I. M. E.*, 1885–86, XIV, 159.

been increased and the thickness of brickwork correspondingly decreased. Each stove has an individual stack; this does away with the underground flues and the large central stack of a set of Whitwell stoves.



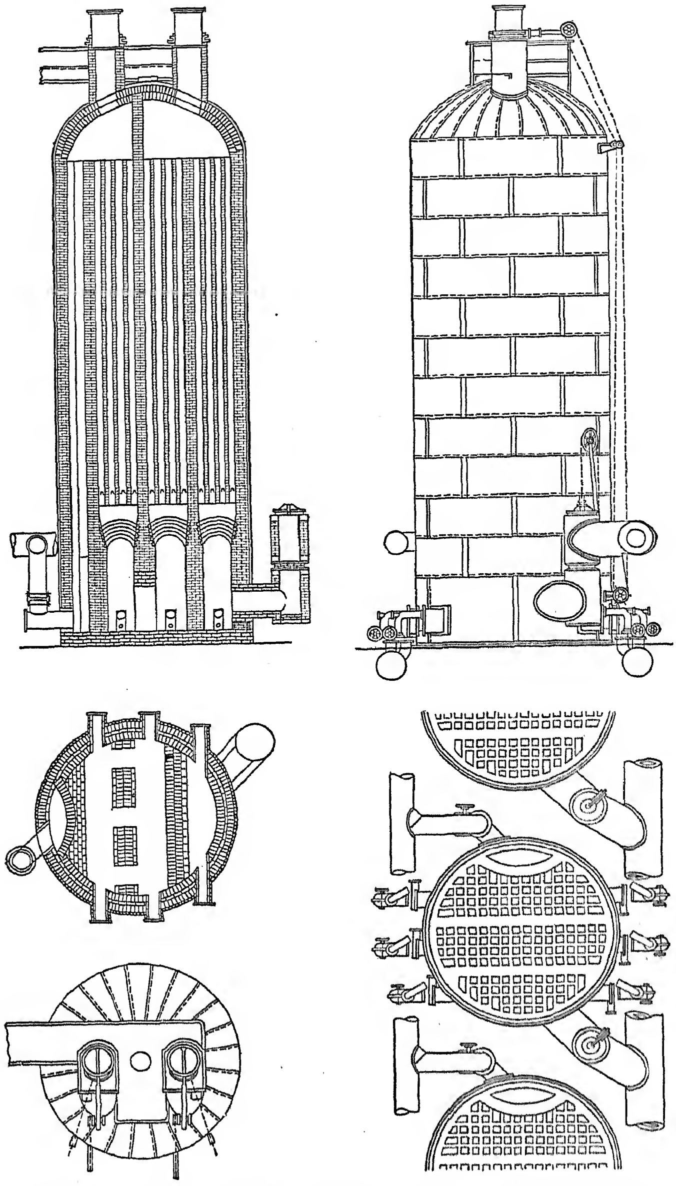
FIGS. 724 to 727.—Gordon three-pass hot-blast stove.

FIGS. 724 to 727.—*A*, draft-stack; *B*, cold-blast valve; *C*, man-hole; *E* and *D*, regenerators; *F*, lintels for support of checkers; *G*, longitudinal arches for support of regenerators; *H*, cleaning-door; *I*, chamber; *K*, hot-blast valve; *L*, gas inlet-valve; *M*, combustion-chamber; *N*, regenerator flue; *O*, cleaning-hole; *P*, gas-pipe for cleaning regenerator *N*; *R*, platform; and *Y*, a division-wall.

3. The Hugh Kennedy One-pass Hot-blast Stove.[1] Figs. 728 to 731 represent a stove 16×45 ft. with 57,840 sq. ft. heating-surface. The burning-gas

[1] Coffin, *Tr. A. I. M. E.*, 1892–93, XXI, 720.

arrives in two underground mains, Fig. 729, enters the stove near the bottom through six Spearman (see Fig. 751) burners, Fig. 728, 730, and 731, and burns in three chambers. The products of combustion rise in these chambers,

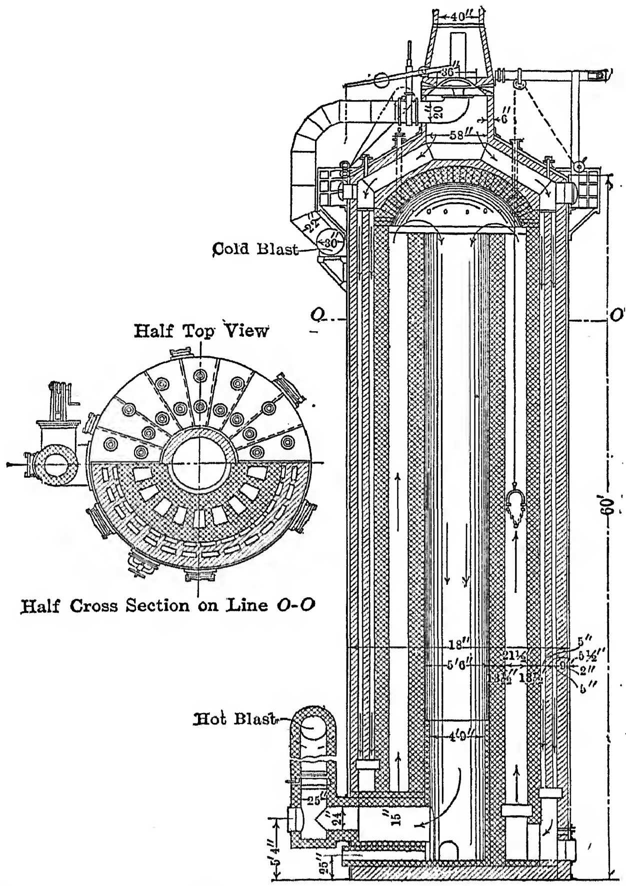


FIGS. 728 to 731.—Hugh Kennedy one-pass hot-blast stove.

make a single pass and leave through two small stacks, Figs. 728 and 729, provided with dampers. The cold-blast enters at the left of Fig. 728, makes four passes, as in a Whitwell stove, and leaves at the right as hot-blast. The

advantages claimed for this construction are, cheapness, uniformity of tempera ture, and freedom from dust troubles. A doubt is expressed whether the hea of the products of combustion is as well utilized as in other types.

4. The Massicks and Crooke Three-pass Hot-blast Stove.[1] Figs. 732 an 733 represent a stove 18×60 ft. with a heating surface varying from 35,000 t 45,000 sq. ft. depending upon the purity of the gas. The burning-gas enter the combustion-chamber through the inlet near the bottom and burns; th



FIGS. 732 and 733.—Massicks and Crooke three-pass hot-blast stove.

products of combustion rise, are deflected by the dome and make a down-pas in the first row of annular flues, turn, rise through flues of a double annula chamber and pass off at the top.

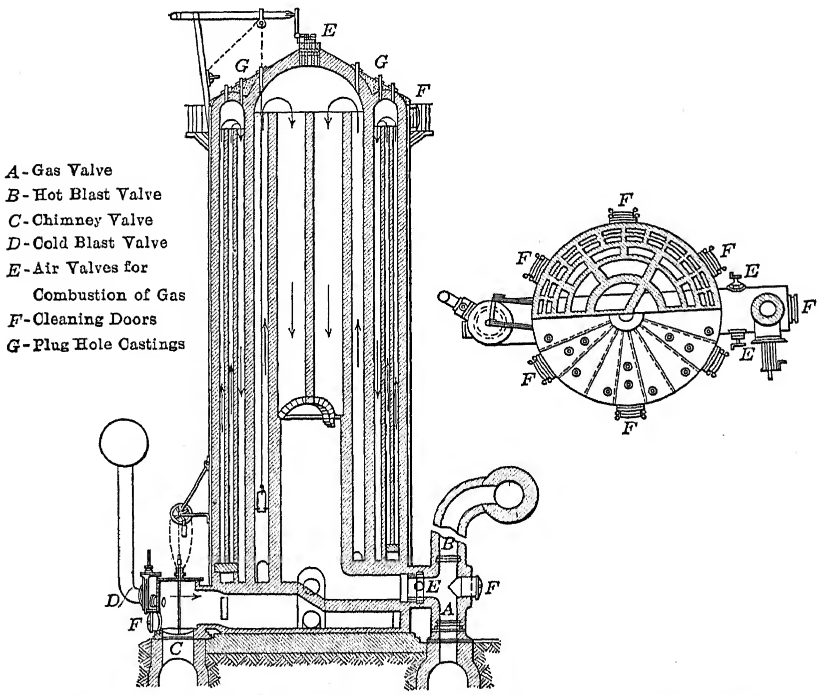
The cold-blast traverses a slide-valve, enters the stove at the top, travel in a direction opposite to that of the gases, as shown by the arrows, and leave the stove through the hot-blast pipe near the bottom.

A more recent form is shown in Figs. 734 and 735. This is a 4-pass stove

[1] Crooke, *Tr. A. M. E.*, 1890–91, XIX, 1036. *J. I. and St. I.*, 1890, II, 340.

the legend denotes the different parts. Another 4-pass stove is that of the Anhalt Maschinen Actien Gesellschaft.[1]

5. The Cowper-Kennedy Hot-blast Stove with Lateral Combustion-chamber.[2] In the Whitwell stove,[3] 70 per cent. of the heat of the products of combustion of the burning gas is taken up by the combustion chamber and the first regenerator. If the heating surface of the regenerator be sufficiently large, one down-pass should be enough to absorb practically all the heat. This is the ground-idea of the Cowper type of stoves, all of which have two passes. Figs. 736 and



FIGS. 734 and 735.—Massicks and Crooke four-pass hot-blast stove.

737 represent a stove 21 × 72 ft. 6 in. constructed for the iron blast-furnaces *D* and *E* of the Edgar Thomson Steel Works having a heating surface of 30,200 sq. ft.

The bricks forming the stove are hexagonal; they are 10 in. across and have circular openings 6 in. in diam. In later constructions[4] these dimensions have been increased to 12 in. across and 9 in. in diam. The bricks are 12 in. deep, except in the bottom and top courses where they are 6 in. deep.

6. The Cowper-Kennedy Stove with Central Combustion-chamber.[5] Figs.

[1] *Iron Age*, 1912, XC, 916.

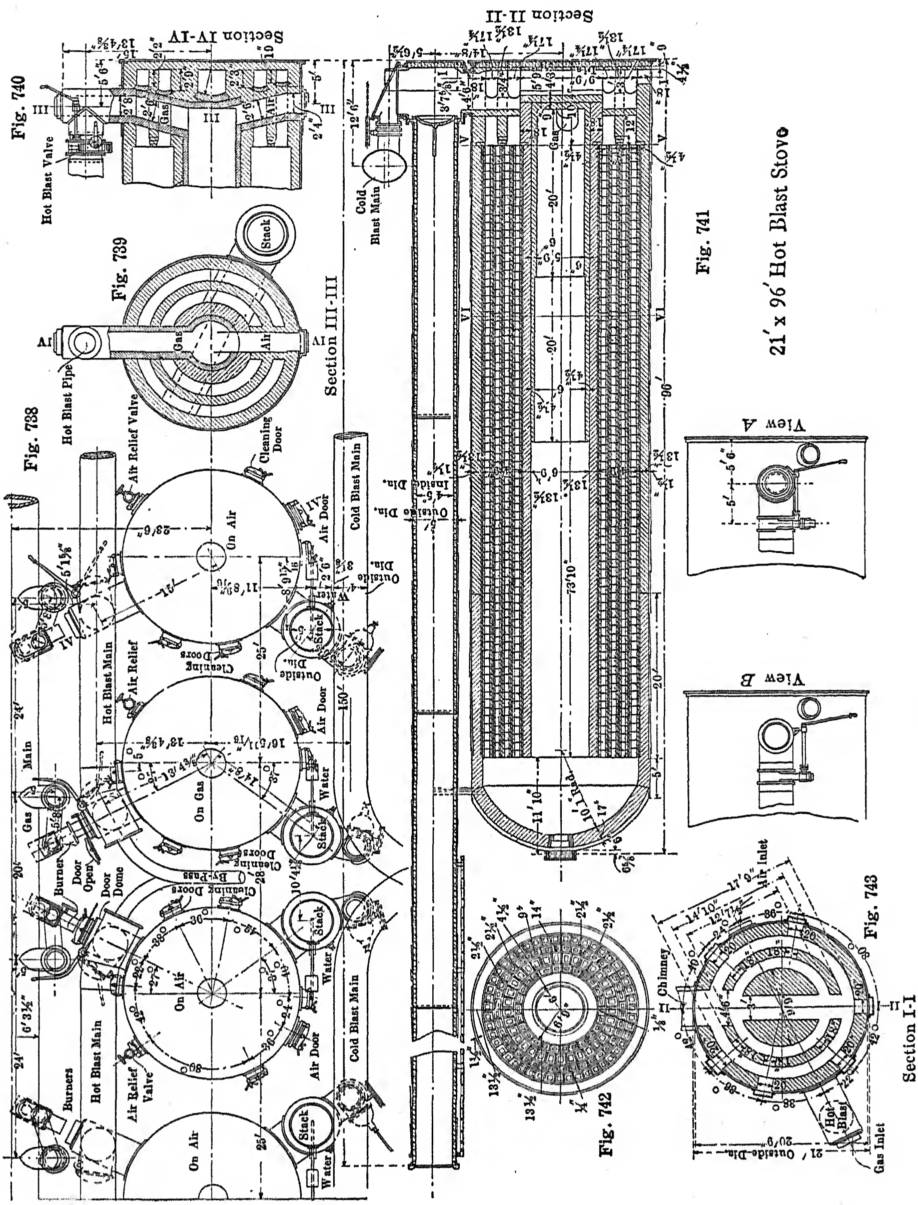
[2] Kennedy, J., *Tr. A. I. M. E.*, 1881–82, X, 495.

[3] Coffin, *op. cit.*, 1892–93, XXI, 720.

[4] *Iron Age.*, 1890, XLVI, 43.

[5] Duquesne Plant, Pittsburgh, Pa., *Iron Age*, March 25, 1897, p. 10; Buffalo & Susquehanna Iron Co., Buffalo, N. Y., *op. cit.*, Aug. 6, 1903, p. 1; Republic Iron & Steel Co., Youngstown, O., *Iron Trade Rev.*, 1907, XL, 621.

738 to 743 represent the four stoves serving a blast-furnace of the Duquesne Iron-smelting Plant, Pittsburgh, Pa. Fig. 738 gives a general plan. Figs. 739 to 743 show the details of a single stove. A stove is 27×97 ft. and has a heat-

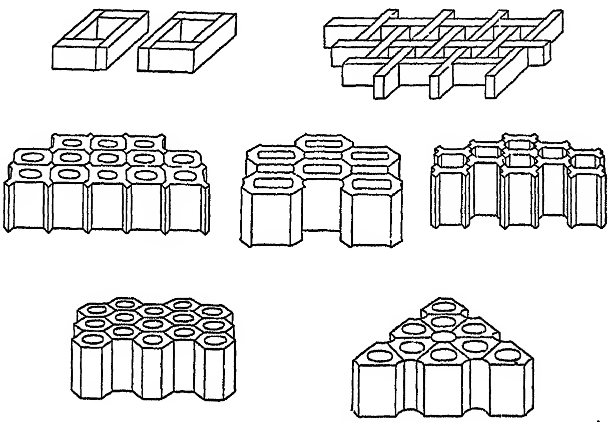


FIGS. 738 to 743.—Cowper-Kennedy hot-blast stove with central combustion chamber.

ing surface of 32,035 sq. ft.; the tiles are 14 in. deep and 12 in. high; they have openings approximately 9 in. square with filleted corners; the walls are 2 1/2 in. thick. Owing to local conditions each stove has an independent chimney

which is 130 ft. high and 5 ft. in diam.; a chimney is provided with a valve resting on a bronze water-cooled seating-ring. The passage of burning-gas and air in heating, and of cold- and hot-blast are indicated in the drawings. A similar stove built in 1903 for the Buffalo & Susquehanna Iron Co., Buffalo, N. Y.,[1] is of smaller diameter and higher, viz., 22×102 ft.

7. Besides these leading types there may be mentioned[2] the forms of Stevenson-Evans, Cabot-Patterson, Foote, Kloman, Nelson, Foote-Lamond, Amsler, Ford-Moncur, Moore and Hartman-Kennedy. In the last an external combustion-chamber (12 ft. long by 5 ft. diam. for a 19×76 ft. stove) has been attached to the stove to make the combustion of the burning-gas perfect, to



FIGS. 744 to 750.—Forms of checker brick.

provide a place for the deposition of dust whence it is readily removed by blowing. The stove proper is a Hartman 2-pass stove with partition-wall in the center.

8. The bricks[3] of the different stoves show a great variety of forms[4] and ways of joining so as to give large and smooth surfaces, and to allow for expansion and contraction. Some of these are shown in Figs. 744 to 750; others have been given in the preceding figures and references.

9. The burners used have different forms. Fig. 751 illustrates a Spearman-

[1] *Iron Age*, Aug. 3, 1903, p. 4.

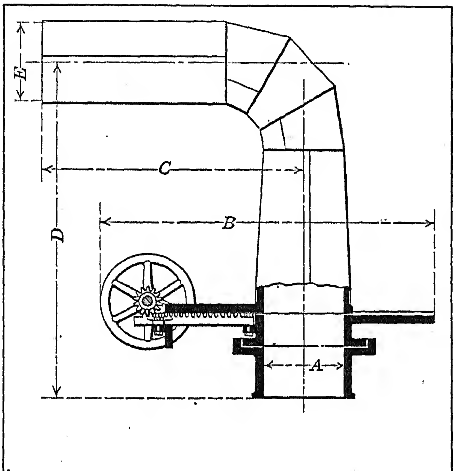
[2] (Stevenson-Evans), *Eng. Min. J.*, 1899, LXVII, 591; (Cabot-Patterson), *Iron Age*, Oct. 10, 1901, p. 3; (Foote), *Am. Mfr.*, 1901, LXVIII, 138; (Kloman), *Iron Trade Rev.* Apr. 16, 1903, p. 44; (Nelson), *Iron Age*, 1909, LXXXIV, 194; *Stahl u. Eisen*, 1909, XXIX, 1457; (Foote-Lamond), *Iron Age*, 1909, LXXXIV, 1159; (Amsler), *Iron Trade Rev.*, 1907, XLV, 887; (Ford-Moncur), *J. I. and St. I.*, 1896, I, 20; (Moore), *Iron Age*, Feb. 12, 1903, p. 10; (Hartman-Kennedy), *op. cit.*, Aug. 18, 1904.

[3] Canaris, *Stahl u. Eisen*, 1911, XXXI, 1245.

[4] Lürmann, *op. cit.*, 1884, IV, 484; 1890, X, 766; 1896, XVI, 907.
Stevenson-York, *Iron Age*, 1892, XLIX, 769.
Roberts, *op. cit.*, Jan. 9, 1896, p. 139.
Hall, *J. I. and St. I.*, 1896, I, 20, Plates II and III.
Foote-Lamond, *Iron Age*, 1907, LXXXIV, 1159.

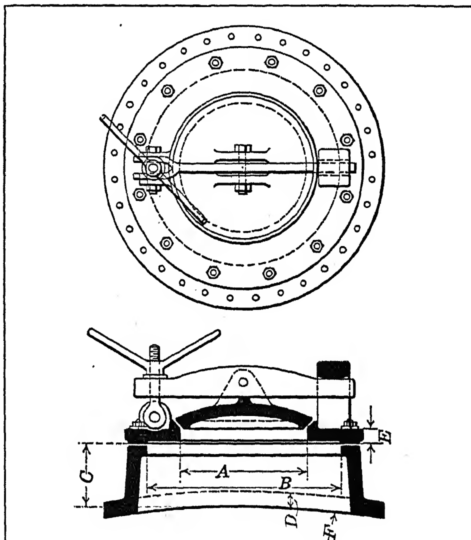
Kennedy burner with valve.[1] The heavy lines represent castings machined on the top surfaces. The valve is a sliding cast-iron plate machined on the lower surface. It is actuated by rack and pinion. Upon one end of this plate is cast a port-opening to which is attached the wrought-iron elbow. In the figure the port is open; when closed, the solid part of the valve covers the port.

10. The valves found in a stove are the air valves, cold- and hot-blast valves, and gas-escape valves. Figs. 752 to 753 represent an air- or gas-inlet valve of cast-iron, with heavy flanges; the spherical valve is firmly held in the beveled



A	B	C	D	E
12″	39″	27″	34″	8″
14″	58″	45″	57″	14″
18″	70″	50″	57″	18″

FIG. 751.—Spearman-Kennedy burner.



A	B	C	D	E	F
16″	24″	8″	1½″	1¾″	Any radius
20″	24″	8″	1½″	1¾″	"

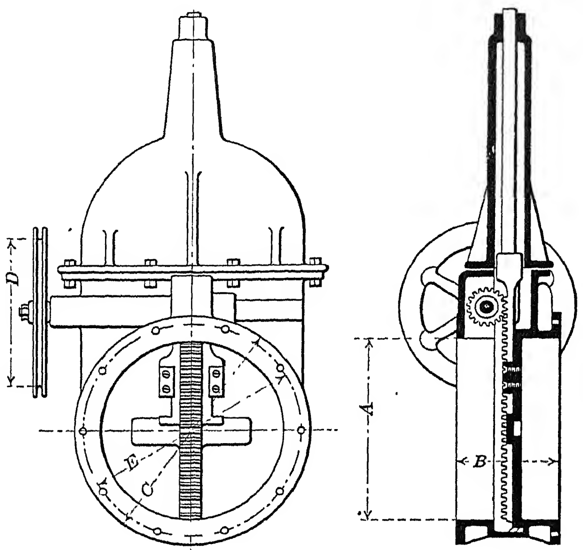
FIGS. 752 and 753.—Spearman-Kennedy air- or gas-valve.

seat by a cast-iron bar secured to the frame by a fixed leg and a link bolt. Cleaning-doors are of the same general construction. The cold-blast valve shown in Figs. 754 and 755 is of the gate type and is operated by means of rack and pinion. Hot-blast valves are either sliding or puppet valves. The sliding valves are similar to the cold-blast valves, except that the gate and the seat are air- or water-cooled, being cast around a course of 1-inch pipe. The puppet valves, Figs. 756 and 757 are usually water-cooled. The seats are made of bronze and are hollow, and air- or water-cooled. The valve-stem is of seamless drawn tubing. The gas-escape valve, Figs. 758 and 759, for the hot-blast main serves to carry off the gas which backs up from the furnaces when the blast is shut off. The valve,

[1] Other forms: Moore, *Iron Age*, Feb. 12, 1903, p. 10.

being heavier than the lever, is held in position by the pressure of the blast; when this is taken off it falls, and when the blast is put on[1] it closes up again.

11. In operating a stove the gas is admitted through the gas valve. After it has burned 3 to 4 hr. the gas and chimney valves, the burner and the air-inlet openings are closed, the cold-blast valve and then the hot-blast valve are opened. The stove now remains 1 1/2 to 2 hr. on air. The next stove is then



A	B	C	D	E	Thickn's Body	Thickn's Flange
10″	8″	15½″	10″	13½″	7/16″	5/8″
17″	13″	22¼″	20″	20¼″	½″	13/16″
20″	13″	25¾″	20″	23¾″	½″	7/8″
22″	13″	27½″	20″	25½″	½″	13/16″
22″	13 5/8″	27 7/8″	20″	25 7/8″	¾″	1 1/8″
24″	13″	29¼″	20″	27¼″	5/8″	1″
24″	13″	29¼″	20″	27¼″	¾″	1″
26″	13″	31¾″	30″	29¼″	5/8″	1″

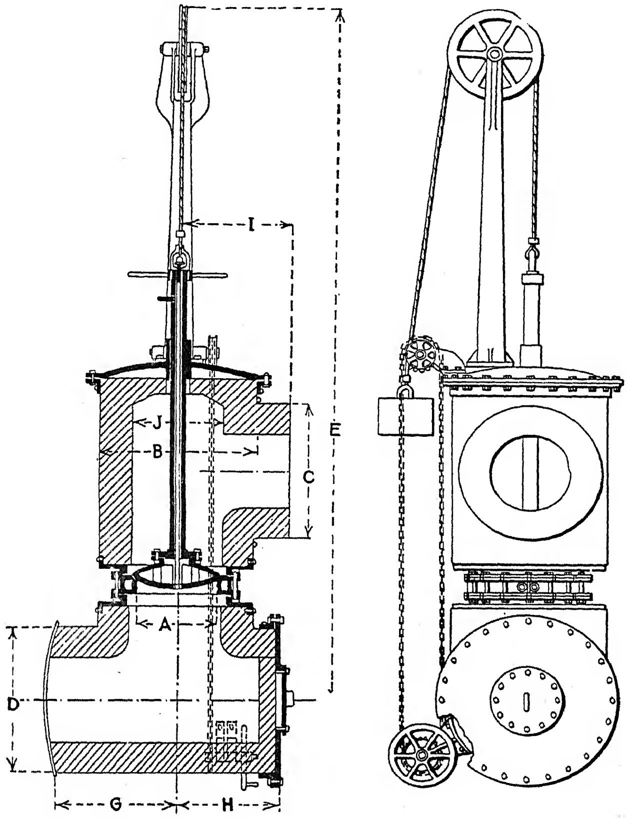
FIGS. 754 and 755.—Gate cold-blast valve.

cut out from the blast-furnace (the hot-blast, then the cold-blast valve is closed and the chimney valve opened) and the stove put on gas, *i.e.*, the air- and gas-inlet valves are opened. The temperature of the blast should not vary over 10° C. between the changes.

12. In order to do away with all variations of temperature of hot-blast,

[1] See also Kennedy, *Iron Age*, Feb. 1, 1900, p. 16; *Iron Trade Rev.*, Feb. 1, p. 12. Rust, *Iron Age*, 1907, LXXX, 432.

Gjers[1] constructed a hot-blast equalizer for the Ayresome Iron Works, Middlesbrough, England. It is, Fig. 760, a brick-stove with checkerwork having a vertical partition wall passing through the center. The hot-blast from a freshly heated stove passing up one-half the checkerwork and down the other gives up some of its excess of heat to the checkerwork before it passes on to



A	B	C	D	E	F	G	H	I	J
24"	45"	42"	45"	8'-4"	9'-9"	34½"	30"	32½"	28"
24"	45"	45"	42"						

FIGS. 756 and 757.—Puppet hot-blast valve.

the furnace. When the temperature of the hot-blast tends to sink below that of the checkerwork of the equalizer, it takes up the excess of heat it had given up before and leaves the equalizer at a normal temperature. The equalizer thus forms a large reservoir for heat, correcting any variation of temperature of the hot-blast. Another form is that of Hartman-Kennedy.[2]

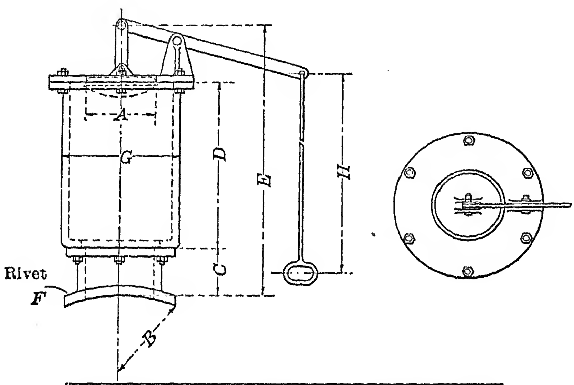
[1] Gjers-Harrison, *Eng. Min. J.*, 1899, LXVIII, 337; *J. I. and St. I.*, 1900, I, 154; 1902, II, 282; *Iron Age*, Aug. 27, 1903, p. 30.

Hartman, *op. cit.*, May 19, 1904, p. 15.

[2] *Iron Age*, Aug. 18, 1904, p. 8.

13. At certain intervals the flues in brickwork have to be freed from accumulated dust. This is done by lowering with a chain a weighted disc, *m* Fig. 720, which will scrape the sides clean. Thwaite[1] studied the effect of flue dust upon the thermal efficiency of hot-blast stoves.

14. Calculation[2] of the size of a Cowper Hot-blast Stove.—An iron blast-furnace produces 300 met. tons of pig iron in 24 hr.; 1 ton pig iron requires 1 ton coke; 3.8 cbm. of air burns 1 kg. coke. The problem is to find the size of stove necessary to heat blast from 25 to 800° C.; the temperature at the top of the checkers is 1,200° C. at the start and 1,150° C at the finish; the correspond-



A	*B*	*C*	*D*	*E*	*F*	*G*	*H*
10″	15½″ to 23½″	6″	24″	40″	⅝″ 3½″ cntrs.	16½″	123″

FIGS. 758 and 759.—Gas-escape valve.

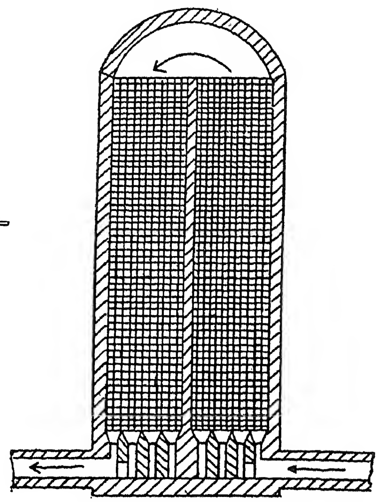


FIG. 760.—Gjers hot-blast equalizer.

ing figures at the bottom are 300 and 150° C.; the checkers are to have vertical flues 25 cm. sq.; the pressure of the blast is to be 1 atmosphere above normal; the velocity of the air is to be 1 m. per sec. on entering and 3 m. per sec. on leaving; the stove is to be on air 1 hr.

(A). Volume of free air required at standard conditions. This is $\frac{300 \times 1000 \times 3.8}{24} = 47{,}500$ cbm. per hr. $= 791.7$ cbm. per min $= 1{,}023.6$ kg.

(B). Heat required to raise air from 25 to 800° C. The mean specific heat $= 0.251$ kg. cal.; hence, $1023 \times 0.251 \times (800 - 25) = 199{,}000$ kg. cal. per min. $= 11{,}940{,}000$ kg. cal. per hr.

[1] *J. I. and St. I.*, 1903, I, 246.

[2] Osann, *Stahl u. Eisen*, 1909, XXIX, 1060, 1107, 1112, 1692.
Aldendorff, *op. cit.*, p. 1690.
Osann, *op. cit.*, 1910, XXX, 2001, 2008.
Aldendorf, *op. cit.*, 2004.
Aldendorf, *op. cit.*, 1910, XXX, 1275.
Osann, *Tonind. Z.*, 1909, XXXIII, 1405.
Částek, *Oest. Zt. Berg Hüttenw.*, 1911, LIX, 1, 20, 35.
Gugen, *Stahl u. Eisen*, 1911, XXXI, 62, 101.
Bulle, *op. cit.*, 1911, XXXI, 1451.

(C). Heating surface required to furnish 198,200 kg. cal per min. The coefficient of internal transfer[1] $=0.000028(2+\sqrt{v})$ C.G.S. $=0.000028(2+\sqrt{200})=0.0004519$ g. cal. per sec. per sq. cm. surface $=0.00419$ kg. cal. per sec. per sq. m. surface $=16.27$ kg. cal. per hr. per sq. m. surface for each deg. C.

The mean temperature of the brick at the top of the stove is $\frac{1200+1150}{2}=1{,}175°$ C.; at the bottom $\frac{300+150}{2}=225°$ C. The mean temperature of the air at the top is 800°; and at the bottom 25° C. The difference in temperature of air and brick at the top of the stove is $1{,}175-800=375°$ C.; at the bottom $225-25=200°$ C.; the mean difference (sufficiently correct for the purpose), $\frac{375+200}{2}=287°$ C. The heating surface required per hour is $\frac{11892000}{16.2\times 287}=$ 2,557 sq. m.

(D). Height of stove. The cross-sectional area of a heating flue is 625 sq. cm. $=0.0625$ sq. m.; the velocity of air at exit is 3 m. per sec. The volume of normal air is 791 cbm. per min. $=\frac{791\times(800+273)\times 1}{2\times 273}=1{,}554$ cbm. at 800° C. and 2 atm. pressure. The number of flues therefore is $\frac{1554}{3\times 0.0625\times 60}=138$. For each meter in height, a flue has $4\times 0.25\times 1=1$ sq. m. heating surface. The height of the stove must be $\frac{2557}{138\times 1}=18.6$ m.

(E). Thickness of brick between flues. There must be enough brickwork in the stove to store 11,940,000 kg. cal. in 1 hr. with a reduction of temperature of not more than 50° from 1,200° C. The mean specific heat of fire-brick is 0.25; the weight of fire-brick required $=\frac{11{,}940{,}000}{0.25\times 50}=955{,}200$ kg. $=\frac{955{,}200}{138}=6{,}922$ kg. around each flue; and this $=\frac{6922}{18.6}=372$ kg. for each meter in height of a flue. The specific gravity of fire-brick $=2.5$; hence $\frac{372}{2.5\times 1000}=0.15$ cbm. brick for each meter in height of a flue, and this $=0.15$ sq. m. brick around each flue $=$ 1,500 sq. cm.; hence the wall is $\frac{\sqrt{1500+625}-25}{2}=10.5$ cm. thick.

(F). Diameter of stove. The horizontal cross-section of the checkerwork $=(0.15+0.0625)138=29.32$ sq. m. The horizontal section of the combustion, chamber = area of flue-openings $=0.0625\times 138=8.62$ sq. m. The total cross-section of the stove excluding the outside walls $=29.32+8.62=37.94$ sq. m., or, $\frac{\pi D^2}{4}=37.94$; $D=6.95$ m.

(G). Summary. This stove must have 138 flues which are 25 cm. sq. and have walls 10.5 cm. thick. The checkerwork must be 18.6 m. high. (There would be added 3 m. at the bottom for the combustion-chamber, and 3 m. at the top for the dome.) The combustion-chamber must be 8.62 sq. m. in cross-section. The diameter of the stove will be 6.95 m.

[1] Richards "Metallurgical Calculations," I, 180.

327. Drying Air by Refrigeration. THE GAYLEY DRY-AIR BLAST.[1]—In §170 and Fig. 262 it has been shown that the amount of moisture air can contain as aqueous vapor increases with the temperature, slowly at first and then very quickly.[2] In order to free from moisture a current of air by cooling, the aqueous vapor must be congealed by lowering the temperature below the freezing-point of water; if this point is not reached, the aqueous vapor remains suspended in the air in the form of a mist or fog and is carried as such into the furnace.

In 1905 Gayley published the first results he attained by the application of his air-refrigerating or dry-blast system to the Isabella iron blast-furnace near Pittsburgh, Pa. The output in pig iron per day was increased by 24.86 per cent.; the consumption of coke per ton of pig iron was decreased by 19.6 per cent.; the blowing engines were run 15.8 per cent. more slowly, and this saving

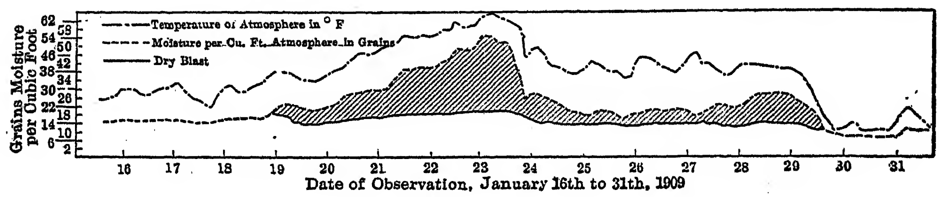


FIG. 761.—Temperature and moisture of air, moisture of Gayley dry-blast.

in power made up for the power required to run the refrigerating plant; further, the temperature of the hot-blast and the percentage of CO_2 in the tunnel-head gases was increased, and the amount of flue dust formed diminished. Beside the saving of cost of production, the dry-blast system has the great advantage over natural blast of complete control of the working of the furnace and thereby of the character of the product. The only uncertain feature in the management of the blast-furnace, the varying amount of moisture[3] in the air has been eliminated from the operation. The importance of this feature is evident when it is recalled that the weight of the air entering an iron blast-furnace at the tuyères is over 50 per cent. of the weight of the solid material charged at the throat. The uniformity of the dry-air blast attained is shown, *e.g.*, in Fig. 761 which gives for 10 days the temperature and moisture of the atmosphere and the moisture of the dry-blast.[4]

[1] *Tr. A. I. M. E.*, 1905, XXXV, 747 (Gayley), 1022–1042 (discussion); 1906, XXXVI, 315 (Gayley), 470 (Johnson), 745–798 (discussion); 1906, XXXVII, 201 (Meissner), 216–237 (discussion); 1907, XXXVIII, 901–912 (discussion); 1908, XXXIX, 695 (Raymond), 705 (Cook); Langdon, 1907; XL, 614; *Iron Age*, 1905, LXXV, 1520 (Editor); 1912, LXXXIX, 52 (Gayley); 826 (Döhl); *Stahl u. Eisen*, 1906, XXVI, 784, 844 (Osann); 1909, XXIX, 1430, 1602 (Drees); *Cass. Mag.*, 1908, XXXIV, 122; *J. Frankl. Inst.*, 1909, CLXVIII, 67 (Report Committee); *Eng. Mag.*, 1909, XXXVII, 186 (Waterhouse); *J. I. and St. I.*, 1909, II, 150 (Moore); *Eng. Min. J.*, 1909, LXXXVIII, 1170 (new patent); *Met. Chem. Eng.*, 1912, X, 471.

[2] Also Carle, *Comp. Air*, 1909, XIV, 5489.

[3] Jones, Uniform moisture in blast, *J. I. and St. I.*, 1909, II, 144.

[4] Other records: *Iron Age*, 1909, LXXXIV, 1233; 1911, LXXXVII, 309.

There have been many adverse criticisms[1] of the use of dry-blast, but accomplished facts have outweighed imperfect theory. The experience gained at various iron blast-furnace plants since 1905 shows[2] that the output of pig iron has been increased 10 per cent., and the fuel consumption diminished 10 per cent. The good effect of dry-blast is probably due to the following four causes:

(1) The removal of the moisture in the blast results in the saving of the heat required for its decomposition. Thermochemical calculations by Le Chatelier[3] and Richards[4] have shown that at the Isabella furnace this accounts for from 3 to 5 per cent. of the heat value of the coke used.

(2) A second cause given by Richards[5] is that, since dry air of low temperature contains more O than ordinary air of average temperature, more O is pumped into the furnace with dry-blast, and this raises the temperature at the tuyère-section and increases the rate of driving.[6]

(3) A third cause is that stated by Johnson[7] and extended by Howe,[8] viz., that the amount of reserve heat necessary between the critical temperature (the one at and above which the essential work of the blast-furnace must be done) and the actual temperature which is developed to insure safety, is smaller with dry-blast than with natural blast, or, in Baker's words,[9] you can sail closer to the wind.

(4) Less work has to be done, hence less fuel is required.[10]

Dry-air blast has been proposed for the bessemer converter[11] to insure, with uniform iron from the mixer, absolute regularity of work.

In carrying out the process at the Isabella furnace, air is drawn by the blowing engine through a refrigerating chamber and then forced through the hot-blast stoves. The chamber, which is lined with compressed 2-in. cork plates, has 60 vertical lines of coils, each with 75 two-inch pipes 20 ft. long, altogether 90,000 linear feet of 2-in. pipe, through which $CaCl_2$-solution of sp. gr. 1.21 is circulated. The solution required, amounting to 40,000 gal., is cooled by an ammonia ice-machine in a tank containing 20 coils of pipe. The air is forced into the chamber by one blower and distributed evenly between the pipes by two additional ones placed on the floor; it deposits water and frost on the lower pipes, frost only on the upper pipes, and is withdrawn at or below

[1] Summary of German Critique (*Stahl u. Eisen*, 1905, XXV, 3, 55, 73, 152, 213, 410) given in *Iron Age*, 1905, LXXV, 464, 1080; *Ann. Mines*, 1904, VI, 551 (Picard, Lodin, Le Chatelier); *Rev. Mét.*, 1905, II, 166, 794 (Boudouard), 589 (Pourcel).

[2] Gayley, *Met. Chem. Eng.*, 1913, XI, 71.

[3] *Rev. Mét.*, 1904, I, 652.

[4] *Tr. A. I. M. E.*, 1906, XXXVI, 745.

[5] *Op. cit.*, 1906, XXXVII, 223.

[6] Moore, *J. I. and St. I.*, 1909, II, 150.

[7] *Tr. A. I. M. E.*, 1906, XXXVI, 470.

[8] *Op. cit.*, 1906, XXXVII, 216.

[9] *Tr. A. I. M. E.*, 1906, XXXVI, 794.

[10] Langdon, *Tr. A. I. M. E.*, 1909, XL, 614; see also Wüst, *Metallurgie*, 1910, VII, 403.

[11] Dudley, *Iron Age*, 1908, LXXXII, 39.

freezing temperature from the top of the chamber by the blowing engines. When the pipes have become covered with frost, the cold $CaCl_2$-solution is shut off from several vertical lines of coil, and hot $CaCl_2$-solution pumped in, which melts the frost in a few minutes, when connection is again made with the cold-solution system. The melted frost collects in a trough in the floor and runs off.

Plants have been erected at Illinois Steel Co., South Chicago,[1] Warwick, Pa.[2] and other places.

The Gayley refrigerating plant is expensive and occupies much space. Its operation is also costly in that it aims to remove all the moisture in a single operation. According to Bruce[3] a considerable saving can be effected by removing the moisture in stages. A two-stage system has been developed which does this economically.[4] The drying or refrigerating chamber consists of a cylindrical steel shell about 100 ft. high and 20 ft. in diam. It has two divisions. In the lower, the air, drawn in by a fan, is made to pass through a series of superimposed screens wetted with water held near its freezing-point. This removes about 2/3 of the moisture. The air rises through moisture eliminators which remove entrained moisture, and enters the second set of superimposed screens which are wetted with cooled $CaCl_2$-solution. The remaining water of the air is condensed; the air passes again through moisture eliminators and enters the fan which delivers it to the pipe leading to the blowing engine.

Another method of cheapening the drying of the air is that suggested by Miles[5] who sprays the air after it has left the blowing engine, first with cold water and then with cooled $CaCl_2$.

328. Drying Air with Calcium Chloride—the Daubiné-Roy Process.—A method differing entirely from the above is that of Daubiné-Roy,[6] who dessicate the air with solid $CaCl_2$. This rests in pieces on grids superimposed in a tower having an annular chamber and a central well. The salt $CaCl_2+8H_2O$ becomes liquid at 15° C; $CaCl_2+4H_2O$ at 40° and $CaCl_2+H_2O$, formed at 175° C., begins to lose H_2O at 235° C. which it reabsorbs with difficulty; that is, the salt acts slowly, is "tardy." The hydrates available lie between $CaCl_2+1H_2O$ and $8H_2O$. The absorption of H_2O evolves heat; this is neutralized by having cooling coils imbedded in the charge in which water circulates.

The air to be dried is drawn downward in the annular chamber for 4 hr. and passes through the salt on the grids into the central well to the fan. By this

[1] *Iron Age*, 1908, LXXXII, 998.

[2] *Tr. A. I. M. E.*, 1908, XXXIX, 917; *Iron Age*, 1908, LXXXI, 53; LXXXII, 906; *Electrochem. Met. Ind.*, 1908, VI, 12.

[3] Bruce, *Proc. Eng. Soc. West. Pa.*, 1912, XXVIII, 277; *Met. Chem. Eng.*, 1912, X, 471.

[4] Gayley, *Iron Age*, 1912, LXXXIX, 52.

[5] *Iron Age*, 1911, LXXXVII, 1354, 1912, LXXXIX, 1023; *Iron Trade Rev.*, 1912, L, 901 (Plant at Stanish, N. Y.).

[6] *Bull. Soc. Ind. Min.*, 1909, XI, 397, 477; *J. I. and St. I.*, 1911, I, 28; *Iron Age*, 1911, LXXXVII, 1270.

Osann, *Stahl u. Eisen*, 1909, XXIX, 1781.

Van Brussell, *Eng. Min. J.*, 1912, XCIII, 595.

time the skins of the pieces of salt will show signs of liquefying. The air is shifted to a second tower, and the salt regenerated in about 2 hr. by heating between 30 and 200° C. with waste dust-free gas or hot air which rises in the well, travels outward through the salt, and leaves by the annular chamber. During the regeneration period the circulation of water in the coils is stopped. The regenerated salt has to be cooled before it can again absorb H_2O; this is accomplished by allowing the water to circulate in the pipes for about 2 hr. before the air is turned on. For the completion of the cycle of operations there are therefore required three towers. The process has been put into operation at the iron works of Differdange, Luxemburg, but has not yet (January, 1913) become standardized.

PURIFICATION OF GASES

329. Smelter Smoke.[1]—In nearly all pyrometallurgical processes some smoke or fume is produced. This may contain solids and vapors besides the leading constituents which are gases. The solid matter, flue dust, consists of fine particles of the original charge which have been carried off by the gas-current and may or may not have been changed chemically in the time they were exposed to the process going on in a furnace. Thus, there may be finely-divided fuel which gives the dust a dark color if present in sufficient quantity; metallic sulphides of the ore-charge may have been converted wholly or in part into sulphates or oxides; oxides may have been sulphatized and so on. The vapors consist of metals such as Pb, Zn, As, Sb, Hg, etc., and their volatile compounds which have been volatilized in the hotter parts of a furnace and not condensed in the cooler; they are more likely to have undergone chemical changes than the dust while passing through and from a furnace. When the gases carrying these vapors are cooled, the vapors form a fog or mist which settles out if given sufficient time. Thus, if the gases from a furnace are passed through flues and chambers on their way to a chimney, there will be collected in the part nearer the furnace the solid particles that the gas-current had carried off, and nearer the chimney the condensed vapors.[2] Fig. 762 shows diagrammatically how the values of Pb and Ag change in the material that is collected in a flue carrying the gases from a silver-lead blast-furnace to its stack. Near the furnace the

[1] Plattner, C. F., "Die Metallurgischen Röstprocesse," Engelhard, Freiberg, 1856, pp. 322–377.
Hering, C. A., "Die Verdichtung des Hüttenrauches," Cotta, Stuttgart, 1888 (bibliography, pp. 61–63).
Iles, *School. Min. Quart.*, 1895–96, XVII, 97.
Hamor, Bibliography, *Min. Ind.*, 1908, XVII, 901.
Baskerville, *Eng. Min. J.*, 1909, LXXXVII, 884 (legal aspect).
[2] Shelby, *Eng. Min. J.*, 1908, LXXXV, 204.
Ebaugh,, *J. Ind. and Eng. Chem*, 1909, I, 686; *Eng. Min. J.*, 1909, LXXXVIII, 1020.
Lee, *op. cit.*, 1910, XC, 505.
Haas, *op. cit.*, 1910, XC, 814.
Moore, *ibid*, 104.
Wright, *ibid.*, 111.

dust is rich in Ag and low in Pb, the values corresponding closely to those of the ore-charge; near the stack, on the contrary, only lead-fume carrying little silver has been collected.

The gases consist mainly of N, CO_2 and CO, but in many instances they contain sulphurous gas, Cl and Cl-compounds, which are injurious to animal and to vegetable life. These obnoxious gases are frequently discharged into the atmosphere through high chimneys whereby they become sufficiently diluted before they descend to the earth so as to be harmless; under certain conditions, especially in populous or agricultural regions, they have to be neutralized and rendered harmless; in some cases they can be converted into useful products.

The treatment of smelter smoke will vary with the result one is trying to attain. The aim may be: (1) To collect as flue dust the solid particles and the

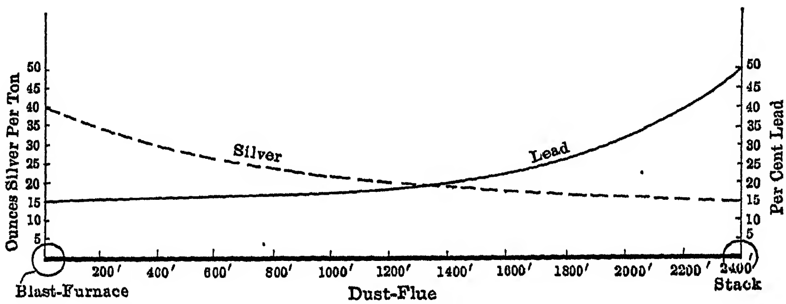


FIG. 762.—Distribution of lead and silver in dust-flue.

condensed vapors in order to recover their values as is the case with most non-ferrous furnaces;[1] (2) to free gases from dust in order that they may be used for industrial purposes as, for instance, the tunnel-head gas of the iron blast-furnace which is used to drive gas-engines; (3) to render harmless or convert into useful products the obnoxious constituents produced in many non-ferrous processes.

In all three cases the gases will be conducted through flues or pipes or chambers and perhaps some auxiliary apparatus to the chimney which discharges the waste product into the open air. The size of the conduit will be governed by the volume of gas that is to pass through it per sec. Assume that it has been found, in calculating the theoretical balance-sheet of materials, that the gas produced from a 1,000-kg. charge of a lead blast-furnace weighs 1,933.8 kg.; that its volume, as figured from the analysis of the flue-gas, is 1,353 cbm. at 0° C., and that the gas passes off at 200° C. The volume a gas occupies is proportional to its absolute temperature (−273° C.), or $V_1 = V_0 \times \frac{T}{t} = 1{,}353 \times \frac{200+273}{273} = 2{,}307$ cbm. Assuming that in 24 hr. there are fed 150 charges,

[1] Mathewson, *Eng. Min. J.*, 1909, LXXXVII, 562.
Schott, *Stahl u. Eisen*, 1910, XXX, 192, 332, 367, 803.
Anon, *Engineer*, 1910, CX, 470; *Rev. Mét. Extr.*, 1911, VIII, 334.

this gives 346,050 cbm. or $\frac{346050}{24\times60\times60}=4.05$ cbm. = 143 cu. ft. per sec. A fair speed of travel of gas for settling out solid particles has been given as 7 ft. per sec.; Iles[1] gives for flues in roasting-furnaces 200—400 ft. per min., or 3.33—6.67 ft. per sec.; Ingalls[2] makes 4 ft. per sec. the basis of his calculations. Taking 6 ft. per sec. as the speed of the above gas, $\frac{143}{6}=22$ sq. ft. will give the necessary cross-sectional area of the conduit. The travel of gases through furnaces has been studied by Roitzheim.[3]

The amount of dust carried off by flue gases varies greatly, from 1 to 3 per cent. in hand reverberatory roasting-furnaces to over 20 per cent. with some fine-ore shaft roasting furnaces; from about 1 per cent. with coarse-ore reverberatory smelting furnaces to 10 to 15 per cent. with reverberatory furnaces treating roasted concentrates; from 3 to 5 per cent. with coarse-ore blast-furnaces to over 20 per cent. with blast-furnaces treating finely divided iron ore.

For the determination of the amount of solid particles in gases various methods and apparatus have been devised. Some that have become prominent are those of Martins,[4] Hubendick,[5] Gemünd,[6] Messler,[7] Wynne,[8] Johannsen.[9] The different methods of condensing may be grouped under three heads, as *dry*, *electric* and *wet;* the last may or may not be connected with the recovering or rendering harmless obnoxious gases.

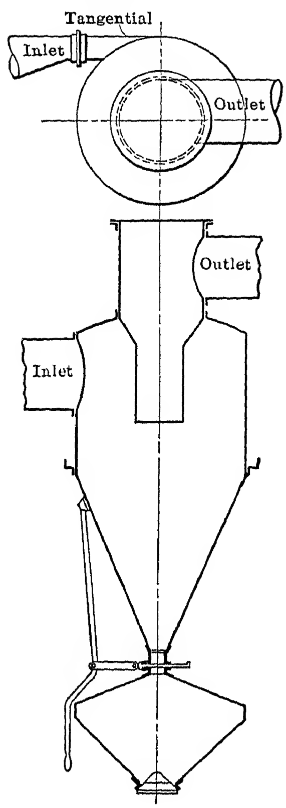
330. **Dry Methods of Condensation.**[10]—This is the most common plan, as it is effective, cheap in the simple forms of installation, and furnishes the flue dust in a form that is readily compacted for further treatment (§279). The separation of solids is effected by the cooling of the gas-current; by retardation of its velocity involving enlargement of flue, change of direction of current, or increase of surface friction; by centrifugal motion; by filtration and by freezing. Several of these procedures are usually combined in a single system.

331. **Cooling the Gas-current.**—With the reverberatory furnace and the blast-furnace, especially when run with a hot top, reduction of temperature of flue gases is an essential preparatory step to the saving of values, and preeminently so if the ore-charge contains any volatile metals. Cooling a gas reduces its volume and condenses vapors, the suspended particles come closer together

[1] "Lead Smelting," Wiley, New York, 1902, pp. 155, 176.
[2] "Metallurgy of Zinc and Cadmium," Eng. Min. J. New York, 1903, p. 63.
[3] *Stahl u. Eisen*, 1912, XXXII, 969, 1098, critique by Esslich, *ibid*, 2090.
[4] *Op. cit.*, 1903, XXVIII, 735.
[5] *Oest. Zt. Berg Hüttenw.*, 1905, LIII, 200.
[6] *Braunkohle*, 1907, VI, 30.
[7] *Iron Age*, 1908, LXXXI, 1687.
[8] *Eng. Min. J.*, 1909, LXXXVIII, 603.
[9] *Stahl u. Eisen*, 1912, XXXII, 16.
[10] Iles, *Eng. Min. J.*, 1886, XLI, 74, 93, 148; *Sch. Min. Quart.*, 1895–96, XVII, 97.
Bauer, *Freiberg. Jahrb.*, 1894, p. 39.
Austin, *Min. Sc. Press*, 1907, XCIV, 668.
Schott, *Stahl u. Eisen*, 1910, XXX, 192, 322, 367.

and unite into flaky masses, as the attraction is inversely proportional to the square of their distance asunder; they then settle readily.

Let the temperature of gas, V_2 at 1,100° C., as it escapes from a reverberatory furnace be reduced to 100°, and assume its volume V_1 at 100° to be $=1$; then $V_2 = V_1 [1+0.00366(1{,}100-100)] = 4.665\ V_1$. Now the distance of two particles A_2 and A_1 in the hot and the warm gas is as $\sqrt[3]{4.665} : \sqrt[3]{1} = 1.6709 : 1$; hence the attractions of A_2 and A_1 are as $1^2 : 1.6709^2 = 1 : 2.7919$.



FIGS. 763 and 764.—Roberts' centrifugal dust-catcher.

Aitken[1] has shown experimentally that a hot surface repels particles of dust, especially if it is moist, while a cold surface attracts them. Similar phenomena have been observed by Russell.[2]

The usual way of cooling is by the use of air; when insufficient it may be supplemented by cooling the flue, through which the gases travel, externally with water or by having cooling-water circulate in pipes placed in the flue. The flue itself is of metal, of brick, or of concrete which may be reinforced.

The down comer of the iron blast-furnace is a sheet-iron pipe, of large diameter to reduce the velocity of the gases to about 10 ft. per sec. It is placed at an angle of at least 40 to 45° to prevent dust (angle of repose about 36°) from settling out. The dust is collected in the dust-catcher whence it is removed at intervals from the bottom by a balanced trap-door. The dropping out of dust is assisted by a sudden change in direction of the gas-current; often provision is made for further removal of dust by having hoppers[3] on the lower side of the sheet-iron flue leading from the dust-catcher to boilers and pre-heating stoves.

Roberts[4] constructed the centrifugal dust-catcher shown in Figs. 763 and 764 for the iron blast-furnaces at Lebanon, Pa. The gas enters the apparatus tangentially, is given a rotary motion by its own pressure, and leaves through the outlet which is placed centrally at the top of the collector. The dust that has been dropped is collected in a conical pocket which has a slide-valve and a hopper provided with a bell. When the dust in the hopper is to be discharged, the slide-valve is closed and thus the entrance of air prevented. The collector

[1] *Proc. Roy. Soc. Edinburgh*, Jan. 21, 1884; "Nature," 1883–84, XXIX, 322. *Wagner, Jahresber.*, 1884, XXX, 1307.

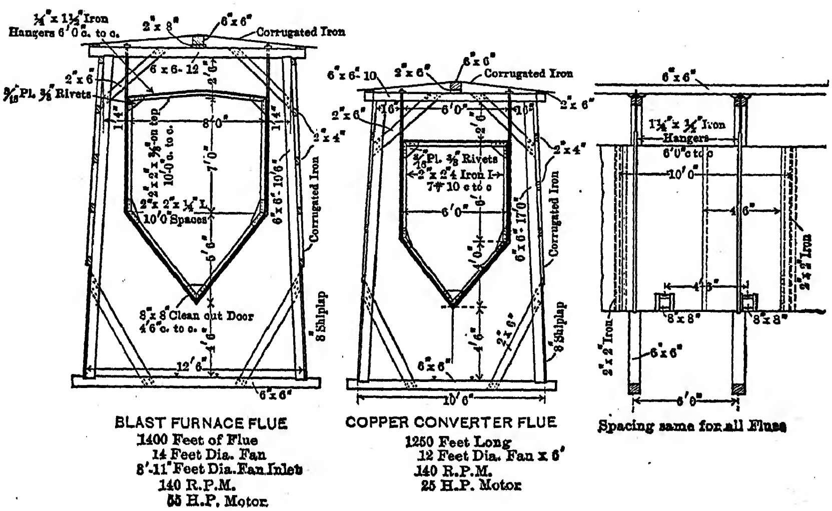
[2] *Phil. Trans. Roy. Soc.*, 1903, CCI, A, p. 185, discussion by Aitken, p. 551.

[3] Lürmann, *Stahl u. Eisen*, 1896, XVI, 955; 1901, XXI, 433, 489. Schmidthammer, *op. cit.*, 1897, XVII, 55.

[4] *Iron Age*, 1707, LXXIX, 1414.

is placed over a railroad track to allow discharging directly into a car. Gases issuing from the dust-catcher contain[1] from 0.66 to 2.64 grains dust per cu. ft. Thwaite[2] shows how the deposition of fine dust by gravity is limited. Thus, to settle dust from gas containing 0.44 grains dust per cu. ft. a flue 337.3 ft. in length is required; if the gas contains 0.33 grains, deposition practically ceases.

In lead- and copper-smelting works a suspended or supported rectangular or balloon-shaped sheet-iron flue with V-shaped bottom provided with discharge-doors, or a rectangular flue the lower side of which consists of V-shaped hoppers,[3] serves to carry the gases from the furnaces to the dust-chambers.



FIGS. 765 to 767.—Suspended rectangular sheet-iron flues, Omaha, Neb.

Figs. 765 to 767[4] represent the rectangular sheet-iron flues for the blast-furnaces and converters of the Omaha plant of the American Smelting & Refining Co.; while Figs. 768 to 772 show the complete drawings of the graceful form of the balloon-shaped supported sheet-iron blast-furnace flues of the lead plant of the International Smelting & Refining Co., Tooele, Utah. The drawings are self-explanatory.

A third type is one which resembles the pipe-stove (Figs. 707 to 714) for pre-heating air. This apparatus is common with lead ore-hearths,[5] the furnaces used in roasting and smelting antimony ores, and with iron blast-furnaces treating ores that are rich in zinc. The apparatus erected by the New Jersey Zinc Co. for its new spiegeleisen blast-furnace at Palmerton,

[1] Sahlin, *Cass. Mag.*, 1905, XXVIII, 442.

[2] *J. I. and St. I.*, 1903, I, 246.

[3] Egleston, *Tr. A. I. M. E.*, 1883, XI, 410, plate II.

[4] Eilers, *Proc.* VIII *Internat. Congr. Appl. Chem.*, 1912, III, 76; *Tr. A. I. M. E.*, 1912, XLIV, 708.

[5] Dewey, *op. cit.*, 1889–90, XVIII, 673, 696 700.

Pa., is given in Figs. 773 to 775. The blast-furnace (64 ft. high, 14-ft. bosh, 9-ft. crucible, seven 5-in. tuyères), using a mixture of 2/3 coke and 1/3 anthracite, smelts in 24 hr. with 254 cu. ft. air per min. at a pressure of 9 to 14 lb. per sq. in. 135 to 140 tons of residuum, and produces with the spiegeleisen considerable amounts of flue dust rich in zinc, which is settled in iron condensers, from which the purified gases pass off to be burnt in pipe-stoves similar to those shown in Figs. 712 to 714.

The tunnel-head gases enter the condensing apparatus through the inclined down-comer *A*, which has to be beaten daily with wooden hammers to prevent

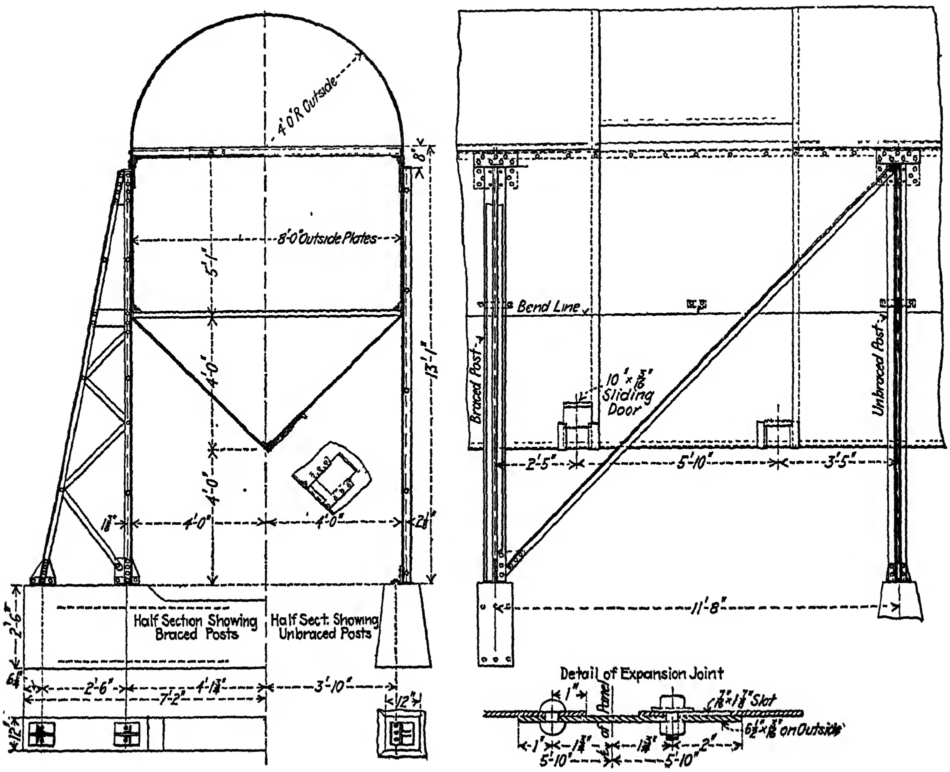


FIG. 768 to 772.—Supported baloon-shaped flue, Tooele, Utah.

the dust from accumulating. The down-comer ends in the dust-chamber, *B;* this is closed at the bottom by balanced doors, *b*, and has in addition cleaning doors, *b'*, at front and back. On the sides are the two gas-outlet pipes, *C* and *C'*, with dampers, *c* and *c'*, to permit cutting out one-half of the condensing plant. The outlet-pipes lead through four branch-pipes, D^1, D^2, D^3, and D^4 to the two systems of pipe-condensers, *E* and *E'*, which are supported 8 ft. from the ground by a steel structure. Each branch-pipe conducts its portion of gas through seven pipe-condensers, *F*, connected in series. The purified gas from

the four rows of condensers is collected in the main *G*, which leads to the stoves. The arrows in the figures show the courses of the gas through the densers. A condenser consists of six vertical cast-iron tubes, *t*, 12 ft. long 15 in. in diam., which are connected top and bottom by cylindrical colla 7 ft. in diam.; the upper and lower collars are alternately connected for the sage of the gas; the lower collars have cleaning doors, *h*, and are provided hopper-bottoms, *I*, for collecting dust. The dust is discharged throu balanced door, *i*. The frequent reversal of the direction of the gas-curre this class of apparatus greatly assists the deposition of dust, but it also c such resistance to the travel of the gas that artificial draft has to be prov

In 1877 Hagen[1] introduced at the Freiberg, Saxony, smelting and refi works flues of sheet lead externally cooled by water for cooling sulphurous carrying As_2O_3. In 1902 they were replaced by brick flues for hot, and by lead flues for cool, gases.

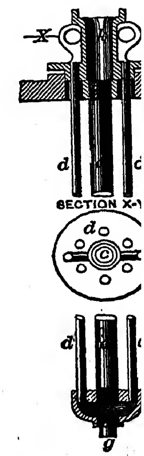


FIG. 776.—B of cooling p

At the smelting and refining works of Tarnowitz, Silesia,[2] the gases are cooled by pipes, with circulating water, suspended in a brick cooling tower 16 ft. 6 in. by 6 ft. 6 in. and 32 ft. 10 in. high. This is closed at the top by an iron plate with 180 holes through which pass bundles of these cooling pipes, Fig. 776. Each bundle consists of a central water-inlet pipe, *c* (16 ft. 6 in. long by 1 5/8 in. in diam.) surrounded by six smaller pipes 25/32 in. in diam. The lower ends are screwed into an iron cap with cleaning-hole, *g*; the upper ends pass through a similar cap, *k*; between the collar, *f*, of the latter and the cover, *e*, of the tower is placed a rubber gasket to insure a tight joint. Dust adhering to the pipes is removed periodically by a jet of steam introduced through a movable central pipe provided with nipples at certain intervals. The use of cooling coils by Schlösser and Ernst[3] accomplishes a similar result.

Apparatus resembling vertical tubular boilers are used for cooling from by-product coke-ovens to condense vapors and recover tar and amm water. They are unsuited for dust-laden gases, as the passages, usually s would be choked in a very short time.

Most of the flues and dust-chambers[4] of non-ferrous plants are built of or hollow brick, cement concrete, and reinforced concrete (Monier syst Glazed earthenware pipes are serviceable for small volumes of corrosive g

Brick forms the common building material. In former times the were made very thick, and flues as well as chambers were placed undergro

[1] *Freiberg Jahrb.*, 1879, p. 151.
Hofman, "Lead," 1898, p. 386.
[2] Saeger, *Zt. Berg. Hütten. Sal. Wesen i. Pr.*, 1893, XLV, 280.
[3] *Berg Hüttenm. Z.*, 1885, XLIV, 464; 1887, XLVI, 134.
[4] Lee, *Bull. Tech. Eng. Soc. Colo. School Mines*, 1909, IV, 197; *Min. Sc.*, 1909, LX, 556; *Ind.*, 1909, XVIII, 497.

At present the side walls are usually 13 in. thick and the arch 9 in.; the flue or chamber is placed on the ground or is elevated in order to facilitate the discharge of accumulated dust. A dust-flue,[1] as seen frequently, is shown in Fig. 777. The gases enter through the opening, *g*, in the arch; on the side near the floor are small openings, *b*, 9 ft, 4 in. apart and diagonally opposite one another, through which settled dust is raked out periodically. This discharge of dust is made more easy by a flue having a sloping bottom the lower side of which is high enough to permit running a car beneath the discharge and taking the dust into it.

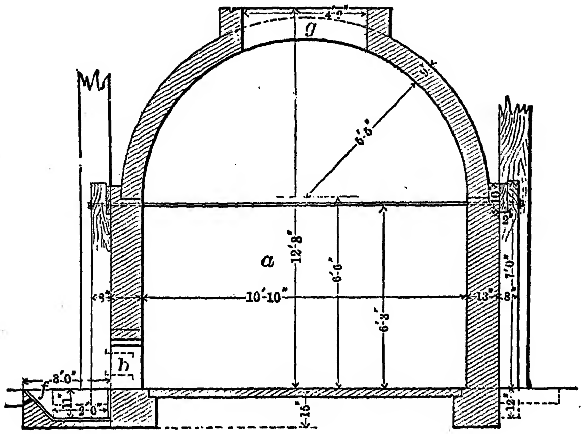


FIG. 777.—Brick dust-flue.

Another form[2] with flatter roof is the one shown in cross-section in Fig. 778. The footing, of concrete, is from 15 to 24 in. deep and from 10 to 18 in. wide. The thrust of the arch is calculated according to the formula $T=1.500\ L^2 : r$, in which T equals the horizontal component of the arch thrust in lb. per linear ft. of arch, w=weight of arch and superimposed load in lb. per sq. ft.; L= length of span in ft.; r=rise of arch in in. The buckstays, skewbacks and tie-rods are so calculated as not to be strained above the elastic limit, should one set fail. In calculating the footing, the bearing power of the soil is taken as 2 tons per sq. ft. The compressive strength of ordinary brick is taken as 200 lb. per sq. in., stresses due to wind pressure are neglected as long as the height of the flue does not exceed 14 ft.

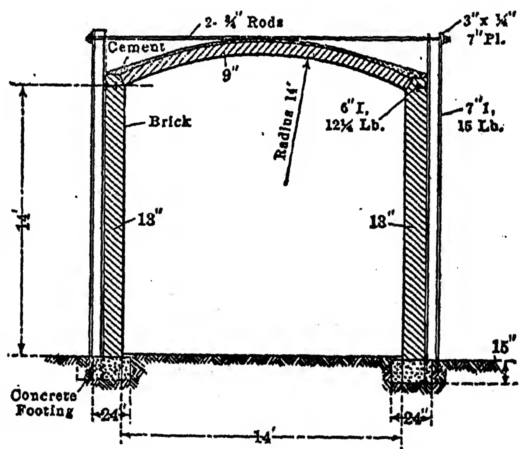


FIG. 778.—Brick dust-flue.

In several modern plants producing much dust, the flue or chamber is elevated, being supported by a steel structure, and the bottom made up of a series of hoppers whence the dust is discharged into cars run beneath them. A typical example of this form is furnished by the chambers of copper-smelting

[1] Hofman, "Lead," 1899, p. 310, Fig. 395.
[2] Lee-Hofman, *Min. Ind.*, 1909, XVIII, 499.

plant at Anaconda, Mont.[1] At this plant the chambers of the roasting, blast-furnace-smelting, reverberatory-smelting, and converting departments are 9 ft. above the ground and have a series of discharging hoppers on the bottoms. These chambers are all built of brick in steel-framings, the roofs are either smaller or larger brick arches supported by I-beams or they are built of concrete and expanded metal The flues from the several chambers (488+1,653+842

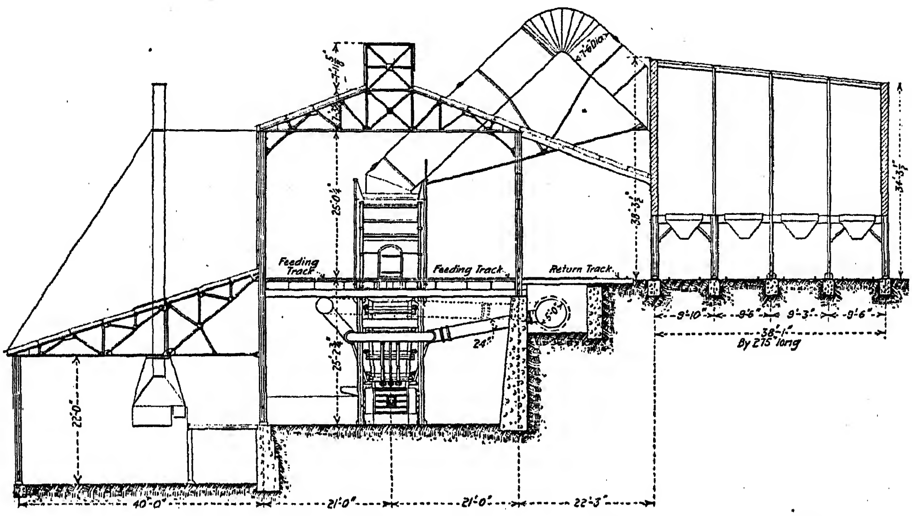


FIG. 779.—Vertical cross-section through blast-furnace-building, Anaconda, Montana.

+703=3,686 ft. long) leading into a main flue are placed partly underground, as is the case with the main flue; the latter is 60 ft. wide by 1,233 ft. long and is duplicated for a distance of 995 ft. to the foot of the stack.

Fig. 779 represents a general cross-section of the blast-furnace building of the Anaconda copper-smelting works. The building has three divisions; in the center is the blast-furnace with feed-floor, bustle-pipe and settler; to the left are the depressed track for the matte-car and the hood for carrying off gases arising from the liquid matte; to the right is the elevated dust-chamber 38 ft. 1 in. by 275 ft. with a slanting roof rising from 34 ft. 3 1/2 in. up to 38 ft. 3 1/2 in. high. The gases pass from the blast-furnace through an inverted siphon into the chamber. The side-walls are brick set in steel framing; the roof is of reinforced concrete; the sulphurous gas had such a corrosive effect that this material had to be replaced by brick which were laid between steel rails placed

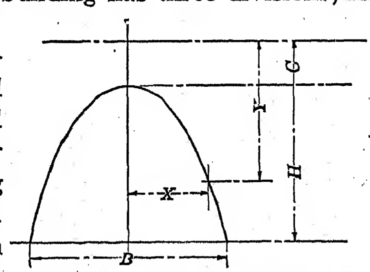


FIG. 780.—Catenary curve.

[1] Correspondent, *Eng. Min. J.*, 1903, LXXVI, 962.
McDougall, *Trans. Canad. Soc. Civ. Eng.*, 1904, p. 357; *Can. Min. Rev.*, 1905, XXIV, 26.
Austin, *Tr. A. I. M. E.*, 1906, XXXVII, 478; *Min. Sc. Press*, 1907, XCV, 649.

9 in. apart. The floor is made up of four rows of hoppers running lengthwise. The roof and floor are supported by three rows of 6-in. columns placed 8 ft. apart. When first erected, the chamber had a longitudinal partition wall in order to lengthen the path of the gases. It was soon found that practicallly all the dust settled in the inlet division, and the partition was then removed. The chamber collects, in 24 hr., 60 to 70 tons of coarse dust which is smelted in the reverberatory furnaces treating roasted concentrates.

The ordinary form of flue with vertical side-walls has to be bound with skewbacks and tie-rods. This is not necessary if the cross-section has the form of an inverted catenary curve,[1] as shown in Fig. 780. Many dust-flues in recent years have been built in this form. Table 249 serves to lay out catenaries in which $H : B = 0.75$ approximately. This fixed ratio requires a value for C which is different for each curve; it has been calculated according to the equation of the catenary, $y = \frac{C}{2}\left(C^{\frac{x}{c}} + e^{-\frac{x}{c}}\right)$ in which $e = 2.7182$, the naperian base.

TABLE 249.—CATENARY FLUES

B	Values for Y-C in feet, with X=										C
	½	1	1½	2	2½	3	3½	4	4½	5	
8	0.064	0.267	0 620	1.148	1 893	2.903	4.244	6 00	....	....	1.90
10	0.057	0.215	0.491	0.901	1 462	2.195	3 129	4.314	5.80	7 659	2 35
12	0.039	0.174	0.396	0 729	1 170	1.728	2.443	3.314	4.373	5.653	2.84
16	0 032	0.129	0.292	0.534	0.851	1.242	1.727	2.297	2.994	3.786	3.80
20	0.026	0.100	0 242	0.429	0.657	0.967	1.334	1.761	2.258	2.839	4.80
B	5½	6	6½	7	7½	8	8½	9	9½	10	C
12	7.205	9 061	....	....	....	....	....	....	....	....	2.84
16	4.721	5.806	7.049	8 489	10.125	12.011	....	....	....	....	3.80
20	3.499	4.262	5.107	6.076	7.149	8.373	9.72	11.215	12.907	14.76	4.80

Thus, with $B=8$ ft.; supposing $x=4$, the end of the curve, the value of $Y-C$ ($=H$) will be 6.

The walls of the flue are usually 9 in. thick.

Fig. 781 is a cross-section of a catenary flue. The most economical cross-section is obtained by making the height 75 per cent. of the base and the cross-sectional area 55 per cent. of the square of the base. A flue of 9-in. brick, 16 ft. high and 22 ft. wide at the base, can stand a wind-pressure of 10 to 12 lb. per sq. ft. of exposed area. In order to guard against accidents due to distortion caused by settlement, excessive external load or corrosion on the inside,

[1] Stewart, *Eng. Min. J.*, 1909, LXXXVIII, 257.
[2] Lee-Hofman, *Min. Ind.*, 1909, XVIII, 498.

some flues recently built have been strengthened by ribs of reinforced concrete, spaced from 6 to 10 ft. apart. In a flue, expansion joints, 2 in. wide, are provided every 100 ft. They are covered with one thickness of brick laid dry.

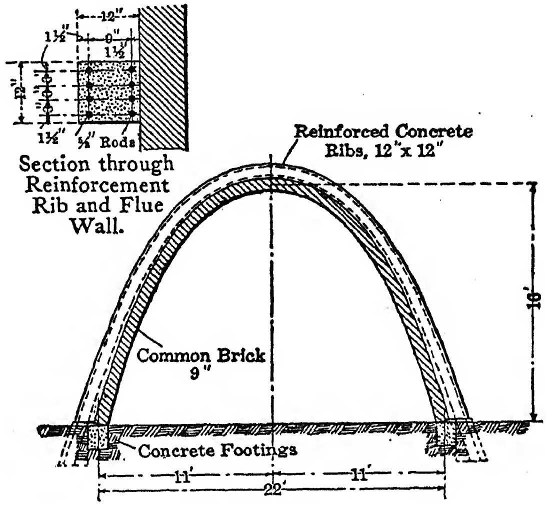
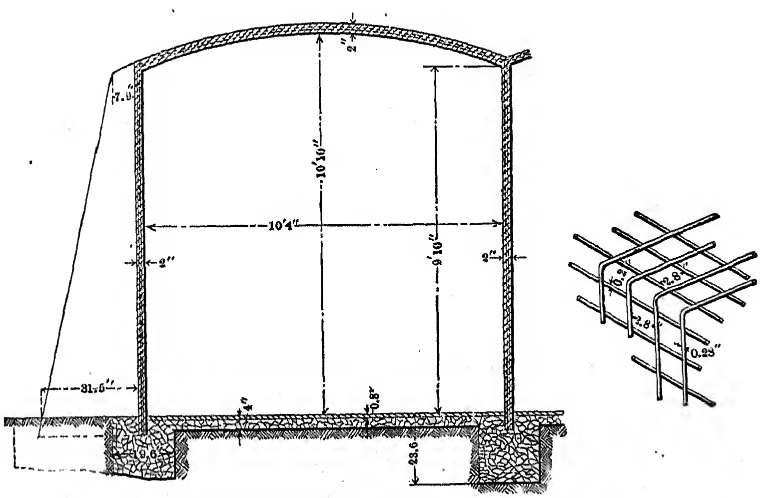


FIG. 781.—Catenary flue.

An acid-proof mortar for laying a 4-in. course of vitrified brick consists of barite mixed with a 10-per cent. solution of commercial (50 per cent. pure) water-glass.



FIGS. 782 and 783.—Monier dust-flue, Freiberg, Saxony.

In the catenary flue of Murray, Utah,[1] the bricks are laid in two courses of stringers with headers every sixth row and 3-in. expansion-joints every 20 ft.

[1] *Min. Ind.*, 1907, XVI, 665.

Ordinary brick has[1] been replaced in some instances by cement concrete which has proved satisfactory as long as the gases are not too sulphurous.

Reinforced concrete has been a success in some instances and a failure in others. In its original form as Monier material it was introduced in the Harz Mountains[2] in 1892, in Silesia[3] in 1893, in Saxony[4] in 1894. The Monier flue of Freiberg, Saxony, is shown in Figs. 782 and 783. It consists of a wire lattice with cement concrete, 1 in. thick, on either side. In building the flue, the wire lattice (2 3/4-in. mesh, 3/16-in. horizontal and 1/4-in. vertical wires with

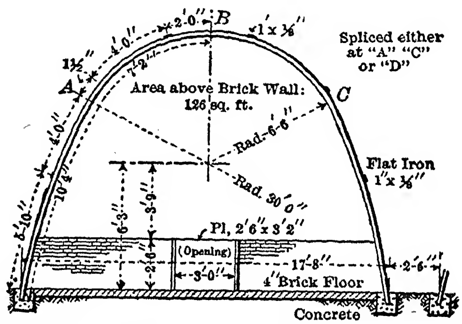


FIG. 784.—Monier dust-flue, Leadville, Colo.

3/8-in. wires at short intervals) is stretched over wooden centers, 70 to 80 ft. long, and kept 1 in. away from them by small blocks; concrete, consisting of two parts cement and one part sand (one-half 0.4–0.6 in., one-half 30-mesh and finer) or one cement and two sand is pressed through the wire lattice until the space underneath is filled and then a layer 1 in. thick tamped down on top, making the wall 2 in. thick. In a few days the concrete has hardened so that the center can be taken down; butresses of similar concrete are placed at intervals of 6 to 7 ft. When a wall is finished, the bottom is tamped 3 1/8 in. thick

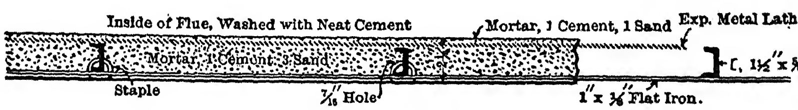


FIG. 785.—Monier dust-flue, Leadville, Colo.

with concrete; the inside and outside of the side-wall is coated with a thin layer of cement mortar while the bottom receives a layer 25/32 in. thick; lastly the inside is coated with acid-proof paint and the outside with tar.

At the lead-smelting works of the Arkansas Valley Smelting Co., Leadville, Colo., Messiter[5] constructed a catenary flue of concrete reinforced by expanded metal, Figs. 784 to 787, to conduct the gases from the roasting furnaces to the

[1] *Stahl u. Eisen*, 1893, XII, 867; *Berg. Hüttenm. Z.*, 1893, LII, 22.
[2] *Stahl u. Eisen*, 1893, XII, 867; *Berg. Hüttenm. Z.*, 1893, LII, 22.
[3] *Zt. Berg. Hütten Salinen Wesen i. P.*, 1893, XLI, 267; *Berg. Hüttenm. Z.*, 1894, LIII, 299.
[4] *Freiberger Jahrb.*, 1894, p. 34; *Eng. Min. J.*, 1895, LIX, 342.
[5] *Eng. News.*, 1899, XLII, 356, through *Min. Ind.*, 1900, IX, 455.

stack. Channel-irons, bent to the form of a catenary, are set 15 in. apart in concrete base-walls, and connected by bars of flat iron fastened by clinched staples. To the inside of the channels and bars is wired a lathing of expanded metal, and the whole embedded in cement concrete making a wall 2 1/4 in. thick. The inside is coated with cement mortar and then a wash of neat cement is applied. Fig. 787 represents one of the openings in the crown of the arch.

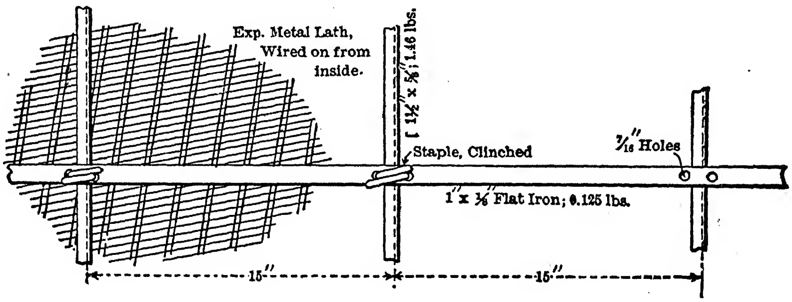


FIG. 786.—Monier dust-flue, Leadville, Colo.

Other forms are given by Welsh,[1] Edwards,[2] and Lee[3]; the different shapes of expanded metal are described in *Iron Age*, 1906, LVII, 13.

The evidence furnished regarding the effect of heat, moisture and acid upon concrete is somewhat conflicting. Since cement begins to give up its water at 500° C., heat alone will have no harmful effect. As long as a flue is well built and of the right mixture, there is no reason why rain-water should percolate through the minute cracks caused by expansion and contraction as long as the flue is well coated with an elastic and impervious material. Flues are

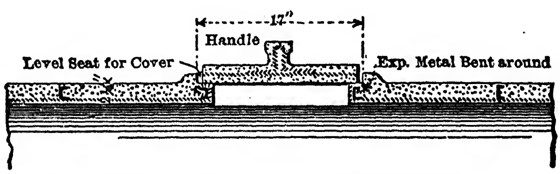


FIG. 787.—Monier dust-flue, Leadville, Colo.

usually covered with a rough shed. There appears to have been some difficulty in finding an acid-resisting coating for the interior. Thus at Anaconda[4] reinforced concrete was so badly corroded that it had to be replaced by brick, while at the Butte Reduction Works[5] it appears to have stood satisfactorily. The general opinion is that concrete flues are short-lived, hence, in order to

[1] *Eng. Min. J.*, 1904, LXVIII, 348.

[2] *Tr. A. I. M. E.*, 1905, XXXV, 75, 965.

[3] *Bull. Tech. Eng. Soc., Colo. School Mines*, 1909, IV, 197; *Mining Sc.*, 1909, LX, 556; *Min. Ind.*, 1909, XVIII, 498.

[4] McDougall, *Can. Soc. Civ. Eng.*, 1904, p. 357; *Can. Min. Rev.*, 1905, XXIV, 260.

[5] Wethey, *Eng. Min. J.*, 1909, LXXXVIII, 1153.

avoid accidents, they are usually built of brick and of vitrified brick if the gases are sulphurous. The catenary form is the more generally used.[1]

The cooling effects of different building materials is shown by the following data from the Freiberg smelting and refining works.[2] The temperature of the gases is reduced 1° C. for 100° C. difference between outside and inside by passing through 9 ft. 10 in. of lead flue; 14 ft. 9 1/8 in. of Monier flue with two sides exposed to the air; 19 ft. 8 1/4 in. of brick flue with two sides exposed; 27 ft. 2 3/4 in. of Monier flue with one side, and 39 ft. 4 3/8 in. of brick flue with one side exposed. Lee[3] states that the loss in temperature in brick flues with sides 13 in. thick and roof 9 in. thick, is in 100 ft. of flue 0.154° C. for every degree C. difference between the temperature of gases inside and atmosphere outside. According to C. L. Norton[4] there is a loss of 2.40 B.t.u. per sq. ft. per hr. for 1° F. difference between an iron pipe and its surroundings.

Friedrich[5] has devised a new plan for cooling the sulphurous gases from roasting furnaces and for collecting the values they contain by combining the regenerative and recuperative systems ordinarily used for pre-heating air. In the regenerative part he cools the gases leaving the roasting furnace at 300° C. down to 200° C. This limiting temperature is chosen, as in concentrating dilute sulphuric acid a loss in acid by evaporation occurs only when the acid has reached a strength of 50 to 60° Bé, and the boiling-point of such acid lies at about 200° C. The regenerator is filled with rhomboid clay cross-bars or Freudenberg plates (see below), or clay brick spaced in such a way as to give free passage to the gases, similar to brick hot-blast stoves. It is operated intermittently in a manner resembling that of the Siemens regenerator, thus utilizing the heat stored in the filling-material. In the recuperator, filled with acid-proof brick in a manner similar to that of the hot chamber, the gases are cooled from 200° C. to atmospheric temperature; the air passes *seriatim* through a number of chambers, resembling in arrangement a continuous brick kiln, and is discharged into the open; one chamber is always cut out of circuit to be cleaned by spraying. Drawings of a plant intended to serve three reverberatory roasting furnaces with three hot and one cooling chamber are given in the original article.

332. Retarding the Gas-current.—The three leading methods for diminishing the velocity of the gas-current are enlarging the volume of the flue, changing the direction of the current, and increasing the contact-surface. Hering[6] has shown that the velocity of an air-current in a flue is greatest along the center-line at 0.7 of the height; it diminishes little upward but much downward, so much so that there may be even a small counter-current on the bottom. Most of the dust collects on the floor, while some adheres to the roof.

[1] Murray, Utah, *Min. Ind.*, 1907, XVI, 665; Rio Grande, N. M., *Mines and Minerals*, 1908, XXVIII, 436.

[2] Bauer, *Freiberg. Jahrb.*, 1894, p. 55.

[3] *Loc. cit.*

[4] Private Communication.

[5] *Metallurgie*, 1906, III, 747, 774, 805.

[6] *Op. cit.*, p. 5.

Enlarging a flue greatly assists the settling of dust. The result is not, however, as effective as might be supposed, as the gas current[1] does not fill the enlarged flue. It passes through it with but a slightly diminished velocity and makes eddies in the remaining space, hence it deposits only a comparatively small portion of the suspended dust. Tapering the approach to the dust chamber improves matters somewhat. In all cases a flue or chamber ought to have transverse walls rising, say 1 or 2 ft., from the floor to hold in place the dust which has settled out.

The experimental results of Lee[2] at the Copper Queen smelter, Douglas, Ariz., indicate the contrary. He used an enlarged chamber and found that as long as the velocity of the gases did not exceed 150 ft., per min. a chamber 125 ft. long was sufficient to settle all the recoverable dust.

Changing the direction of the gas-current is effective with coarse particles of dust, if the change is large enough, as *e.g.*, 180 deg. in the down comer of an iron blast-furnace, and if at the same time the velocity is decreased. Fine particles require additional obstacles to force them out of the current.[3] The practice of many non-ferrous smelting-plants of directing the gas-current in a zig-zag course[4] in dust chambers over one partition and under the next has become about obsolete, because the diminution of the velocity of the current is too small.

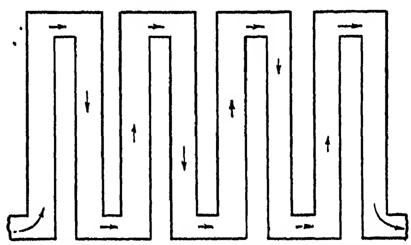


Fig. 788.—Zig-zag flue, plan.

Increase of surface remains as the last expedient. The larger the surface the greater the friction between the stationary wall and the moving gas-current, and hence, the less velocity. In many cases this has been attained by erecting the main stack some distance from the works and connecting it with a long flue. Anaconda (p. 839) with its 3,686 ft. of flues is a striking example. With lack of room for a long flue, a zig-zag flue, as shown in Fig. 788, is in place, but here the bottom and the sides facing one another, will be imperfectly cooled unless the whole system is elevated so as to allow the air to have free access.

Freudenberg[5] increased the surface of the flue by suspending in it sheet-iron plates parallel with the air-current, and found that the amount of flue dust settled was proportional to the surface-area with which the current had come in contact. A cross-section of a flue with suspended plates is shown in Fig. 789. Sheet-iron plates *B*, of 1/32 in. iron, are suspended by riveted iron hooks,

[1] Messiter, *Min. Sc. Press*, 1908, XCVII, 26.

[2] *Eng. Min. J.*, 1910, XC, 504.

[3] Kroupa, *Oest. Zt. Berg. Hüttenw.*, 1905, LIII, 347.

[4] Examples, see Hering, *op. cit.*, Plates 6 and 7.

[5] Die auf der Bleihütte bei Ems zur Gewinnung des Flugstaubes getroffenen Einrichtungen," Ems, 1882, transcription by Egleston, *Tr. A. I. M. E.*, 1883, XI, 379; abstract, *Eng. Min. J.*, 1882, XXXIV, 379; comment by Stetefeldt, *Eng. Min. J.*, 1883, XXXVI, 51; drawings of the Freiberg plant, *Freiberg. Jahrb.*, 1894, p. 39; *Min. Ind.* 1895, IV, 476; Hofman, "Lead," 1898, p. 390.

4 3/8 to 5 7/8 in. apart, from pins passing through cross-bars *L*. The dust when grown to a thickness of 1 1/8 to 1 1/4 in. becomes detached and falls to the bottom on which every 16 to 20 ft. cross-partitions, *E*, 7 3/4 in. high, bolted to cast-iron supports, prevent the settled dust from being taken up again by the draft. The temperature of the sulphurous gases may not fall below 50° C.,

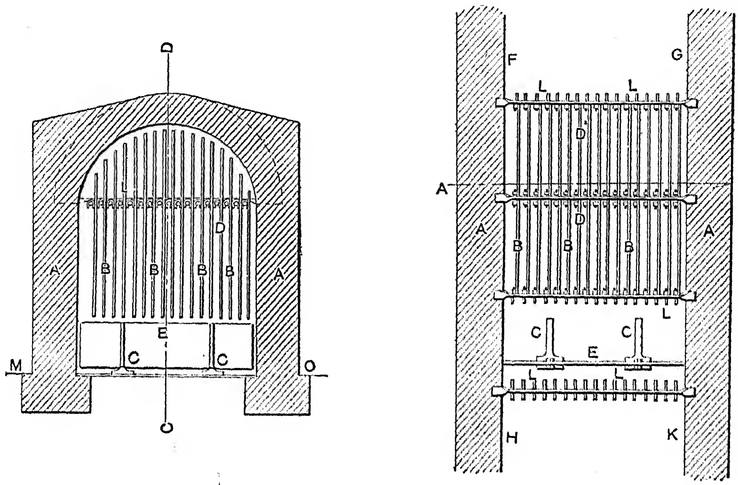
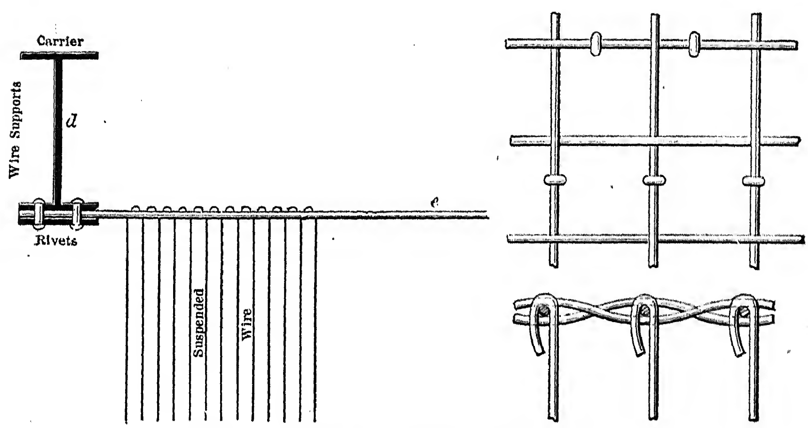


FIG. 789.—Dust-flue with Freudenberg plates, Ems, Prussia.



FIGS. 790 to 792.—Roesing wire-system.

as otherwise the plates are attacked. Drawings of the Freudenberg plates at Freiberg have been published.[1]

Roesing[2] substituted iron wires for Freudenberg plates. At Tarnowitz, Silesia,[3] where this system was first introduced, the wires are No. 10 gauge and

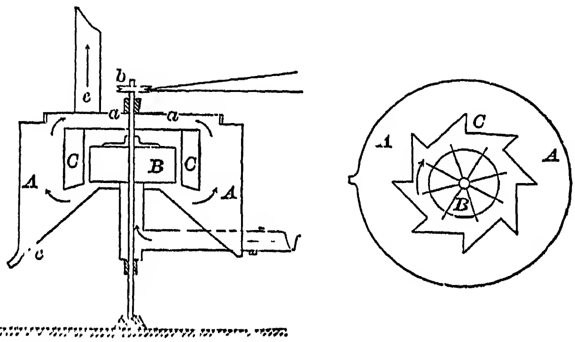
[1] *Freiberg. Jahrb.*, 1894, p. 39; *Eng. Min. J.*, 1895, XCIX, 342; *Min. Ind.*, 1895, IV, 476. Hofman, "Lead," p. 391.

[2] U. S. Patent, No. 432 440, July 15, 1890.

[3] Saeger, *Zt. Berg. Hütten. Salin. W. i. P.*, 1893, XII, 267. Hofman, "Lead," 1898, p. 302.

10 ft. long. The manner of suspension is shown in Figs. 790 to 792. A wire-screen *e* with 1-in. holes is riveted to I-beams, *d*, extending along the top of a flue which is 16 ft. 6 in. high; from it are suspended the wires as indicated. A shaking system assists in detaching the dust. The system is in operation at the lead works of Scheriau, Austria.[1] At the Boston & Montana Works, Great Falls, Mont.,[2] the Roesing system has been introduced on a large scale.

333. **Centrifugal Apparatus.**[3]—The separation of particles having a specific gravity of from 1.5 to 3.0 from furnace-gases with a specific gravity of perhaps 0.013 by means of centrifugal force is promising. The action of this force upon particles of dust and gas of equal volume is proportional to their specific gravities. With the specific gravity of flue dust at 1.5 and that of gas at 0.013,



Figs. 793 and 794.—Hempel centrifugal dust-separator.

the acceleration of the dust would be 1.5 : 0.013 = 115 as great as that of the dust. This figure is excessive, as the dust particles are permeated by gas; nevertheless there is a considerable margin. In a centrifugal fan the dust would be thrown against the casing to be removed by some suitable device. According to Babu[4] this method is in operation in France in an antimony-plant to recover fume from blast-furnace gas. Hempel's laboratory apparatus[5] with which he succeeded in separating zinc fume from a gas-current is represented in Figs. 793 and 794. A fan-wheel *B* with eight blades is driven at 1,000 to 3,000 r.p.m. in a casing, *C*, which is closed at the top and surrounded by a housing, *A*, having a conical bottom with discharge-spout, *c*, and removable cover, *a*. The dust-laden gases enter at *f*, are thrown by centrifugal force against the projections of casing *C*; the dust glides downward and is collected in housing *A*, while the purified gas leaves through pipe *e*. Centrifugal appara-

[1] *Min. Ind.*, 1906, XV, 536.
[2] Havard, *Min. Ind.*, 1908, XVII, 323.
Herrick, *Mines and Minerals.*, 1909, XXX, 259.
Goodale, *Tr. A. I. M. E.*, 1909, XL, 891.
Roesing, *Chem. Z.*, 1910, xxxiv, p. 1066.
[3] Stieber, *Stahl u. Eisen*, 1902, XXII, 391.
Mees, *Zt. Verein. deutsch. Ing.*, 1909, LIII, 602; *Rev. Mét. Extr.*, 1909, VI, 709.
[4] *Métallurgie Générale*, II, 667.
[5] *Berg. Hüttenm. Z.*, 1893, LII, 355, 365.

tus have been used so far mainly for washing tunnel-head gas of the iron blast-furnace (see below).

334. **Filtration of Fume.**[1]—This operation is confined to the recovery of the finest dust or of fume after most of the coarser particles have been settled out of the gas-current. Filtration presupposes that the gases have been cooled.

The resistance that all filters offer to the passage of gases requires the application of artificial draft, usually obtained by means of centrifugal fans placed either behind the settling chamber or before the filter chamber. Lead-smelting plants have fans capable of moving from 150,000 to 250,000 cu. ft. gas per min. All dry-filtering apparatus require periodic cleaning of the filter. Various filtering media have been used, such as coke, charcoal, twigs, wood-shavings, saw-dust, bags made of straw, coco fiber, cotton, wool, etc. With all of them the filtered gas still retains some solid matter, as the openings in a filter suited for furnace-gas cannot be made so small as to hold back all the solid matter.

Towers filled with filtering material have been put in operation repeatedly, but have been failures, at least with flue dust. The careful experiments of Iles[2] with coke, *e.g.*, showed that dry coke caught only part of the fume in roaster gases, and that after a short time the lower part of the column became choked with dust. Filtering towers are in place for removing small residual quantities of solid matter from gases already purified by other means as, *e.g.*, in reducing the solid matter from 0.44 to below 0.044 grains per cu. ft., the limit at which a gas-engine ceases to do satisfactory work.[3] The other filtering materials used in towers work are as unsatisfactorily as is coke. There remains filtering through cloth.[4] Loosely woven cotton (muslin) and unwashed wool are the two materials that are used. It is believed that the nap plays an important rôle in collecting the fumes; straw filters have been a failure. In general, it may be said that the temperature of the gases to be filtered ought not to exceed 90° C if cotton is used, and not more than 120° if wool is used; if it falls below 45 to 50° C. the acid in the gases quickly corrodes the filter. Iles[5] has shown that at 100° C. muslin becomes slightly colored but not weakened; that at 125° it turns light brown and is slightly weakened, and that at 150° it turns a dark brown and tears readily. As a rule the gases have a temperature of from 60 to 85° C. before they enter a bag-house. If warmer, they are cooled by being made to pass through flues or by diluting them with air, and frequently both methods are employed. Cotton may last two years[6] and wool eight years, although usually the life of a filter is much shorter; cotton costs about 1/6 the price of wool and collects about 1/3 less dust. Cotton is the common filtering material when the gases are free from corrosive acids or salts. It weighs 0.4 to 0.7 oz. per sq.

[1] Iles, M. W., "Lead Smelting," Wiley, New York, 1902.

[2] *Op. cit.*, p. 167.

[3] Sahlin, *Cass. Mag.*, 1905, XXVIII, 441.

[4] Ebaugh, *J. Ind. and Eng. Chem.*, 1909, I, 686; *Eng. Min. J.*, 1909, LXXXVIII, 1020.

[5] *Op. cit.*, p. 210.

[6] Eilers, *Tr. A. I. M. E.*, 1912, XLIV, 708.

ft. and contains 42 to 50 threads to the linear inch both in warp and woof; the usual grade weighs 0.6 oz and has 46 threads each way.

The use of filter-cloth either in form of zig-zag sheets or of bags, has been a success so long as the gas does not contain any sulphuric acid or soluble metallic sulphate, as these corrode both cotton and wool, the former a great deal more quickly than the latter. Gases from furnaces roasting sulphide ore have either to be treated in another way or the acid must first be neutralized.

According to the process patented by Sprague[1] the sulphuric acid of sulphurous gases is readily neutralized by means of zinc oxide, and it probably is the presence of zinc oxide in the sulphurous gases produced in the Bartlett process[2] which allows of their being filtered even by cotton bags. Zinc sulphate is practically stable below 720° C., Table 53. At the works of the U. S. Smelting Refining and Mining Co., Midvale, Utah, zincky middlings mixed with about 30 per cent. of fine coal are treated on a Wetherill grate, Fig. 208. The gases laden with zinc oxide are drawn into the flue through which pass the sulphurous gases from lead and copper blast-furnaces, copper converters and blast-roasters on the way to the bag-house. Any lack of zinc oxide is made up by feeding this material into the inlet of the suction-fan. Burnt lime is used to some extent to take the place of the more expensive zinc oxide although it is not as active an agent. Neutralization is accomplished only if the temperature of the gases is below 120° C.; usually they are cooled down to 100° C.

Gases from Mac Dougall and reverberatory roasting furnaces contain much SO_3; and consume correspondingly large amounts of neutralizing agents. Hand reverberatory roasters furnish a dust that shows a tendency to self-ignition and thus endangers the life of the filtering plant; the dust does not burn with a flame, it simply smolders.

Zig-zag filters of Brown and De Camp[3] at one time took care of the gases from ten lead blast-furnaces at Omaha, Neb. (42×120 in. at the tuyère-section). They were drawn through a flue 800 ft. long and 64 sq. ft. cross-section by a 12-ft. Murphy fan[4] making 200 r.p.m., cooled thereby to 65° C., and delivered to the filter-building nearby. This had four chambers, 29×99 ft., so arranged that any one of them could be cut out for the purpose of cleaning, repairing, or burning the settled dust. At a distance of 12 ft. above the floor were placed,

[1] U. S. Patent, No. 931,515, Aug. 17, 1909; *Min. Sc.*, 1908, LVII, 53; *Min. World*, 1909 XXXI, 533.

Ebaugh, *J. Ind. and Eng. Chem.*, 1909, I, 686; *Eng. Min. J.*, 1909, LXXXVIII, 1020.

Rice, *Min. Methods*, 1909, I, 9.

Sprague, *Min. Sc.*, 1908, LVII, 53; *Min. World*, 1909, XXXI, 553; *Eng. Min. J.*, 1910, LXXXIX, 519; *Min. Methods*, 1910, I, 519; *Eng. Min. J.*, 1911, XCI, 614 (Mammoth Smeltery).

Martin, *Mines and Minerals*, 1913, XXXIII, 323.

Hofman, *Min. Ind.*, 1908, XVII, 604; 1909, XVIII, 495; 1910, XIX, 458; 1911, XX, 487; 1912, XXI.

[2] Hofman, "Lead," p. 138.

[3] *Eng. Min. J.*, 1895, LX, 373.

[4] Fraser and Chalmers, now Allis-Chalmers, Milwaukee, Wis.

crosswise, I-beams 9 ft. apart serving as supports for the longitudinal frames over which were stretched up and down in a zig-zag manner, the filter-cloth forming gable-shaped roofs and V-shaped troughs. The dust-laden gases arrived in the lower compartment and were blown upward against the filtering surface. The dust collecting on the lower sides was removed at intervals by beaters manipulated from the outside, while the cleaned gas passed off above the filter into a stack at the end of the building. For one blast-furnace a filtering surface of 16,000 sq. ft. and about 7 h.p. was found to be necessary. Because of the difficulty in keeping the cloth clean and in repair the system was replaced by bags.

Filter-bags.—The first persons to use muslin bags for dry filtering were R. and S. T. Jones[1] in 1850; their arrangement was modified by S. Wetherill who in 1851 collected in bags the zinc oxide produced in his process. In 1876[2] or 1877,[3] G. T. Lewis and E. O. Bartlett applied bag-filtration to the recovery of fumes from the lead ore-hearth at Joplin, Mo.[4] The first experiments to collect fume from a silver-lead blast-furnace were carried on about 1880[5] in Leadville, Colo., at the Grant smelting works. Later, F. L. Bartlett used bags for the production of zinc-lead pigment from mixed zinc-lead sulphides.[6] These are the four leading classes of operations in which filter-bags are employed at present in smelting works. It is said that they have been installed in some antimony smelteries. Their use in fine crushing mills is noticed below.

The usual mode of operating is: (1) to draw the dust- and fume-laden gases by means of an exhaust fan from the furnace through a long flue that the temperature may be reduced and the coarse dust settled; (2) to further cool them, if necessary, by diluting with air admitted through branch-pipes, and, (3) to force the cooled gases into suspended bags, 18 to 24 in. in diam., and 20 to 42 ft. long, either from below or above, when the solid particles remain in the bags and the gases pass through the meshes into the open. Neutralization of acid in gases has been discussed on p. 849. As the pores of a bag in time become clogged, it is necessary to shake or beat the bags to detach dust and fume which fall and collect on the bottom. In the manufacture of zinc oxide or zinc-lead pigment the gas usually enters the top of a bag, the lower end of which is wound up into a coil that rests on the floor of the bag-house. The oxide which requires only some simple mechanical treatment to be ready for the market is thus easily collected.

In lead plants the gas is introduced at the bottom of a bag, as the recovered dust and fume is an intermediary product, which is conveniently collected on the floor to be burnt, as in heap-roasting, whereby usually enough heat is

[1] Ingalls, *Eng. Min. J.*, 1905, LXXX, 55.

[2] Clerc, *Eng. Min. J.*, 1885, XL, 4.

[3] Ingalls, W. R., "Lead and Zinc in the United States," Hill Publ. Co., 1908, p. 33.

[4] Hofman, "Lead," p. 131.

[5] Emmons, S. F., "Geology and Mining Industry of Leadville," *Monograph* XII, U. S. Geol. Survey, Washington D. C., 1886, pp. 673, 717.

[6] Hoke, Experimental Dust Catching Plant of Mapimi, Durango, *Eng. Min. J.*, 1910, LXXXIX, 857.

generated to cause the powder to sinter slightly so that it can be readily handled.

The ordinary bag-house[1] of a modern large lead blast-furnace plant contains from 3,000 to 4,500 bags, 18 in. in diam. and 30 to 33 ft. long. It is divided by a horizontal steel plate, provided with nipples, into a lower and an upper chamber. The lower or dust chamber is from 12 to 16 ft. high, the upper or bag chamber from 30 to 35 ft. high. The house is divided by vertical brick walls into as many compartments as there are flues leading into it. The gas main is usually provided with a corresponding number of branch-pipes. By this arrangement one compartment can be cut off for cleaning or repairing without interfering with the work of the others. It has been found that the life of a bag is much prolonged, if the gas is drawn away as soon as it has been filtered. Hence, in a modern bag-house, each compartment has its own sheet-iron stack, say 100 ft. high, supported by a steel breeching from the roof trusses. In the earlier plants the filtered gases were allowed to pass off into the open through shutters which close the window-openings of the house. This second method remains satisfactory with gases that are free from sulphurous acid. In later plants, a monitor flue, ending in a single stack, passed over the tops of the compartments. In the most recent plants, as stated, each compartment has its own chimney.

There is as yet no generally accepted standard for the relation between filtering area and quantity of gas produced, or quantity of ore treated. In the manufacture of zinc oxide, formerly 200 to 300 sq. ft. of bag surface per sq. ft. grate area was the common figure; this was reduced (1905) to 100 sq. ft. At Cañon City, Colo., in the production of zinc-lead pigment 150 sq. ft. bag surface per sq. ft. grate area has been found necessary. In the Scotch ore-hearth at Joplin, Mo., 3,500 sq. ft. filter surface per ton of ore treated was deemed necessary; this figure[2] has been reduced to 900 to 1,000 sq. ft. At the Globe silver-lead smelting works at first 250 sq. ft. filter area were used per ton of ore treated, but had to be increased to 750 sq. ft. At Murray[3] and Bingham Junction,[4] Utah, silver-lead smelteries, 300 to 500 sq. ft. are considered ample. At the Selby lead works[5] there are provided 410 sq. ft. per ton charge treated, or 3.413 sq. ft. per cu. ft. gas, while 1.14 sq. ft. is believed to be sufficient. Eilers[6] states that at the Murray plant there are provided 3.45 sq. ft. filtering surface per cu. ft. of gas per min., but that when one compartment is shut for sintering, there remain in operation 2.59 sq. ft. Therefore the minimum surface is 2.5 sq. ft.

Figs. 795 and 796 represent in elevation and plan part of the bag-house of

[1] Eilers, Murray plant, *Tr. A. I. M. E.*, 1912, XLIV, 708.

[2] Joplin, *Eng. Min. J.*, 1907, LXXXIV, 862 (Brittain).
Granby, *Eng. Min. J.*, 1907, LXXXIV, 388 (Perkins); *Min. World*, 1908, XXIX, 917 (Buskett); Galena, *Min. World*, 1909, XXX, 1029 (Brinsmade).

[3] Ingalls, *Eng. Min. J.*, 1907, LXXXIV, 527, 555.
Ebaugh, *op. cit.*, 1909, LXXXVIII, 1020.

[4] Rice, *Min. Methods*, 1909, I, 9.

[5] Bennett, *Eng. Min. J.*, 1908, LXXXVI, 451.

[6] *Tr. A. I. M. E.*, 1912, XLIV, 708.

the New Jersey Zinc Co., Bethlehem, Pa.[1] The gases from the oxide furnaces having been cooled and freed from ore- and fuel-dust in dust chambers, enter the 6-ft. gas main from which branch off 3-ft. horizontal pipes having top- and bottom-thimbles. The top-thimbles are covered with wire-gauze which soon becomes coated with oxide and serve as emergency outlets; to the lower thimbles are tied muslin bags 40 to 50 ft. long and 3 ft. in diam. The lower end of a bag is twisted to a coil and rests on the floor. Every 3 hr. the bags are shaken after the gas has been turned off. If shaken at longer intervals, they are quickly corroded and scorched. In dry weather they become so charged with electricity that the men who do the shaking have to wear rubber gloves. The oxide is collected every 12 hr. A bag lasts three months.

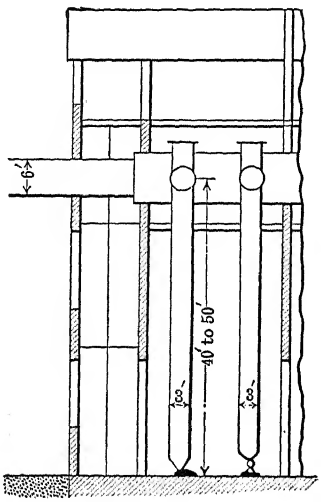


FIG. 795.—Bag-filter, New Jersey Zinc Co., South Bethlehem, Pa.

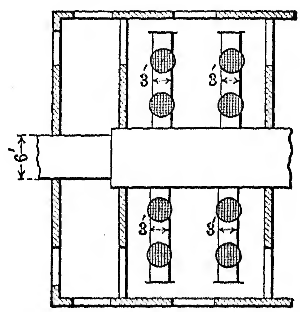


FIG. 796.—Bag-filter, New Jersey Zinc Co., South Bethlehem, Pa.

Fig. 797 shows an elevation of part of the bag-house of the Lone Elm works, Joplin, Mo.[2] in which the lead pigment coming from the slag-eye furnaces is collected after the gases have been cooled and freed from dust. The building is 40 ft. wide by 90 ft. long by 45 ft. high. It has two stories with wooden floors. The lower story, 9 ft. high, has three rows of wooden, iron-lined hoppers, nine in a row, which are suspended by iron straps from 1 1/2-in. pipes. Beneath the hoppers are wooden bins closed with canvas. There are 20 bags to a hopper, each having a surface of 168 sq. ft.

The brick bag-house in which are filtered the fumes from the ore-hearths after they have been drawn through a dust chamber and an elevated pipe 4 ft. diam. and 350 ft. long, is similar to that shown in Fig. 797. It is 95 ft. long by 50 ft. wide by 45 ft. high, has a vertical longitudinal brick wall dividing it into two compartments, and a horizontal iron platform 12 ft. high dividing it into two stories. In the lower story are 4 rows of sheet iron hoppers (truncated pyramids), 9 in a row, closed at the bases by sliding dampers. A hopper stands

[1] Taken from a drawing prepared by the late J. P. Wetherill for the Chicago Exposition of 1893; published in Ehrenworth's Report.

[2] Dewey, *Tr. A. I. M. E.*, 1889–90, XVIII, 701.

on 4 iron pipes, 3 1/2 ft. long, encased in clay pipes; its face is closed by a sheet-iron cover 5 1/16 in. thick which has 16 holes 18 in. in diam., from which thimbles, 12 in. high, extend into the upper story. A bag, made of unwashed wool, is 19 in. in diam. and 33 ft. long. Its upper end is tied with a cord and suspended from a roof truss; the lower end is slipped over the nipple and tied fast. The bags are shaken once in 6 hours, when the filtered fume falls into the hoppers which are discharged every second day. The fume (blue powder) contains enough PbS and C to roast, when gathered into heaps and ignited with oil-waste, and to sinter to a porous mass that is sufficiently strong to stand handling and charging into a blast-furnace.

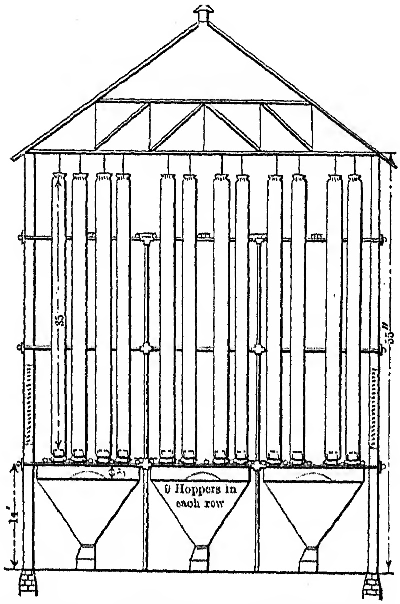


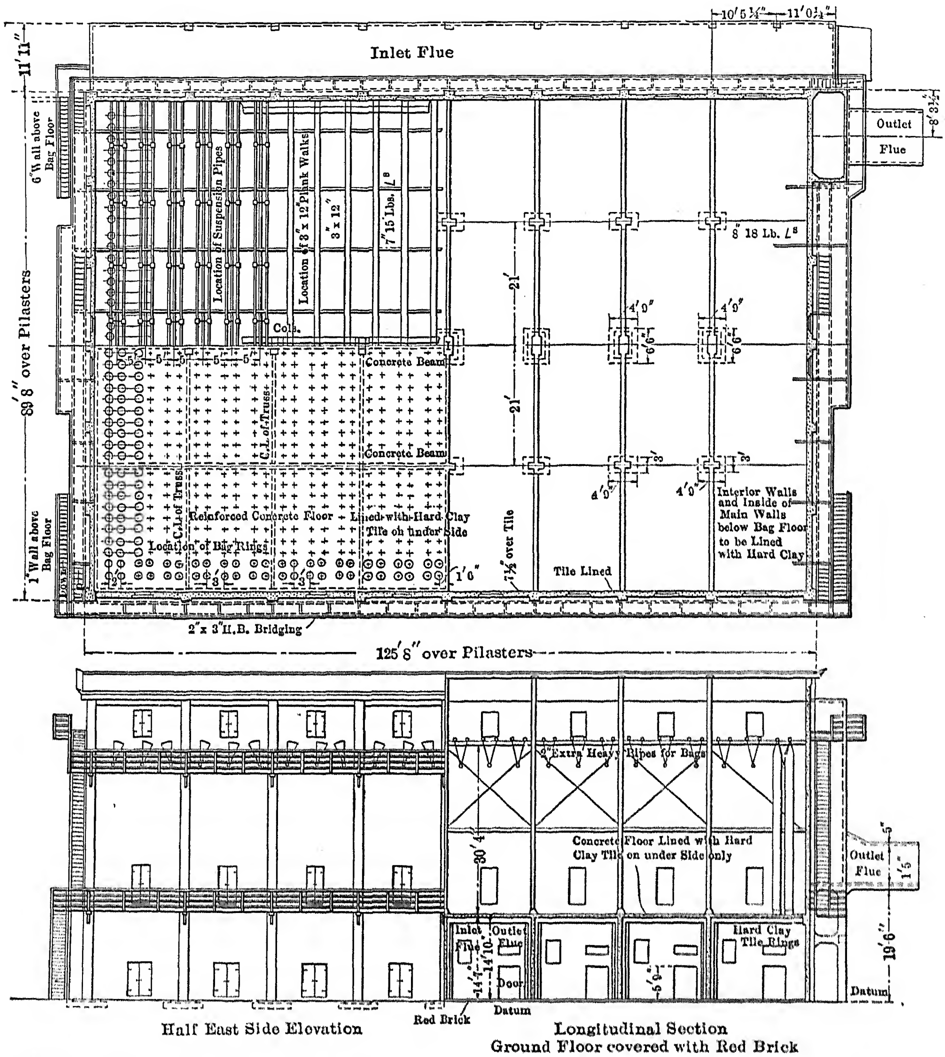
FIG. 797.—Bag-filter, Lone Elm Works, Joplin, Mo.

In making sublimed lead paint it has been found that if the suction fan was placed in the brick flue near the dust chambers, the flue between fan and bag-house was quickly corroded. For this reason the fan is placed near the bag-house in such plants.

This original construction of bag-house for ore hearth-work has been somewhat changed to suit the large volume of gas that has to be filtered in a modern lead blast-furnace plant. The bag-house of the Selby Lead Works, near San Francisco, Cal.,[1] the latest that has been erected (before 1910) may serve as an example.[2] Plan, elevation and sections are given in Figs. 798 and 799. The bag-house, 125 ft. 8 in. long by 89 ft. 8 in. wide by 67 ft. 6 in. high, is built of reinforced concrete to be safe in case of an earthquake; the intake and the side of the house are lined with fire-brick tiling to protect the concrete from acid corrosion and the floor is paved with red brick. The house is designed to take care of the gases from three blast-furnaces, 36 by 144 in., two cupelling furnaces, two 60-ton lead-bullion softening and two 60-ton lead refining furnaces which furnish a total volume of 67,851 cu. ft. of gas per min. at 45° C. The gas traveling with a velocity of 9 ft. per sec. is strained through 1,920 bags, 18 in. in diam. by 28 ft. 8 in. in effective length, representing 259,392 sq. ft. filtering surface. The fan drawing the gases from the furnaces and forcing them into the inlet-flue is a so-called 220-in. machine, 11 ft. in diam. by 5 ft. 6 in. in width; it is rated at 72,500 cu. ft. of air at 17° C. and requires 35 h.p. at 130 r.p.m., furnishing a vacuum of 0.75 oz. It is actually driven at a speed of 116 r.p.m., displaces 76,000 cu. ft. of gas of low temperature per min. and requires 24.75 h.p. In order to increase

[1] Bennet, *Eng. Min. J.*, 1908, LXXXVI, 451.
[2] Eilers, Drawings of Murray Plant, *Tr. A. I. M. E.*, 1912, XLIV, 708.

the draft of the steel stack (10 ft. diam., 180 ft. height) attached to the outlet-flue (9 ft. by 7 ft. 4 in., and situated above the inlet-flue, 8 ft. × 7 ft. 6 in.), hot gases from reverberatory and retort furnaces are introduced at the base; this supplies a vacuum of 0.43 oz. The building has seven partition walls at



FIGS. 798 and 799.—Plan, elevation and section of bag-house, Selby Lead Works, San Francisco.

right angles to the inlet-flue making eight compartments, 15 ft. wide, each of which can be cut out of the circuit. At 14 ft. 10 in. above the ground floor is a sheet-iron floor covered with concrete and tiling which divides the chamber into the bag-room above and the dust floor below. The bag-room floor has rings of

hard clay-tiling serving as nipples to hold the lower ends of the bags. These nipples are arranged in pairs 2 ft. apart; and there are three rows of these pairs 3 ft. apart in each chamber, making a total of 240 nipples. The suspension of the bags is that of H. E. Benedict.[1]

Shafts for a row of bags run across the bag-house on bearings which are placed on channel iron along the bottom cords of the roof trusses. From a shaft are suspended as many levers as there are bags in a row; the end of a lever has a loop for attaching the cord of a bag. Oscillation of a shaft causes a bag to move to and fro giving it a wave-like motion similar to that of a rope shaken at one end. Several shafts are joined together by levers and links so that oscillating one shaft causes the others to move; one of the shafts extends outside of the building where the rocking is done. This shaking device is more efficient than the usual one in which the single bags are shaken by a man dressed in a rubber suit and provided with a helmet holding a supply of compressed air for respiration; it is also cheaper in that the gas-current need not be shut off as is ordinarily the case. The bags can be shaken more frequently thus keeping open the pores of the cloth and increasing the filtering capacity of a given surface.

The dust floor of each compartment has, beside the inlet-flue, an outlet-flue and a door. The outlet-flue serves to carry off gases when the collected fume is heap-roasted; the doors allow the necessary air to enter and serve their usual purpose of admittance. The building has two galleries and the necessary doors.

Descriptions of the bag-houses of the Globe Works, Denver,[2] of the U. S. Smelting, Refining & Mining Co.'s plant at Midvale,[3] Utah, and of the Murray Smelter, Salt Lake City, Utah,[4] have been published.

The shaking of bags causes them to wear out near the top and bottom. Shaking has been entirely done away with by the plan adopted at the lead-plant, Tooele, Utah, a plan discovered accidentally at Midvale, Utah. Figs. 800 and 801 give a partial elevation and section through the bag-house at Tooele. Along the left side of the building is the main flue, *A*, carrying the cooled gases to be filtered; 3-ft. gates, *B*, closed by dampers, admit them to the bags in the dust floors, *C*, underneath the sheet-iron division which separates them from the bag-rooms. Above each 3-ft. gas-inlet is an 18-in. gas-outlet, *F*, closed by a damper, which leads into the flue, *G*, connected with a fan discharging into the main for furnace-gases. If gas-inlet, *B*, is closed and gas-outlet, *F*, opened, the pressure is taken off the bags of a bay, and the bags collapse causing the dust to become loosened and drop. By repeating the closing and opening of *B* and

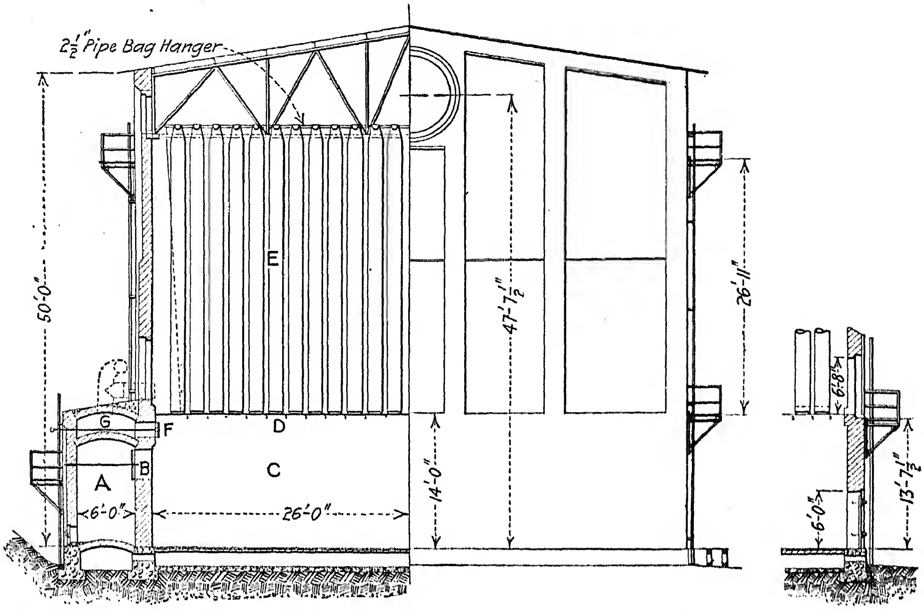
[1] *Eng. Min. J.*, 1908, LXXXVI, 1009; *Min. World*, 1908, XXIX, 633. See also Eilers, *loc. cit.*, Suspension at Murray.

[2] Ingalls, *Eng. Min. J.*, 1905, LXXX, 55.

[3] Rice, *Min. Methods*, 1909, I, 9.
Ebaugh, *J. Ind. Eng. Chem.*, 1909, I, 686; *Eng. Min. J.*, 1909, LXXXVIII, 1020.

[4] Ingalls, *Eng. Min. J.*, 1907, LXXXIV, 527, 575.
Brinsmade, *Mines and Minerals*, 1907, XXVIII, 216.
Eilers, *Tr. A. I. M. E.*, 1912, XLIV, 708.

F about three times all the dust is shaken off. This is done with every bag at the beginning of an 8-hr. shift and thus the filtering surface kept clear and open.



FIGS. 800 and 801.—Part elevation and section, bag-house of lead plant, Tooele, Utah.

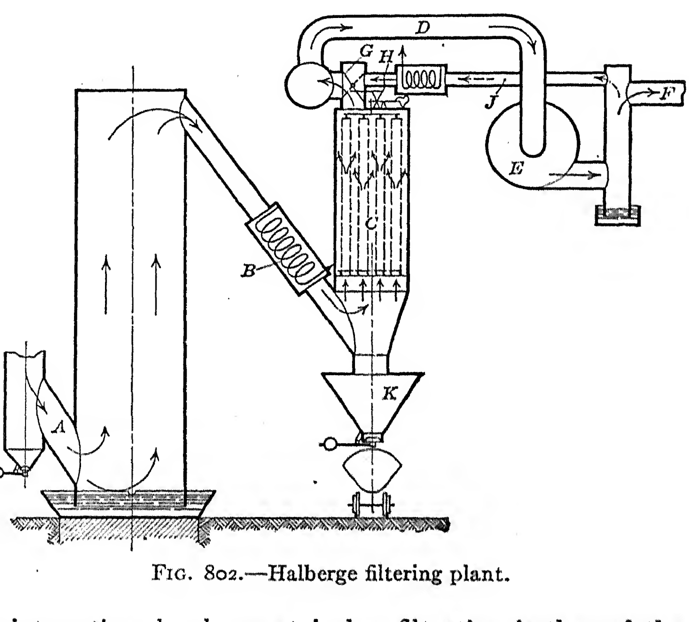


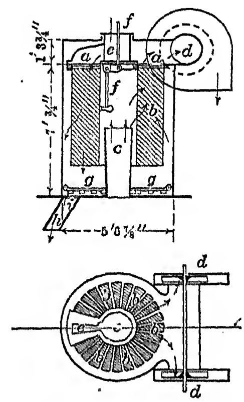
FIG. 802.—Halberge filtering plant.

Another interesting development in bag-filtration is that of the Halberge filtering plant. Müller[1] describes the working of the experimental filtering

[1] *Stahl u. Eisen*, 1911, XXXI, 229; also Gouvy, *Rev. Un. Min.*, 1912, XL, 54.

plant for 30,000 cu. ft. gas per minute at the iron blast-furnace works of Halberge, near Saarbrücken, Germany, the self-cleaning device of which has much interest. The gas enters the apparatus with a temperature ranging from 80 to 120° C. and charged with 6 g. dust per cubic meter, it leaves it at 60° C. and with 0.01–0.30 g. per cubic meter. The apparatus is shown in Fig. 802. The gas arriving through *A* enters the water-sealed settler in which it is cooled to the dewpoint in order that the dust particles may become wetted. Upon leaving, it enters the chamber *B*, warmed by waste steam or waste gas, in order to have its temperature raised to 10 or 20° C. above the dewpoint before it passes into the bag-room *C*, containing from 8 to 12 bags, 10 ft. long and 8 in. in diam. The filtered gas passes the turn-stile damper *G* into pipe *D* and through fan *E*, which delivers it into the outlet pipe *F*. After traveling in this direction for about 4 min., the damper G is turned mechanically to occupy the second position, shown by the dotted line, when the back pressure from the outlet pipe *F* forces cleaned gas through pipe *J* with its heated chamber, and thence through the outer surface of the bags; the dust gathered on the inside is loosened, the bags are shaken by the mechanism *H*, and the dust drops into the hopper K to be discharged at intervals. This cleaning takes from 15 to 20 sec. By means of the device a small filtering surface can handle a large volume of dust-laden gas, and recover the dust automatically.

A filtering apparatus for small quantities of fume is that of Nagle and Kaemp[1] shown in Figs. 803 and 804. It is in operation at the silver-lead works of Tarnowitz, Silesia, to collect the fumes of cupelling furnaces. In a sheet-iron cylinder, 8 ft. 10 1/2 in. high by 5 ft. 6 7/8 in. in diam., are suspended a number of filtering-frames *b* covered with flannel; fume-laden gases enter at *c*, and the filtered gases pass off through slots *a* in the frames into two centrifugal fans *d*. The cloths are cleaned by means of a beater *f*, and a revolving cylinder *e*, which is connected with the outside air and is provided with a sectoral attachment. When the sector is moved over the slot *a*, air enters the frame and blows off the dust that has accumulated on the cloth; the dust falls to the bottom and is transferred by means of mechanical rake *g* to discharge-spout *h* through valve *i*.



FIGS. 803 and 804.—Nagler and Kaemp filter.

There are many filtering devices for removing fine dust in dry-crushing mills which are mostly borrowed from flour mills, as, *e.g.*, the Vibracone,[2] the Morse,[3] the Perfection,[4] the O'Brien,[5] dust collectors. The monographs of

[1] Kohfahl, *Iron Age*, 1893, LII, 468.
Saeger, *Zt. Berg. Hütten. Salinen W. i. P.*, 1893, XLI, 287.

[2] *Iron Age*, 1908, LXXXI, 844.

[3] *Iron Age*, 1908, LXXXII, 114.

[4] *Zt. Berg. Hütten. Salinen W. i. P.*, 1908, LVI, 343; Prinz & Rau Mfg. Co., Milwaukee, Wis.

[5] *Min. Ind.*, 1900, IX, 623.

Gertner,[1] Baldus,[2] Russwurm,[3] and Osborne[4] deal with the various apparatus for collecting dust formed in briquetting plants.

335. Freezing of Moisture in Fume.—This is the reversal of the Gayley dry-blast (§ 327). It has been suggested by Carpenter[5] who proposed to cool furnace-gas to 100° C. by finely-divided water, and then pass it through a refrigerating chamber when the frost formed would carry with it not only all the floating dust and fume, but the sulphurous gas as well. Thawing the frost would furnish concentrated sulphurous acid and a mud containing the valuable constituents. According to Mathewson[6] a plant of this character for the Anaconda copper smeltery would cost $4,000,000 and the cost of operation would amount to $10,800 per day. This figure would be greatly reduced by using the Bruce modification of the Gayley plan.

336. Some Constructive Details.[7]—In all smelting works there is produced more or less smoke; it may issue from furnaces or flues, from temporary openings, and from molten or heated materials. The lack of draft causing smoke to issue from flues may be due to an insufficient area, to a temporary obstruction, to a faulty construction, the use of bends of insufficient radii, to the use of T instead of Y connections, etc. In carrying gases from a blast-furnace by means of a bent iron flue, a stronger draft will be required if the bend is downward than if upward, as the smoke cools steadily from the moment it leaves the furnace. In the first case, the denser gas has to be pushed up by the lighter; in the second the gas ascends, and there is less difficulty, as when once in motion it will reach the top readily. In handling smoke issuing from hot material that has been drawn from a furnace, it is essential to remove it before it becomes diluted with air; hence small hoods placed over the openings or receptacles will be most effective, and here artificial draft is more effective than natural.[8]

In ventilating a smelter-building, the harmful effect of horizontal air-currents which blow smoke out of its normal path and dilute it, is counteracted by partition walls. The usual monitor-roof is always successful in removing gas, if it has balanced swing-doors so connected that when one side is open the other will be closed.

337. Electric Condensation.—Electrostatic discharges have been frequently experimented with[9] for the settling of fine particles suspended in air. While

[1] *Zt. Berg. Hütten. Salinen W. i. P.*, 1908, LVI, 257.
[2] *Glück Auf*, 1908, XLIV, 1728, 1760.
[3] *Braunkohle*, 1910, IX, 65.
[4] Osborne Engineering-Mfg. Co., 141, Broadway, New York City.
[5] *Proc. Colo. Sc. Soc.*, 1908, IX, 65.
[6] *Min. Ind.*, 1908, XVII, 323.
[7] Messiter, *Min. Sc. Press*, 1908, XCVII, 26.
[8] Bennett, Apparatus of the Selby Lead Works, *Eng. Min. J.*, 1908, LXXXVI, 604.
[9] Hutchings, *Eng. Min. J.*, 1886, XLI, 336; *Berg. Hüttenm. Z.*, 1885, XLIV, 253.
Roesing, *op. cit.*, 1885, XLIV, 290.
Bartlett, *Eng. Min. J.*, 1886, XLI, 195.
Editor, *op. cit.*, 1885, XL, 39.
Iles, *Sch. Mines Quart.*, 1894–95, XVII, 109.
Strong, *J. Frankl. Inst.*, 1912, CLXXIV, 239.

they have been successful with air in repose, they have been a failure when the air was in motion until Cottrell[1] devised his direct-current electrostatic process. This is now in operation at the Selby lead works near San Francisco, at the Garfield Works near Salt Lake City, and has been tried at the Balaklala copper smeltery, Coram, Cal.

Experiments of Lodge and Walker have shown that an alternating current will cause the agglomeration of particles suspended in a fairly quiescent atmosphere. In a moving gas-current the agglomeration and settling are too slow to be effective. Cottrell, therefore, uses a high-potential direct current. If a needle point be connected with one pole of such a current and a flat plate with the other, the air space between the two becomes highly charged with electricity of the same sign as the needle point, and any insulated body brought into this space receives a similar charge. If this body is free to move, as is the case with the particles of suspended flue dust and vapor, these will be attracted to the plate of the opposite charge; they will move, therefore, toward the plate and do this at a speed which is proportional to the charges and to the potential gradient between the point and the plate. Observation has shown that in a perfectly transparent gas there is a strong wind from point to plate.

In order to obtain a high-potential direct current, an alternating current from a lighting or power circuit of from 20,000 to 30,000 volts is connected by means of a special rotary contact maker driven by a synchronous motor.

For the needle point of the laboratory there was substituted at first a cotton-covered wire, the loose fiber of which furnished innumerable fine, sharp points for the discharge of the high-potential current, since it was sufficiently conductive from its natural hygroscopic water. The cotton, readily attacked by hot acid gas, was replaced by asbestos or mica, the fiber, or the edges of the scales furnishing the desired discharge points.

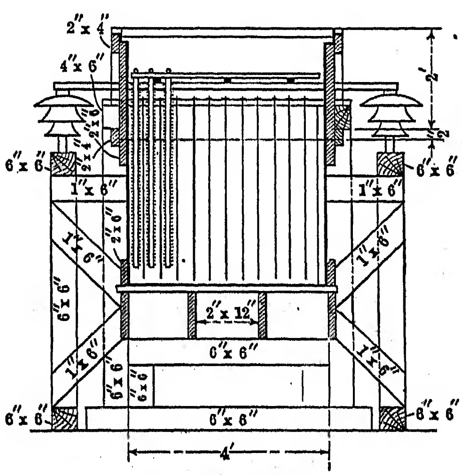


Fig. 805.—Cottrell electric fume condenser, Selby Smelting & Lead Works, San Francisco, Cal.

At the Selby works, Fig. 805, the first unit was introduced in a gas-current of 5,000 cu. ft. per min., to condense the fumes resulting from the sulphuric-acid parting plant. In a lead flue, 4×4 ft., there were placed rows of vertical lead plates 4 in. apart,

[1] Editor, *Eng. Min. J.*, 1908, LXXXVI, 375; Havard, *Min. Ind.*, 1908, XVII, 324; Braden, *Min. Sc. Press*, 1909, XCIX, 192; Editor, *op. cit.*, 1910, CI, 696, 859; Editor, *Min. Sc.*, 1911, LXIII, 338; Cottrell, *J. Ind. Eng. Chem.*, 1911, III, 542; *Eng. News*, 1911, LXVI, 495; *Min. Sc. Press*, 1911, CIII, 255, 286; *Min. Sc.*, 1911, LXIII, 337; *Eng. Min. J.*, 1911, XCII, 763; *Tr. A. I. M. E.*, 1912, XLIII, 512; *Met. Chem. Eng.*, 1912, X, 686.

alternating with suspended lead-covered iron rods carrying mica. A current of 460 volts and 60 cycles was transformed up to 17,000 volts, and then passed through a rectifier to the electrodes. The power consumption was found to be only 2 kw.; the result attained was an elimination and collection of the sulphuric acid in the gas as a liquor of 40° Bé. The plant has been in operation for four years; the monthly outlay is $20; the acid recovered in three years has paid more than five times the cost of the installation.

The second unit was intended to handle 50,000 cu. ft. of gas per min. from the roasting and blast-furnace departments. It was thought at first that the suspended material would be precipitated in a reasonably dry form, which would permit brick and iron construction. Experience showed that this was not the case, and that lead was necessary as building material. A flue 6×6 ft. and 32 ft. long, containing 38 rows of 16 lead plates, each 6 ft. long and 4 in. wide, alternating with discharge electrodes, was sufficient to collect the sulphuric acid, arsenic, antimony, lead, etc., in the gases, provided these had been cooled to below 150° C. The collected material is a grayish mud, easily washed off the plates and run into settlers. For this purpose, every four to six hours the gases were turned into a bypass. For continuous work two or three units would have to be erected so that one could be cleaned at stated intervals. The power consumption was from 10 to 15 kw. A permanent installation has not yet been made.

At the Garfield works the gases from a basic matte converter which carry lead fumes are drawn through an electric condenser. In this case the vertical anodes, iron forked rods holding the mica, are suspended in grounded iron tubes; the converter gases heavily laden with lead fumes enter the bottom of the condenser and leave it completely clarified, the lead having been deposited on the cathode tubes.

At the Balaklala smeltery the combined gases from the blast-furnaces, MacDougall kilns, oil-fired reverberatory smelting furnaces and converters were subjected to electric condensation to collect the dust and fume. The average condensation was 75 per cent.; when the apparatus was under control over 90 per cent. of all the solids and vapors were collected. The works were closed before the electric condensation had been fully perfected.

338. Wet Methods of Condensation in General.[1]—An efficient condensation of smelter smoke in this way must do three things: (1) reduce the temperature of the gas-current to the point at which dissolved moisture forms a mist or fog; (2) saturate the floating particles with moisture, and (3) separate the saturated particles. It is convenient to make a distinction between the apparatus that has been employed in non-ferrous plants and the modern gas-washing apparatus of iron blast-furnaces which have come into use since 1894.[2]

[1] Osann, *Stahl u. Eisen*, 1902, XXII, 153.
Defays-Lanser, *Rev. Mét.*, 1908, V, 121.

[2] H. B. Thwaite's application of patent for purifying iron blast-furnace gas for use in gas-engines was granted May, 1905; the first gas-engine was set to work in 1895 by James Riley in Wishaw, Scotland.

339. Non-Ferrous Plants.[1]—Wet condensation in a non-ferrous plant usually has to meet the harmful effect of the sulphur content of the gases, the dissolved sulphurous and sulphuric acids readily attacking iron, brick and mortar. Wet methods are therefore applied only in places where it is necessary to render these gases harmless. With the method arises the requirement of settling the suspended mud and neutralizing or perhaps utilizing the acid water. At first gases were drawn or forced through water by means of a pump or an Archimedean screw.[2] In order to overcome the difficulty of thoroughly wetting the fumes, screens of copper wire placed one above the other were immersed in the water and thus the gas-bubbles broken up, but the state of division was not sufficiently fine and the consumption of power too great. The next improvement was to pass the gases through filters of faggots, brushwood, coke, etc.[3] If the filter is coarse enough to allow the gas to pass freely, the gas is not sufficiently wetted; if fine enough to accomplish this, the openings very soon become clogged.

Another method which is also effective is to conduct the gas-current through towers having staggered shelves or provided with acid-proof clay or porcelain rings or pipes[4] as is customary in the Glover towers of sulphuric-acid plants.[5] Iles,[6] using staggered shelves, succeeded in recovering 95 per cent. of the solid matter contained in the gases coming from roasting furnaces treating lead ores. The Richter condensation tower[7] is in operation at Freiberg, Saxony, and Tarnowitz, Silesia, and did satisfactory work at Leadville, Colo., in a pyrite-smelting plant in collecting small amounts of lead and silver-bearing fume. The tower of Tarnowitz,[8] which differs little from the original construction at Freiberg, is shown in Figs. 806 to 809. The tower of sheet lead, mounted in a wooden framework similar to that of a sulphuric acid chamber, is 6 1/2 by 10 ft. and 23 ft. high; it contains 90 gable-shaped roofs of sheet lead, 10 ft. long and 7 7/8 in. high, supported by lead-covered iron bars; the slope of the roofs is greater than the angle of repose of the mud. The lower borders of the roofs have incisions about 1 1/8 in. deep in order that the condensing water may run off freely and leave open the interstices necessary for the ascending

[1] French, *Eng. Min. J.*, 1880, XXIX, 49, 84.
Iles, *Sch. Min. Quart.*, 1895–96, XVII, 103.

[2] Eilers, *Tr. A. I. M. E.*, 1874–75, III, 310.

[3] Fallize, *Rev. Univ.*, 1862, XI, 367.
Percy, "Lead," p. 441.

[4] *Op. cit.*, p. 105, example of the Saint Louis S. & R. Co.
Hofman, "Lead," 1899, p. 381.

[5] Lunge, G., "Sulphuric Acid and Alkali," 1913, I.

[6] Iles, *op. cit.*, p. 168.

[7] *Freiberg. Jahrb.*, 1889, p. 57; *Berg. Hüttenm. Z.*, 1890, XLIX, 129; *Eng. Min. J.*, 1890, XLIX, 196.
Doolittle-Jarvis, *Tr. A. I. M. E.*, 1910, XLI, 709.
The De-Arsenizing Tower" (Bräuning, *Zt. Berg. Hütten. Sal. W. i. P.*, 1877, XXV, 133) for freeing sulphuric acid from dissolved As_2O_3 by means of H_2S, works on the same principle as the tower of Richter.

[8] Babu, Metallurgie Générale, vol. II, p. 672.

gases. The roofs are so placed that the discharge of one set strikes the ridge-poles of those next below. The condensing water is fed through nine pipes, *l*, on to tilting distributors and thus spread evenly over the perforated box *m*.

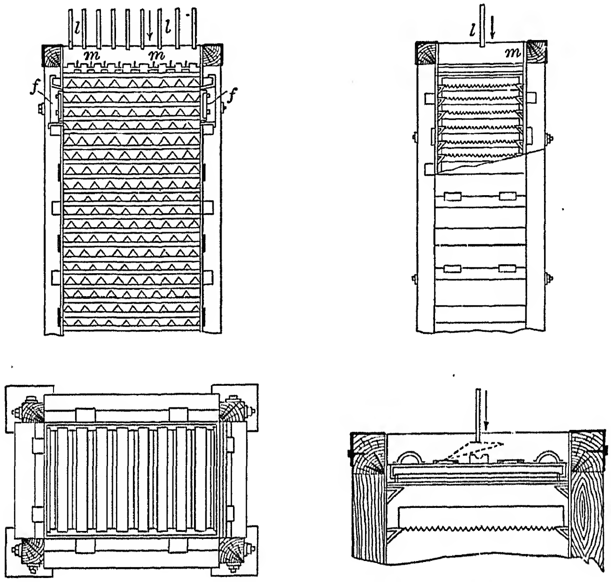


FIG. 806 to 809.—Richter condensation-tower.

At Freiberg the temperature of the entering gases of 100 to 115° C. is reduced to 40–50°. The water trickling down the roofs carries with it the condensed fumes and flows from the bottom of the tower into settling tanks.

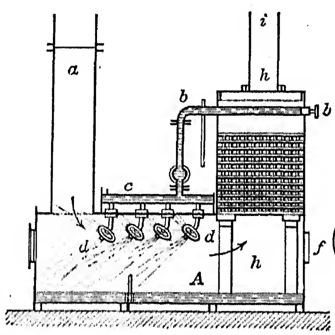


FIG. 810.—Lundin gas-washer.

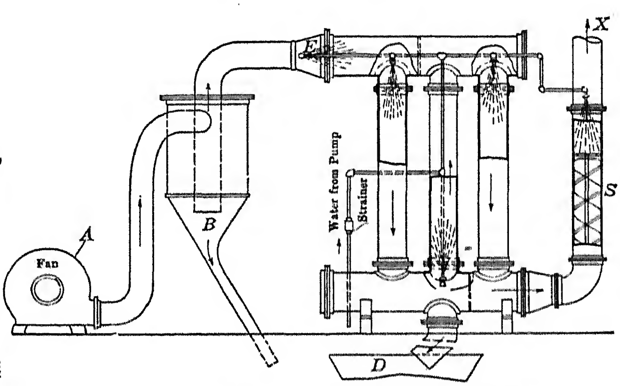


FIG. 811.—Koerting & Schutte wet condenser.

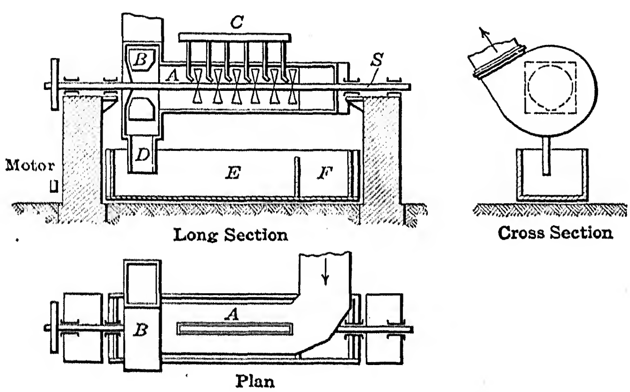
Introduction of a spray of water under pressure into the gas-current gives satisfactory results. The gas-washer of Lundin,[1] originally constructed for washing tunnel-head gas from the iron blast-furnace, may serve for treating

[1] Vicaire, *Bull. Soc. Ind. Min.*, 1867–68, XIII, 642; *Berg. Hüttenm. Z.*, 1867, XXVI, 317.

gases from non-ferrous furnaces if made acid-proof. In Fig. 810 fume-laden gas arrives in pipe *a*, meets in tank *A*, fine sprays of water projected from nozzle *d*, enters tower *h*, filled with bars laid cross-wise and sprayed from the distributing-pipe *b*, and leaves through pipe *i*; *f* represents man-holes.

A more recent form of spraying device is that of Koerting and Schutte,[1] Philadelphia, shown in Fig. 811. Dust- and fume-laden gas is delivered by fan *A*, to settler *B*, where coarse dust is separated from fine dust and fume. It now enters the spray-condenser at *E*, travels in the direction of the arrows and leaves the apparatus through pipe *x*, provided in the spiral *S*, to insure final intimate contact of water and gas.[2]

The primary gas-washers which have come into use in iron blast-furnace plants represent the more recent types of apparatus for wet condensation. They are applicable to non-ferrous plants provided the material of construction is adapted to come in contact with sulphurous gases. The wet condensation



FIGS. 812 to 814.—Wet condensation plant, Lead Works, Neudorf, Harz Mts., Germany.

plant of Neudorf, Harz Mts.,[3] Figs. 812 to 814, may serve as an example. At the entrance of the pipe, conducting hot gases from the furnace, into the branch-flue is placed a small spraying nozzle which delivers just the amount of water that the heat of the gases can vaporize. The mixture of gas and steam travels about 75 ft., and reaches the main flue in which is installed a larger spraying nozzle to cool the gas steam current to 70 to 90° C., the water-vapor condensed to mist, forming small drops, thoroughly wets the fume and dust. About 150 ft. back of the large sprayer is installed the collector shown in Figs. 812 and 813. Fan *B* draws the moistened gas and fume into the box *A*, 3 ft. 3 3/8 in. square, which is inclined toward the fan. In it revolves shaft *S*, carrying six propeller shaped fans 7/64 in. thick, which are wetted by 1 1/2-in pipes delivering spray water from the launder *C*. The water is thrown out radially by the fans and forms curtains through which the gases are forced to travel. This collects the

[1] *Eng. Min. J.*, 1909, LXXXVII, 863.
[2] Another form, *Iron Age*, 1910, LXXXV, 865.
[3] Wynne, *Eng. Min. J.*, 1909, LXXXVIII, 602.

small drops in the gas which carry the particles of dust and fume, runs down box *A*, and passes through channel *D*, into settling tank *E* and overflow-compartment *F*, from which the clarified water is drawn off by a centrifugal pump to a cooling tower, whence it returns to launder *C*. A plunger pump draws the settled mud from *E* into a filter-press, from which the clear water runs back into *F*. A small amount of fresh water is added at intervals to launder *C*, to make up for loss by leakage and evaporation. In order to neutralize the acidity of the water daily, from two to three bucketfuls of a creamy emulsion of lime in water is added to tank *E*; the lime in addition acts as a binder for the mud collected in the filter-press. The plant treats 10,590 cu. ft. gas per min. and requires 17 h.p.; 80 per cent. of the solid matter is recovered.

The flue between the larger spray-nozzle and the collector is 8 ft. 2 1/2 in. wide by 7 ft. 2 5/8 in. high to spring of arch; the walls, 4 1/2 in. thick, are of red brick, laid in a mortar consisting of clean dry quartz, sand and water-free tar; they are strengthened every 8 ft. 2 1/2 in. by buttress-walls of the same thickness; the thrust of the arched roof is taken up by channel irons 4 3/4 in. high, tied at intervals of 4 ft. 1 1/4 in. by 7/8-in. iron rods covered by 1-in. lead pipe.

340. Ferrous Plants.[1]—The tunnel-head gases of an iron blast-furnace have a temperature ranging from 120 to 200° C. and carry[2] under normal conditions perhaps 1.3 to 5.3 grains coarse and fine dust, and 150 to 160 grains water per cu. ft. After having passed through one or more dust catchers (dry cleaners) this amount is reduced to from 0.66 to 2.64 grains, and the temperature to about 80° C. These data vary greatly; thus Grimm[3] gives 1.5 to 6.1 grains, Lürmann[4] 2.2 to 8.7, Thwaite[5] 4, Grüber[6] 5.2 to 19.6, Myjes[7] 3.5 to 6.5 grains. From the dust catcher the gases pass to the washer, which has to fulfil three functions: (1) to reduce the temperature from 80 to 20° C.; (2) to diminish the dust from 0.66–2.64 to 0.130 grains if the gas is to be burned in stoves or boilers, and to below 0.044 grains if it is to be used in gas-engines, and (3) to remove the condensed water and thus dry the gas. The washing is usually divided into two steps, primary and secondary; primary washing purifies the gas sufficiently for stoves and boilers, secondary makes it suited for gas-engines.

[1] Lürmann, *Stahl u. Eisen*, 1884, IV, 35; 1898, XVIII, 247; 1901, XXI, 434.
Osann, *op. cit.*, 1902, XXII, 153.
De Mocomble, *Rev. Mét.*, 1905, II, 190.
Reinhardt, *Tr. A. I. M. E.*, 1906, XXXVII, 669.
Junge, *Iron Age*, 1906, LXXVIII, 542, 602.
Review, *Stahl u. Eisen*, 1909, XXIX, 1791, 1833.
Düsseldorff Exposition, *op. cit.*, 1910, XXX, 1397, 1437; *Met. Chem. Eng.*, 1910, VIII, 475.
Freyn (South Chicago), *Iron Trade Rev.*, 1910, XLVI, 1189, 1220, 1273; XLVII, 29, 28, 127, 180, 224, 635, 673, 743; *Iron Age*, 1910, LXXXV, 1518; *Stahl u. Eisen*, 1910, XXX, 1609.

[2] Sahlin, *Cass. Mag.*, 1905, XXVIII, 437.

[3] *J. I. and St. I.*, 1901, I, 59.

[4] *Stahl u. Eisen*, 1901, XXI, 439.

[5] *J. I. and St. I.*, 1903, I, 246.

[6] *Stahl u. Eisen*, 1904, XXIV, 9.

[7] *Op. cit.*, 1906, XXVI, 27.

Gas-washers are conveniently classed as stationary washers, slow-revolving fresh-contact washers, and rapid-revolving atomizing washers.

Before the introduction of special washing-apparatus the settling of dust was assisted in many instances by forcing the gas to impinge upon water and then travel over it. A flue serving this purpose, called *S*- or Lotharingian gas-washer, is shown in cross-section in Fig. 815. This shows a cylindrical gas-main *a* open at *b* and closed at the bottom by a pan, *c*, and a water-seal; the gas enters at *f* and leaves at the opposite end; *d* and *e* are manholes and safety-valves spaced 7 ft. apart; accumulating dust[1] is raked out periodically. The washer at Tarnowitz, Silesia, is 5 ft. in diam. and 26 ft. long.

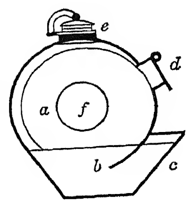


FIG. 815.—S- or Lotharingian gas-washer.

A. Stationary washers were the first to be used. Some of the older forms are discussed by Lürmann.[2] Later, towers[3] filled with coke or loose material to act as a filter found favor for some time; the open spaces, however, soon became clogged and necessitated a considerable increase in the number of scrubbers to allow for periodic cleaning. The expense of this was the cause of their being replaced by self-cleaning washers.

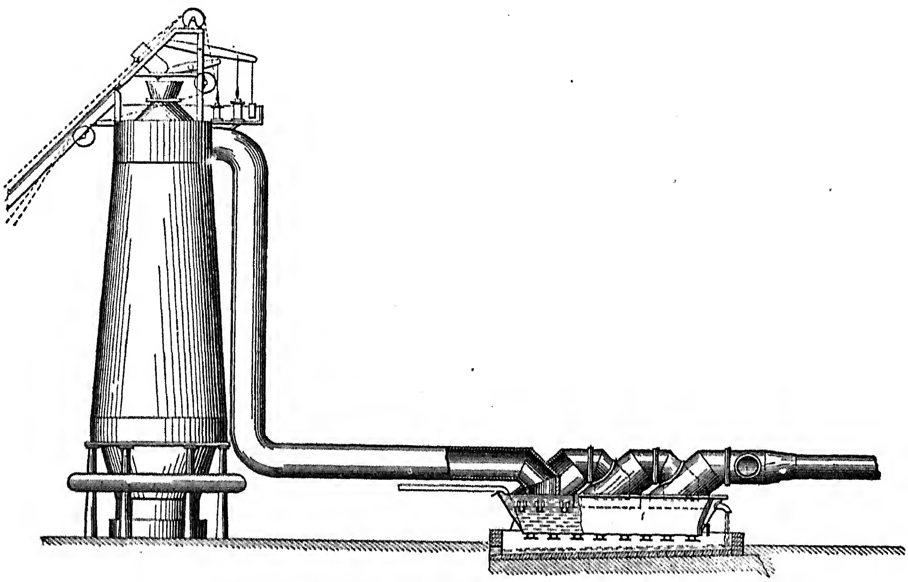


FIG. 816.—Brier Hill gas-washer showing connection to furnace.

The Brier Hill Gas-washer,[4] which is in operation at Youngstown, O., is intended to remove the large amount of fine dust from Messabi ore carried off by the gases. It is shown in Fig. 816; it acts upon the principle of the ordinary

[1] Steuer, Removal of Mud, *Stahl u. Eisen*, 1911, XXXI, 1759.
[2] *Stahl u. Eisen*, 1884, IV, 35.
[3] Thwaite plant, *Iron Age*, Dec. 3, 1903. p. 24.
[4] *Iron Trade Rev.*, Jan. 22, 1903, p. 52.

wash-bottle. The gas is forced to impinge upon the surface of the water in every downward passage, when the heavier particles of dust are wetted and sink to the bottom of the tank. The Meehan[1], Nisbet[2] Steinhart[3] and Feld[4] gas-washers work on a similar principle.

The Bachman Centrifugal Gas-washer,[5] in operation at Port Henry, N. Y., and shown in Fig. 817, has the usual form of a dust-catcher, water is made to flow down the interior surface, and the gas, admitted through a tangential inlet, takes a helical path in its passage through the apparatus. The solid particles

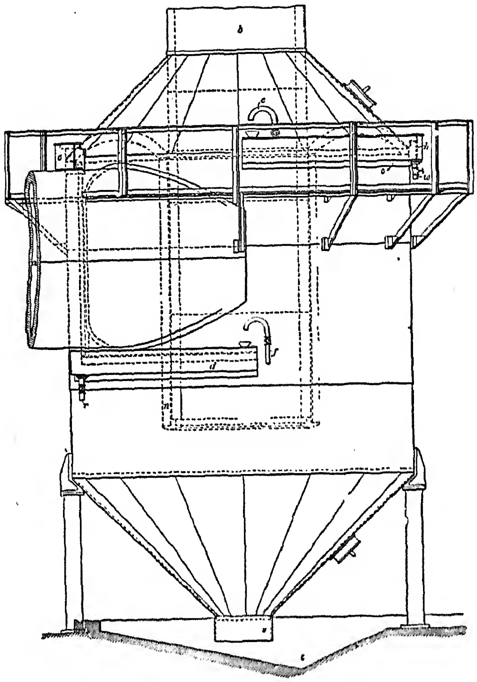


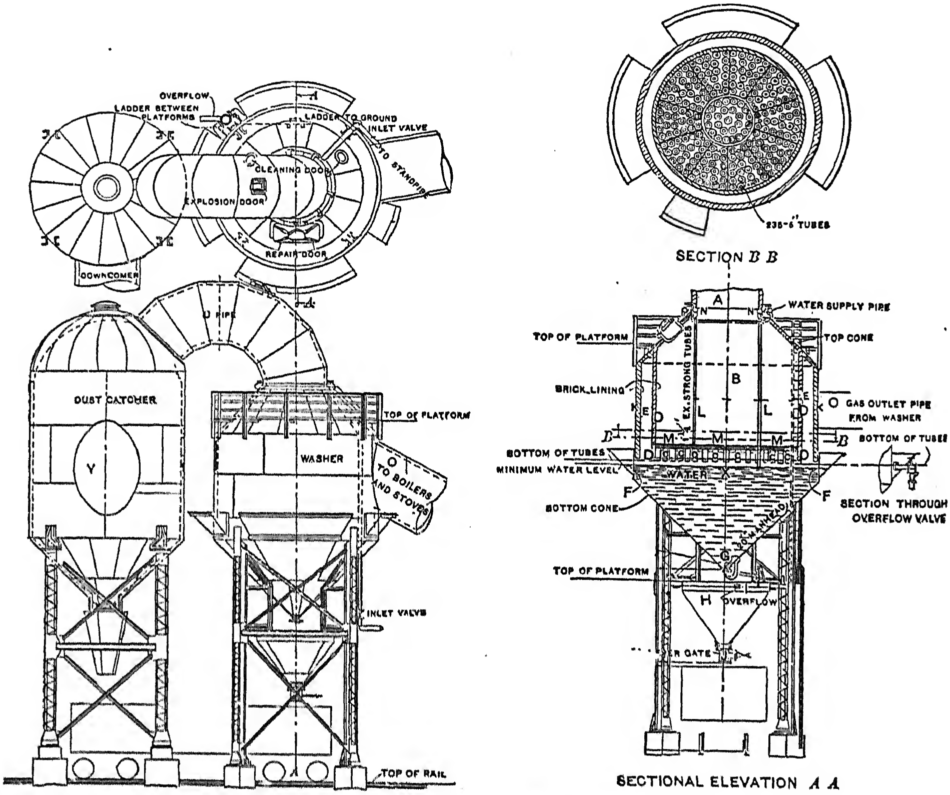
FIG. 817.—Bachman centrifugal gas-washer.

are thrown against and into the descending sheet of water, wetted and carried down. A washer 18 ft. in diam. will clean the gas from an iron blast-furnace taking 50,000 cu. ft. air per min. An average of three tests gave gas-temperature at entering 240° C., at leaving 200° C., increase in moisture 4.4 grains per cu. ft., dust entering 14.1 grains per cu. ft. and leaving 0.03.

The Lamond gas-washer[6] aims to combine gas-washing and gas-reheating by utilizing the heat from the unwashed gas for heating the washed gas before it enters the stoves and boilers.

[1] *Iron Trade Rev.*, July 26, 1906, p. 25.
[2] *Iron Age*, 1908, LXXXI, 136.
[3] *Iron Trade Rev.*, 1911, XLVIII, 647.
[4] *Met. Chem. Eng.*, 1912, X, 436.
[5] *Iron Trade Rev.*, 1907, XL, 593; *Iron Age*, 1907, LXXIX, 1124.
[6] *Iron Age*, 1907, XIX, 198.

The Mullen Gas-washer[1] is represented in Figs. 818 to 821 as used at the works of the Cherry Valley Iron Co., Leetonia, O. Figs. 818 and 819 are the elevation and plan of a dust catcher connected with the dust-washer. The gases from the blast-furnace enter the catcher tangentially through down-comer *Y*, drop the coarse dust, leave the apparatus near the top, pass through an inverted U-pipe and enter the washer, Figs. 820 and 821, at the top through pipe *A*. The washer consists of an outer sheet-iron cylinder *K*, with conical top and bottom, to which is attached inner shell *D*; the latter has an angle at the bottom



FIGS. 818 to 821.—Mullen gas-washer.

for the support of the rim of tube-plate *M*; pipe-hangers *L* give the plate additional support and furnish the wash-water. The plate has 235 openings to receive tubes *C*, 6 in. in inside diam., each of which is cast with a small collar to hold it in place when it has been slipped into its hole in the plate. The ends of the tubes are 4 in. above the level of the water in the conical water-seal basin; this is provided with bell-valve *G* for the discharge of mud into hopper *H*, and valve *G* with a lower valve *J* and water-overflow *I*. The water is admitted through pipes *L* and overflows at *F*. Whenever bell-valve *G* is opened, the

[1] *Op. cit.*, 1905, LXXV, 998; *Iron Trade Rev.*, July 16, 1905, p. 61.

water-level in the hopper sinks far enough for the pressure of the gas to blow to the side fine dust that has accumulated on the surface, and drive it through openings in the outer shell *K* into the overflow-trough *F* whence the waste water carries it away. Thus the surface of the water is readily kept clean. The gas entering at *A*, passes downward through the 235 pipes. The subdivided current impinges upon the water, rises in the space between the pipes, which has a crosssectional area 20 per cent. larger than that of the pipes, descends in annular space *E* and leaves through outlet-pipe *O*, Figs. 818 and 819.

At Gary, Ind.,[1] this washer has been modified. The cylinder filled with tubes has been replaced by an inverted funnel with a deeply fluted surface which subdivides the gas-current. There are three washers for two furnaces, one washer being held in reserve for cleaning purposes.

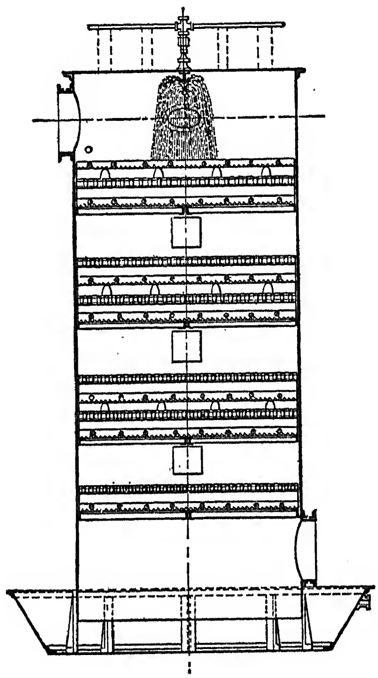


FIG. 822.—Zschocke gas-washer.

The Zschocke gas-washer[2] works on a principle different from that of the preceding apparatus. It has found much favor in the condensation of ammonia in by-product coke-oven plants, and is now making headway as a primary gas-washer. A vertical section of the tower and a perspective view of the washing boards are given in Figs. 822 and 823. The plate-iron tower is carried by supports placed in a pan filled with water to form a seal. It has a water-spray at the top and is filled with grids of beveled washing boards placed horizontally and crosswise one above the other. The boards in a grid, separated by washers, are held together by bolts; the openings in the grids placed near the entrance of the gas at the top are larger than those near the outlet at the bottom. The gas coming from a dust-catcher with a temperature of 80° C., with 2.6 to 3.0 grains of dust and 30 to 35 grains steam per cu. ft. leaves the washer at 20 to 25° C. retaining 0.6 to 0.9 grains dust and 9 grains steam per cu. ft. The precipitated dust is washed away by the water as a mud, and collects in the pan.

At Gary, Ind., eight Zschocke washers (14 ft. in diam. and 50 ft. high) treat 63 per cent. of the gas of a blast-furnace (88 ft. high, 21 ft. 6 in. bosh, 15 ft.

[1] *Iron Age*, 1909, LXXXIII, 1; *Iron Trade Rev.*, 1909, XLIV, 83.

[2] Editor, *Tonind. Z.*, 1899, XXIII, 996.

Nagel, *Iron Age*, July 14, 1906, p. 16.

Josse, *Zt. Verein deutsch. Ing.*, 1904, XLVIII[1], 913.

Baum, *Glück Auf*, 1904, XL, 457.

De Mocomble, *Rev. Mét.*, 1905, II, 247.

Reinhardt, *Tr. A. I. M. E.*, 1906, XXXVII, 678.

crucible), receiving it from a modified Mullen washer, and deliver it to eight Theisen washers (see below); of the remaining 37 per cent., 30 go direct to the stoves and seven to the boilers. Only the upper part of the Zschocke washer is filled with grids, the lower part contains umbrella-shaped baffle-plates; the gas enters from below.

Other forms are that of Sépulchre,[1] several described by Myjers,[2] and that of Kubierschky.[3]

B. Slow-moving Fresh-contact Washers. Of these the Bian[4] Washer is the leading type. It cleans blast-furnace-gas sufficiently for use in stoves and boilers, but not in gas-engines. An elevation and cross-section of the washer are shown in Figs. 824 and 825, and a longitudinal section of a plant in Fig. 826.

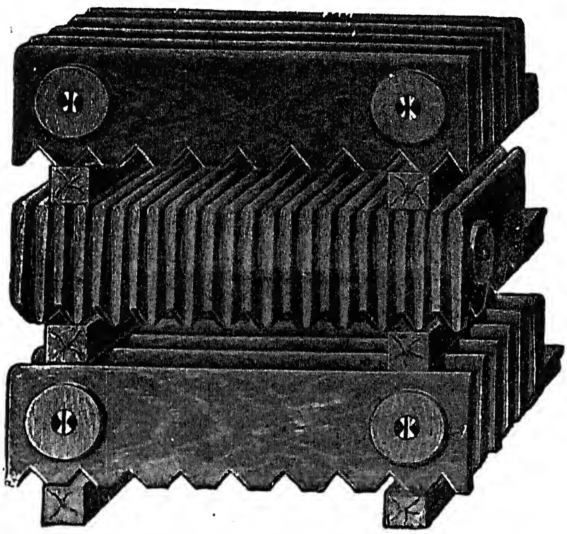
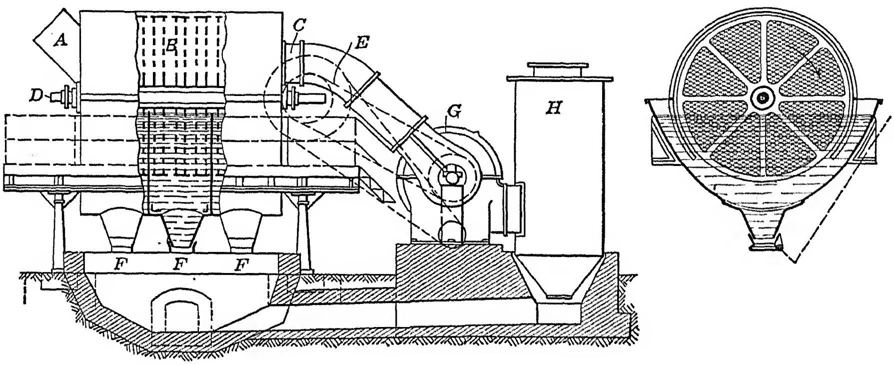


FIG. 823.—Zschocke scrubber.

The washer consists of a horizontal stationary cylindrical steel chamber *B*, 126 in. in diam. and 118 to 200 in. long, with closed ends and open bottom. The lower half of the vessel is immersed in a trough filled with water. The trough has division-walls to regulate the flow of water admitted near the gas-outlet at the right, and hoppers to collect the mud which is removed periodically through a gate-valve into trough *F*. Shaft *D* carries spiders covered with a wire screen having 3/8-in. holes. When it is rotated at the rate of 10 to 12 r.p.m. by means of a motor or transmitter, *E*, Fig. 824, the screens rise out of the water

[1] *Rev. Mét.*, 1908, v, 125.
[2] *Stahl u. Eisen*, 1906, XXVI, 28.
[3] *Op. cit.*, 1910, XXX, 1476.
[4] Bian, *Iron Age*, 1905, LXXVI, 669
Krull, *Oest. Zt. Berg. Hüttenw.*, 1905, LIII, 533; *Glück Auf.*, 1905, XLI, 1353.
Junge, *Iron Age*, 1906, LXXVIII, 602, 605.
Simmersbach, *Stahl u. Eisen*, 1906, XXVI 465.
[4] Bian, *J. I. and St. I.*, 1907, III, 210.

and carry in the meshes a certain amount of water. The gas entering the washer through pipe *A* at say 185° C., evaporates more or less of the water of the first few screens and becomes enriched in water-vapor, while at the same time its temperature is lowered and the dust thoroughly wetted. This goes on as the gas travels toward the middle of the apparatus when an equilibrium is reached; beyond this stage the process is reversed, the cold water admitted



Figs. 824 and 825.—Bian gas-washer.

near the gas-outlet *C* condenses water-vapor into globules, and these carry down with them the wetted dust. There is a sprinkling device (not shown) to free the screens from adhering dirt that is not washed off automatically when these dip into the water. *G* is a centrifugal hydraulic fan, and *H* a dry scrubber. The washer lowers the dust-content of a gas from 3–5 to 0.22 grains per cu. ft. Assuming the gas to have a temperature of 100 deg. C., the consumption of cool-

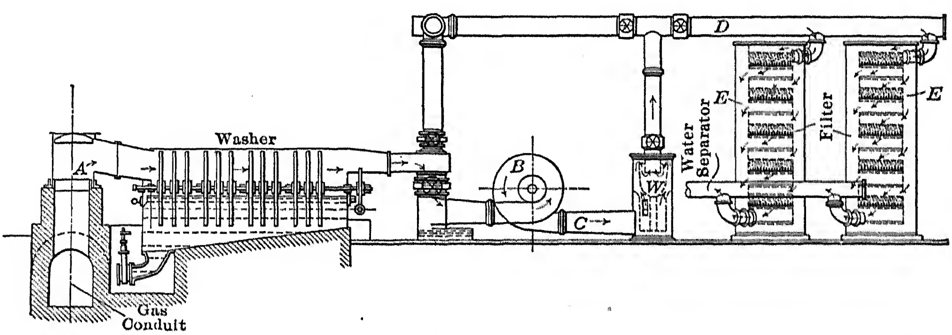
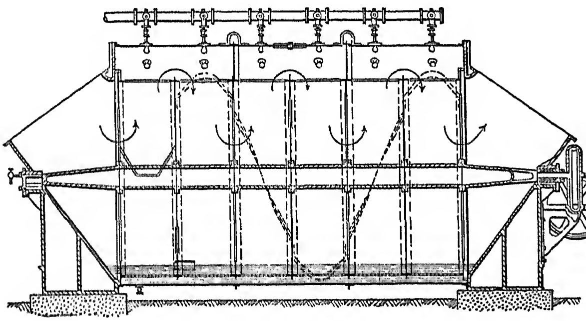


Fig. 826.—Bian gas-washing plant.

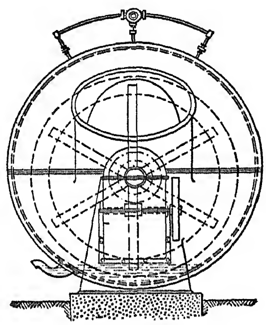
ing water in the washer is 7.5 gal. per 1,000 cu. ft. gas, and of cleaning-water in the fan 3.7 to 7.4 gal., making a total of 15 gal.; for gases hotter than 100° C., these figures are raised to 15 and 7.5 gal. respectively. The power required for both washer and fan is about 45 h.p. for a blast-furnace producing 100 tons pig iron in 24 hr.

Fig. 826 represents a Bian plant having a capacity of 706,000 cu. ft. gas per

hr. of which 90 per cent. is to be used in stoves and boilers, and the remaining 10 per cent. in gas-engines. The gas arriving in the conduit has a temperature of 140° C. and contains 3.5 to 5.2 grains dust per cu. ft. In the washer the temperature is reduced to 40° C. and the dust-content to 1.1 grains. The gas drawn through the washer by fan *B* is driven through the water-separator *W*, and leaving this is divided into two streams; the larger goes to stoves and boilers, the smaller travels in the opposite direction through pipe *D* and passes through filters *E* filled with slag-wool to be further cleansed and also dried; the purified gas retains 0.01 to 0.005 grain dust per cu. ft. The water-consumption is 22 gal. per 1,000 cu. ft. gas; the power required is 70 to 80 h.p. if the gas is to be cleaned for use in gas-engines, and 20 h.p. if in stoves and boilers.



Longitudinal Section.



End View.

FIGS. 827 and 828.—Sahlin gas-washer.

Another form of slow-moving washer, similar to the Bian is that of Sahlin,[1] shown in Figs. 827 and 828. In a stationary horizontal cylindrical chamber with gas inlet and outlet revolves slowly a perforated drum. The horseshoe shaped space between the chamber and drum is closed by plate diaphragms placed behind the spider-wheels; the bottom of the cylinder is filled with water to a depth 6 in. above the lowest point of the perforated drum; spray-water is admitted at the top; the mud is scraped toward the gas-inlet end by a spiral conveyor, bolted to the drum, where it is discharged through a goose-neck; the gas zig-zags through the chamber, and passing through a screened surface of greater area than in the Bian washer, the Sahlin washer ought to have a larger capacity.

c. Quick-revolving Atomizing Washers. The first atomizing apparatus put in operation was a hydraulic centrifugal fan.[2] Fig. 829 represents the Dinnen-

[1] *Cass. Mag.*, 1905, XXVIII, 442; *J. I. and St. I.*, 1905, I, 321.

[2] *Greiner*, *J. I. and St. I.*, 1901, I, 56.

Lürmann, *Stahl u. Eisen*, 1901, XXI, 448.

De Mocomble, *Rev. Mét.*, 1905, II, 232.

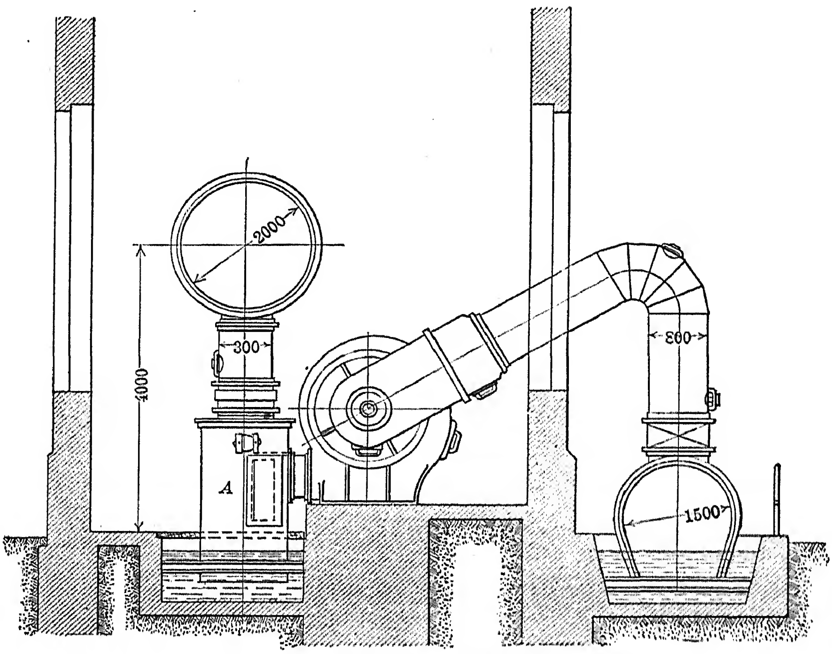
Reinhardt, *Tr. A. I. M. E.*, 1906, XXXVII, 682.

Junge, *Iron Age*, 1906, LXXVIII, 548; *Iron Trade Rev.*, 1908, XLII, 28.

Myjes, *Stahl u. Eisen*, 1906, XXVI, 33.

Coleman, *Proc. Am. Soc. Mech. Eng.*, 1910, p. 1837; *Stahl u. Eisen*, 1911, XXXI, 651 (Lackawanna Steel Co.).

dahl fan with water-seal gas-inlet pipe to the right, settling-tank A, and gas-outlet to the left. Its construction resembles that of the ordinary suction fan excepting that the vanes and bearings are made stronger. A water-inlet is provided at the suction-opening, and an arrangement is provided for atomizing the water. The atomized water becomes intimately mixed with the hot gas and is partly converted into steam; by centrifugal action the wetted dust and water particles are thrown toward the walls of the spiral casing and collect in sump A while the purified gas leaves by the top outlet. In order to cleanse the gas sufficiently for power-purposes, it is necessary to have as many as four



Dimensions = millimeters.

FIG. 829.—Dinnendahl hydraulic centrifugal fans.

fans arranged in series. In the first fan the water-spray is transformed in part into steam and thus wets the dust; in the second and third fans the steam is condensed, permitting the water-particles charged with dust to be readily collected. The fans have capacities of 15,000 to 70,000 cu. ft. gas per hr. and require from 40 to 110 h.p. The circumferential velocity of the impeller is as high as 186 ft. per sec. with a diameter of 3 ft. 9 in. to 5 ft. 10 in. For 1,000 cu. ft. of gas from 11.3 to 15.1 gal. water are required; the dust is reduced to 0.08 grains per cu. ft.

The Schiele-Brener fan[1] is similar to the Dinnendahl.

The Theisen Gas-washer.[2] This apparatus, shown in Figs. 830 to 833, is

[1] *Rev. Mét.*, 1905, II, 247.

[2] Theisen, *Stahl u. Eisen*, 1900, XX, 1037; 1902, XXII, 371; 1904, XXIV, 285, 1012; Simmers-

considered as the most efficient washer. It consists of a stationary conical casing, *A*, with gas-inlet *a* at one end and the outlet, *B*, at the other, and mud-outlet *G* beneath the gas-inlet. The inner wall is provided with an annular

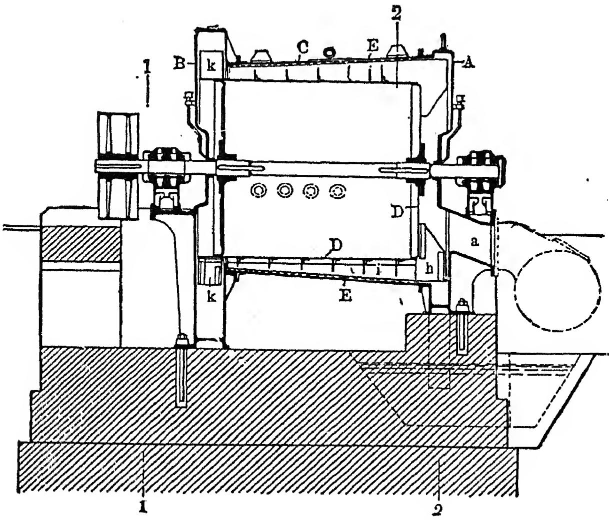


FIG. 830.—Theisen gas-washer cross-section.

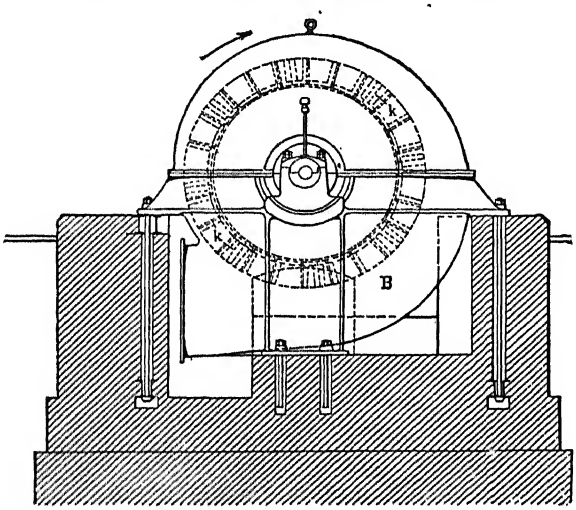


Fig. 831.—Theisen gas-washer, section on line 1–1, Fig. 830.

wire screen *E* causing the spray-water admitted through pipes *F* to bubble through the meshes and offer a large condensing surface. The casing contains a revolving horizontal drum, the surface of which carries a number of spiral

bach, *op. cit.*, 1899, XIX, 57; Lürmann, *op. cit.*, 1901, XXI, 437; Editor, *Iron Age*, Aug. 14, 1904, p. 4; Sahlin, *Cass. Mag.*, 1905, XXVIII, 448; Junge, *Iron Age.*, 1906, LXXVIII, 602, 605; *Iron Trade Rev.*, 1908, XLII, 28; Reinhardt, *Tr. A. I. M. E.*, 1906, XXXVII, 679; At Gary, Ind., *Iron Trade Rev.*, 1909, XLIV, 834; *Iron Age*, 1909, LXXXIII, 7.

vanes, *i*, which force the gas to travel in a spiral path while passing through the machine. The shaft of the drum carries at the ends suction fans *h* and *k*. In the diagram the drum is represented as being driven by belting, but ordinarily a motor is direct-connected to the drum-shaft which makes 850 r.p.m.

The principle of the machine is to force a film of water traveling from left

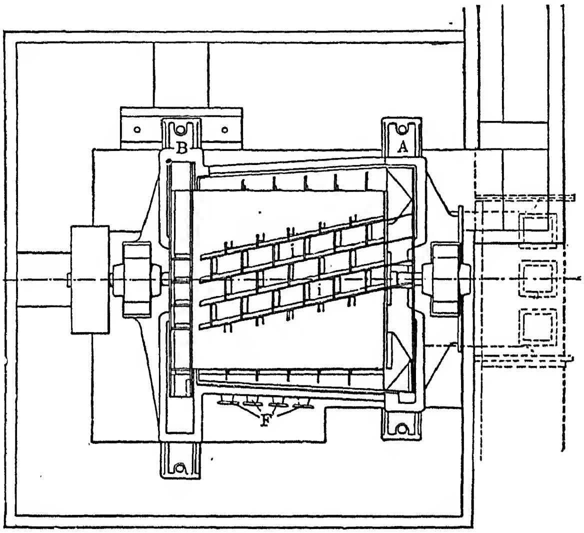


FIG. 832.—Theisen gas-washer, plan.

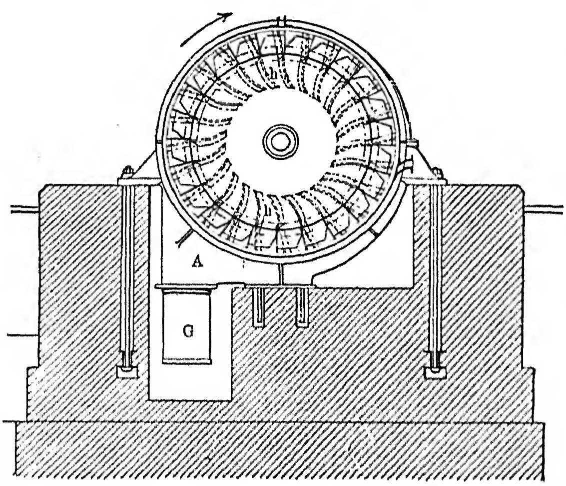


FIG. 833.—Theisen gas-washer, section on line 2–2, Fig. 830.

to right into frictional contact with the gas traveling from right to left, and thus to condense and absorb the impurities of the latter. The gas, entering at *a*, is sucked by fan, *h*, into the narrow space between the casing and drum, and made to travel through it in a spiral path until it is discharged by fan, *k*, into the delivery chamber, *B*, at a pressure of 2 to 4 in. water; at the same time the centrifugal action causes the water to be distributed over the surface of the screen.

Working with gas from the downcomer, the temperature is readily reduced from say 145 to 30° C. while the temperature of the wash-water, using 6 gal. per 1,000 cu. ft., is only slightly increased. Here, as in the previous apparatus, the hot gas at first vaporizes some of the water and drops the larger and heavier particles of dust. The water-vapor wets the finer particles, which are thrown against the film of water spread over the casing, absorbed and carried off through pipe *G* into a settling-tank. The clarified water is pumped through air-cooled radiators into an elevated supply-tank. It appears, however, to be preferable[1] to treat gas that has passed through a primary washer as is the case, *e.g.*, at Gary, Ind., where the blast-furnace gases first pass through Mullen and Zschocke washers before they arrive with a velocity of from 3 to 4 ft. per sec. at the Theisen. A Theisen washer reduces the dust from 1.4 – 1.7 to 0.08 –0.14 grains per cu. ft., consumes from 6 to 11 gal. water per 1,000 cu. ft. of gas, and is driven at a speed of from 300 to 450 r.p.m.; the sizes range from 21,000 to 1,555,000 cu. ft. per hr. and require from 50 to 150 effective h.p. At Gary, Ind., the cost of installation of a washer per 1,000 cu. ft. gas is $23; the water consumption is 9 gal., the h.p. 15, the operating expense 3.8 cents. Table 250[2] gives some data of four German iron blast-furnace plants.

TABLE 250.—RESULTS WITH THEISEN WASHER

	Hochdahl		Schalke	Horde		Rombach
	Apparatus I, hot, uncleaned gas	Apparatus II		Apparatus I, cool, cleaned gas	Apparatus II	
Dust, grains per 1,000 cu. ft.						
Before washing	2.6	2.6	1.3-1.7	1.1	1.0	0.87
After washing	0.017	0.008	0.008	....	0.004	0 008
Water, grains per 1,000 cu. ft.						
Before washing	7.8	10.4	15% vol.	13.0	15.8	18.3
After washing	3.1	2.2	12-20% vol.	1 5	1.3	13.9
Temperature of gas, deg. C.						
Before washing	144	158	144	46	45	43
After washing	30	37	30	33	28	36
Temperature of water, deg. C.						
Before washing	14	7	12	28	20	18
After washing	39	40	55	37	34	19
Cooling-water consumed.						
Cubic feet per hour	667	424	360	565	247	360
Gallons per 1,000 cu. ft.	8.22	7.48	7.48	7.78	7.93	8.45
Volume of gas per hour, cu. ft.	607,160	423,600	360,060	529,500	211,800	317,700

[1] Osann, *Stahl u. Eisen*, 1902, XXII, 153.
[2] Theisen, *Stahl u. Eisen*, 1904, XXIV, 288.
Junge, *Iron Age*, 1906, LXXVIII, 603.

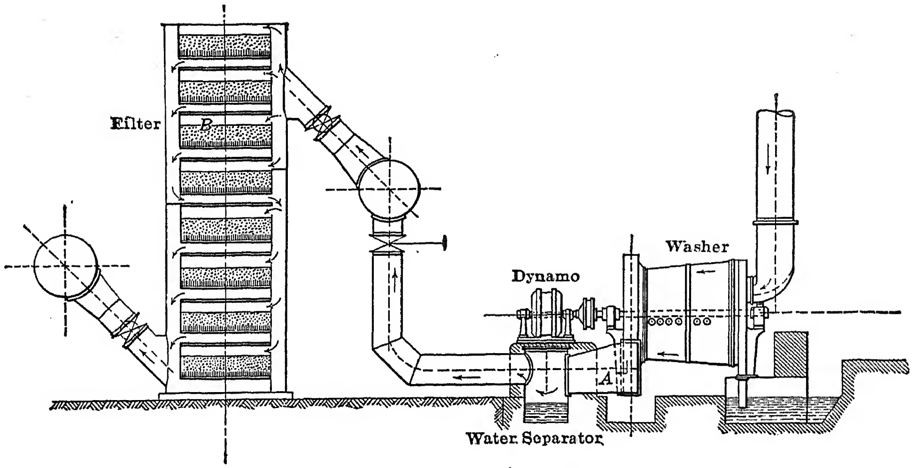


FIG. 834.—Theisen gas-washing plant.

TABLE 251.—COMPARISON OF GAS-WASHERS

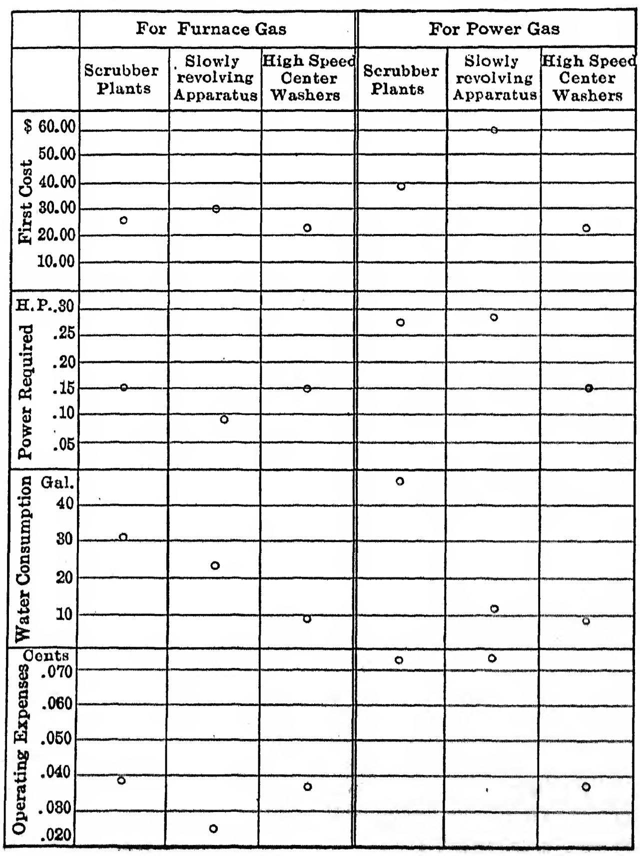


Fig. 834 is a Theisen[1] plant having a capacity of 847,200 cu. ft. gas per hr. Partly cleaned gas with 0.43 to 0.69 grains dust per cu. ft. and a temperature of 40° C. leaves the washer with 0.013 grain at 25° C., and is cleaned to 0.01 grain in the drying filter.

A comparison[1] of the work of the three types of gas-washers as given by Junge,[2] taking into consideration the first cost, power requirements, water consumption and operating expenses, based upon 1,000 cu. ft. gas, is shown in Table 251.

341. Neutralization and Utilization of Sulphurous Gases in General.—The sulphurous gases produced in metallurgical plants treating sulphides have harmful effects upon vegetation and human health. They contain both SO_2 and SO_3 which combining with H_2O form H_2SO_3 and H_2SO_4.

The deleterious effect upon plants[3] is due to direct contact of acid with leaves or needles which causes a derangement of water circulation and a loss of water. Leaves exposed to sulphurous gases resemble those that are decaying in the autumn; they are imperfectly developed at the start, become spotted and pale, then bleach and soon assume a dirty autumnal coloring and wither. In woods the effects are seen first in patches here and there, and these unite later on to form barrens. Evergreens show a behavior similar to that of deciduous trees. The withering effect is due mainly to the direct contact of SO_2 with the leaves and needles, to a smaller extent to that of SO_3; it can be valued by the percentage of S they contain in excess of similar plants in the same region not exposed to fumes.[4] The sensitiveness of plants varies with the species. Soil quickly converts H_2SO_3 into H_2SO_4 and neutralizes it, hence there is no direct action of these two acids upon the roots. Apart from the effect of the transposition of constituents of soil, no change occurs in composition.

In England the law[5] says that gases with over 1.5 grains per cu. ft. (=0.12 per cent. by vol.[6]) may not be discharged into the air; in Prussia they may not

[1] De Mocombe, *Rev. Mét.*, 1905, II, 252.
Myjes, *Stahl u. Eisen*, 1906, XXVI, 27.

[2] *Iron Age*, 1906, LXXVIII, 606.

[3] Lunge, G., "Sulphuric Acid and Alkali," Gurney-Jackson, London, 1913, I, 154; Haselof, E., and Lindau, G., "Die Beschädigung der Vegetation durch Rauch," Bornträger, Leipsic, 1903; Wieler, A., "Untersuchungen über die Einwickung der schwefligen Saeure auf die Pflanzen," Bornträger, Berlin, 1905; Wislicenius, H., "Sammlung von Abhandlungen über Abgase und Rauchschäden, Parey, Berlin, 1908 and foll.; "Rauch und Staub," Bagel, Düsseldorf, 1910 and foll.; Hamor, W. A., Bibliography, *Min. Ind.*, 1908, XVII, 901; Widsoe, *Eng. Min. J.*, 1904, LXXVIII, 515; Ebaugh, *J. Am. Chem. Soc.*, 1907, XXIX, 971; Frazer, Bibliography, *Tr. A. I. M. E.*, 1907, XXXVIII, 520; Harkins-Swain, *J. Am. Chem. Soc.*, 1907, XXIX, 970; 1908, XXX, 915, 928; Haywood, *J. Am. Chem. Soc.*, 1907, XXIX, 998; *Bull.* 89 and 113 (revised in 1910), *Bureau of Chemistry*, U. S. Dept. Agriculture; Austin, *Min. Sc. Press*, 1907, XCV, 649; Havard, *Tr. A. I. M. E.*, 1910, XLI, 631.

[4] Sorauer, Sammlung von Abhandlungen, etc., Heft. 7; *Min. Ind.*, 1911, XX, 490.
Hasenbäumer, *Rauch-Staub*, 1913, III, 109.

[5] Walker, *Eng. Min. J.*, 1907, LXXXVIII, 1134.

[6] Number grains per cubic foot × 2.29 = grams per cbm.

contain over 0.02 per cent. by vol. (=0.25 grain per cu. ft.); in the United States[1] the regulations vary in different states.

Needles and twigs of pines near Freiberg, Saxony,[2] from healthy unaffected trees showed 0.162 per cent. SO_3; in areas where injury was just observable to the eye, the percentage was 0.210 to 0.300; a higher grade of injury gave 0.3 to 0.5 per cent.; a still greater injury 0.5 per cent. and over. Haywood found that at Redding, Cal., 80 per cent. of the injured trees contained more SO_3 than the uninjured; at Ducktown, Tenn., the figure was 94 per cent.; at Anaconda, Mont., 92 per cent. Ebaugh's experiments with alfalfa, sugar beet and other plants with succulent leaves showed them to be less affected than conifers. The damage done by sulphurous gases is proportional to the amount of SO_2 present. Long-continued action of as little as $1SO_2$ in 1 million air proves hurtful; it is inversely proportional to the height at which it is discharged into the air, and depends upon the direction of the wind and the moisture of the air. The discharge of gases through high chimneys[3] lessens the amount of damage to vegetation to a degree less than is usually supposed. An interesting development in the disposal of smelter smoke is the Wislicenus smoke stack.[4] The sulphurous smoke issuing from the stacks of lead and copper smelteries remains undispersed for a long distance, the dust falls out gradually, but the gases come down not sufficiently diluted to make them harmless for vegetation. In order to dilute the gas, it is necessary that air be introduced at an early stage in such a manner as to divide the smoke column into strips surrounded by air. This is done by the Wislicenus stack, shown in Fig. 835, which consists briefly

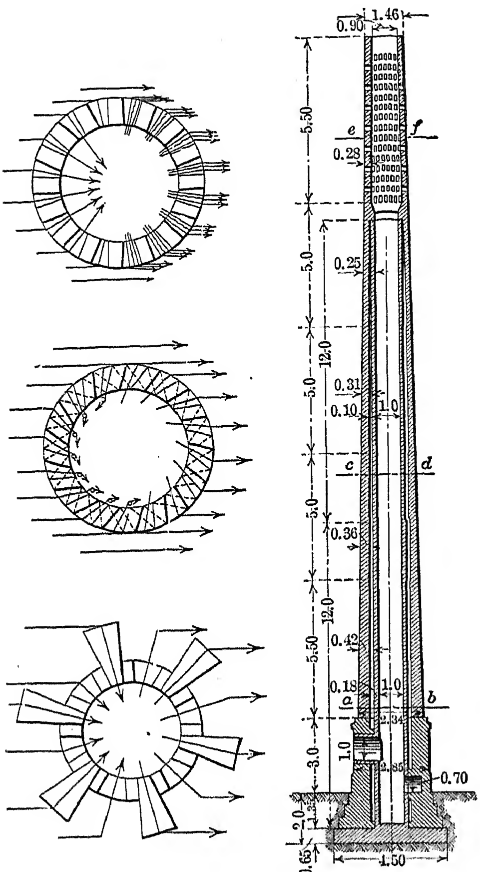


FIG. 835.—Wislicenus smoke stack.

[1] Baskerville, *Eng. Min. J.*, 1909, LXXXVII, 884.

[2] Schröder-Reuss, "Die Beshädigung der Vegetation durch Rauch und die Oberharzer Hüttenrauchschäden," Parey, Berlin, 1883.

[3] Isaachsen, *Verh. Verein. Beförd. Gewerbefleisses.*, 1902, LXXXI, 169.
Proposition of Lang, *Eng. Min. J.*, 1907, LXXXVIII, 1227.

[4] Austin, *Min. Meth.*, 1911, II, 269.
Hahn, *Eng. Min. J.*, 1911, XCII, 596.

in placing a lattice-like extension on top of the ordinary chimney. Air will pass through the lattices from whatever direction the wind may be blowing; it will come in contact with the ascending gases, divide them into bands and pass out with them at the opposite side. Thus, instead of a single column of smoke rising from the stack, strips, each surrounded by air, will pass off at the sides, the column having been broken up at the point of issue. The smoke of the single bands will readily diffuse into the enclosing air, and the smoke be dissipated so that, when it finally comes down, the sulphurous gas will be diluted sufficiently to be harmless.

Sulphurous gas is more harmful in the presence of light, in dry (not generally accepted) and warm air than under reverse conditions, and hence is more injurious by day than by night.

Contact with flue dust has no injurious effect upon conifers; any poison found in a plant is due to soluble salts having been formed in the soil and taken up by the plant, the effect of which is only very slight. Plants with succulent leaves are affected by flue dust. Needles and twigs of pine trees at Freiberg according to Schröder-Schertel[1] contained in 1,000 parts: Pb 5 to 50, As 3.3 to 14.3, SO_2 62 to 120 parts.

Human health[2] is unfavorably affected by sulphurous acid. The latter irritates the mucus membrane of nose, mouth and throat causing increase of flow of saliva; working any length of time in sulphurous fumes causes violent coughing, spitting of blood, vomiting, loss of appetite, irregularity of action of the bowels, and inflammation of the eyes. Air with 0.012 parts SO_2 per 1,000 does not affect the average person,[3] air with 0.015 parts does; workmen accustomed to fumes are little affected by air with 0.037 parts in 1,000; air with 0.7 per cent. SO_2 is considered the limit of what man can bear. Sulphuric acid, like all mineral acids, has a corrosive action, destroying tissue and coagulating albumen.

The methods of handling sulphurous gases vary with the percentage of SO_2. They may be divided: (1) methods suited for gases from heaps, stalls and reverberatory furnaces, which contain less than 2 per cent. by vol. of SO_2 and are contaminated with fuel gases; and (2) methods suited for gases from muffle-furnaces and continuous kilns with 4 per cent. by vol. of SO_2 and over, and are free from fuel gases. Gases containing less than 2 per cent. SO_2 by vol. have no industrial value; the SO_2 has to be neutralized unless the surroundings make it possible to pass off the gas into the open air. The SO_2 from gases with 4 per cent. by vol. and over may be converted into sulphurous anhydride, sulphuric anhydride, or sulphuric acid. A number of processes have been suggested to utilize it in other ways.

[1] *Freib. Jahrb.*, 1884, p. 93.

[2] Füller, M., Meissner, C., and Saeger, O., "Hygiene der Berg, Tunnel und Hüttenarbeiter," Fischer, Jena, 1895, p. 448.

Bauer, S., "Gesundheitsgefährliche Industrien," Fischer, Jena, 1903.

Rambousek, J., "Gewerbliche Vergiftungen," Veit & Co., Leipsic, 1911.

[3] Lehmann, *Zt. angew. Chem.*, 1893, p. 612.

342. Gases with 2 Per Cent. by Volume of SO_2 and Under.[1]—Condensation by water has been repeatedly tried, but has not proved effective. Although the solubility[2] of SO_2 in water at ordinary temperature is great, as shown in Table 251, the acid in the smoke is so dilute that the SO_2 is imperfectly collected in the

TABLE 251.—SOLUBILITY OF SULPHUR DIOXIDE IN WATER

H_2O, degrees C.	20	30	40	50	60	70	80	90	100
SO_2, per cent. dissolved	8 6	7.4	6.1	4 9	3.7	2.6	1.7	0.9	0.0

usual condensation towers. Although mechanical washers, *e.g.*, of bronze, may remedy this drawback, there remains the difficulty of disposing of the acidulated water. Carpenter's freezing-process, already referred to (p. 858), may prove a solution although its economic success is doubted.

Neutralization with limestone was introduced by Winkler[3] at the ultramarine works of Schneeberg, Saxony. Although the process works satisfactorily, it is essential that the gases be cool (with 38° C. much $CaSO_4$ is produced, forming a protective coat), that a large quantity of water be fed, and that the draft be strong so as to overcome the resistance of the long path which the gases have to travel in order to become completely neutralized. The method has not made any headway.

Neutralization with milk of lime,[4] on the other hand, is in operation at a number of plants; the SO_2 forms $CaSO_3$, and this upon exposure to air gradually changes into $CaSO_4$, which, after having been dried and ground, is marketable as a fertilizer and as a raw material for the manufacture of plaster of Paris. However, the price of natural gypsum in this country is too low to promise any profit in the preparation of the salt.

At the Hohenlohe zinc works in Silesia,[5] the gases from blende-roasting reverberatory furnaces containing SO_2 1 per cent., CO_2 3.5 per cent., O 16 per cent., N 79.5, are treated as follows: They are made to ascend in a series of towers 20 ft. high connected with a chimney 328 ft. high. The towers are sprayed at the top with a large excess of milk of lime, and the precipitate and liquor are drawn off at the bottom into sumps where the former settles while the milky liquor is pumped again on to the towers. The precipitate contains about 40 per cent. CaO, 37.5 per cent. SO_2, 4 per cent. CO_2, 2.5 per cent. SO_3, 3.4 per cent. $(FeAl)_2O_3$, residue 8 per cent. From 75 to 80 per cent. of the SO_2 of the gases is thus recovered.

[1] Schütz, E., "Die Darstellung von Bi-Sulphiten und Sulphiten," Knapp, Halle, 1912.
[2] Harpf, A., "Flüssiges Schwefeldioxyd," Encke, Stuttgart, 1900, p. 21.
[3] *Freib. Jahrb.*, 1880, p. 68.
[4] Korten, "Mechanische Kalkmilchbereitung," *Glück Auf.* 1912, XLVIII, 1042.
[5] Schmieder, *Oest. Jahrb.*, 1889, XXXVII, 397.
Ingalls, W. R., "Metallurgy of Zinc and Cadmium," Engineering and Mining Journal, 1903, p. 164.

Other absorbents have been suggested; magnesium hydrate, aluminum hydrate and alumina by Precht (1881), zinc carbonate by Schnabel (1882) with the object of heating the salts formed and thus concentrating the SO_2 and making it available for acid and other processes. The use of these compounds has not gone beyond the experimental stage.

At Flône, Belgium,[1] dilute gases from blende-roasting reverberatory furnaces are conducted into old dumps of alum shale in order to dissolve the alumina as sulphate. At Salt Lake City, Utah, sulphurous gases have been conducted into slag-heaps (Sörensen-Westley process).[2] The use of dry zinc oxide or calcium oxide for neutralizing sulphurous and sulphuric acid in furnace gases by Sprague has been discussed on page 849.

343. **Gases with 4 Per Cent. SO_2 and Over.**—The principal methods of utilizing gases of this character aim to convert them into sulphuric acid (H_2SO_4) by the chamber process, into sulphuric anhydride (SO_3) by the contact process, or into liquid sulphurous anhydride (SO_2) by the Schröder-Hanisch press, all of which are treated in G. Lunge's "Sulphuric Acid and Alkali," Gurney & Jackson, London, 1913. The absorption tower[3] of the last process has recently been improved by W. Borchers.[4] Other methods for utilizing concentrated SO_2 are the production of sodium sulphate, of sulphur, and the solution of metal.

Sodium sulphate is produced by the process of Hargreaves and Robinson[5] in which SO_2 in the presence of air and steam is made to act upon NaCl between 400 and 500° C., viz., $2NaCl+SO_2+O+H_2O=Na_2SO_4+2HCl$.

It has been proposed to produce S from SO_2 in various ways.[6] Direct reduction by red-hot carbon, $SO_2+C=S+CO_2$, appears to work only with concentrated gas because the free air in dilute gases oxidizes the C, and because the reduction of dilute SO_2 is imperfect. Dilute gas has to be enriched as was indicated above. Recent experiments by De Lay and Carson[7] demonstrate again that the reduction of SO_2 by C is easily effected, and that lamp-black from the imperfect combustion of crude oil works satisfactorily.

The so-called Thiogen process of Young[8] aims to reduce SO_2 by means of C_xH_y. It consists of the following steps: (1) Action of dry SO_2 upon warm CaS, as expressed by $2CaS+3SO_2=2CaSO_2+3S$; (2) recovery of sulphur from

[1] Firket, *Ann. Mines de Belgique.*, 1901, VI, 235.
Ingalls, *op. cit.*, p. 165.

[2] Westley, *Eng. Min. J.*, 1910, XC, 1164; *J. Ind. Eng. Chem.*, 1912, IV, 725; *Min. Eng. World*, 1912, XXXVII, 709.

[3] Eilers, *Tr. A. I. M. E.*, 1891, XX, 336.
Harpf, A., "Flüssiges Schwefeldioxyd," Enke, Stuttgart, 1900; also Ingalls, W. R., "Metallurgy of Zinc and Cadmium," Engineering and Mining Journal, New York, 1903, p. 166.

[4] Metallurgie, 1909, VI, 316.

[5] Lunge, *op. cit.*, Vol. II[2].

[6] Nagel, *Rauch-Staub*, 1912, III, 68.

[7] *Min. Sc. Press*, 1908, XCVII, 401.

[8] *Min. Sc. Press.*, 1911, CIII, 375, 386; 1912, CIV, 491; *Eng. Min. J.*, 1912, XCIII, 873; 1913, XCV, 369; *Met. Chem. Eng.*, 1912, X, 710.

the intermediary product by sublimation; (3) treatment of the latter with a hydrocarbon at an elevated temperature whereupon the reaction $2CaSO_3 + CH_2 = 2CaS + 2CO_2 + H_2O$ takes place with the regeneration of the original reagent. It appears that the presence of pyrite is favorable, being decomposed according to $3 FeS_2 + 2SO_2 = Fe_3O_4 + 8S$, although very slowly. Instead of the imaginary CH_2 given in the formula, there would be used C_2H_4, CH_4, petroleum vapor, etc.

Carpenter[1] proposes to cool the reduced sulphur vapor and to collect the solidified flour of sulphur by filtering.

Another method[2] of obtaining S is to produce H_2S by conducting SO_2 and water-vapor through a tower filled with glowing coal ($SO_2 + H_2O + 3C = H_2S + 3CO - \text{cal.}$; $2SO_2 + 2H_2O + 3C = 2H_2S + 3CO_2 + \text{cal.}$; $5SO_2 + 5H_2O + 9C = 5H_2S + 6CO_2 + 3CO + \text{cal.}$) enough air passing through the coal to keep the generators hot, but not to oxidize the H_2S, and then to produce S either by having the H_2S act upon SO_2—$2H_2S + SO_2 = 2H_2O + 3S$—or by conducting the H_2S through a Claus-Chance kiln:[3] $H_2S + O$ (from Fe_2O_3) $= H_2O + S$.

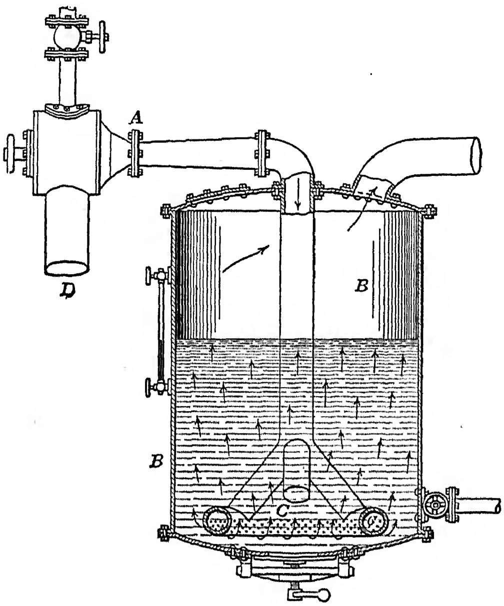


FIG. 836.—Rössler converter.

A third method is the one suggested by Kosmann[4] who proposes to reduce $CaSO_4$ by C to CaS, to change this into CaH_2S_2 by treatment with H_2O, and then to pass SO_2 through it whereupon the following reaction will take place: $5SO_2 + 2CaH_2S_2 = 7S + 2CaSO_4 + 2H_2O$; the S is to be extracted from the mixture by superheated steam, and the residue used over again.

The solution of metals, such as finely divided Fe, Cu or Zn, has not been an economic success. Rössler[5] used the apparatus shown in Fig. 836 for condens-

[1] *Proc. Colo. Sc. Soc.*, 1908, IX, 69.

[2] Buddëus, *V. Internat. Kongress für Angew. Chemie.*, 1903, II, 116; *Zt. Electrochemie*, 1903, IX, 858.

[3] Lunge, *op. cit.*

[4] *Glück Auf*, May 2, 1894; *Berg Hüttenm. Z.*, 1894, LIII, 192 (see also 149); *Oest. Zt. Berg. Hüttenw.*, 1895, XLIII, 699.

[5] *Dingler, Polyt. J.*, 1881, CCXLII, 278.
Fischer Jahresberichte, 1881, XXVII, 184.
Wendt, *Tr. A. I. M. E.*, 1883–84, XII, 274.
Stetefeldt, *op. cit.*, 1892–93, XXI, 74.

ing the acid fumes arising from a cast-iron kettle in which doré silver is parted by means of concentrated sulphuric acid, and for preparing cupric sulphate from finely divided copper. In the figure, *B* is the lead-lined tank 8 to 9 ft. high and 5 ft. in diam., *A*, the lead-lined pipe 6 in. in diam.; *C*, perforated ring; *D*, the pipe through which gases are drawn by means of a Koerting ejector, Fig. 646. The tank is charged with a solution of $CuSO_4$ and precipitated metallic Cu which may carry some Ag and Au. With the hot sulphurous gases from the parting kettle a large amount of air is drawn into the tank and the solution thoroughly stirred. The $CuSO_4$ acts as an oxygen-carrier for the oxidation of SO_2 to SO_3, and the H_2SO_4 formed readily dissolves the finely-divided Cu in the presence of air forming $CuSO_4$. When the solution has reached a strength of 35° Bé., it is replaced by a fresh charge. The extension of this process to the utilization of ordinary acid smoke has not proved successful.[1]

344. Neutralization and Utilization of Chlorine and Hydrochloric Acid.— The gases formed in chloridizing roasting carry Cl, HCl, and volatile metallic chlorides, besides some SO_2 and SO_3. They have an injurious effect upon vegetation. Carrying them off through high chimneys is less efficacious than with sulphurous gases, as they are not diffused in the air, but return to the earth in compact clouds.

In the chloridizing roasting of copper ores (Longmaid-Henderson process),[2] the condensation of gases is of great importance, as the condensed acid (tower-liquor) is used as a solvent, and as volatilized copper chloride is recovered. The condensation is effected in towers which must have large dimensions if the roast is carried on in reverberatory furnaces. Large towers (10 ft. sq. and 50 ft. high) are built of acid-proof brick laid in tar, or of wood lined with lead,[3] or small towers (36 in. diam.) of sewer pipe. Large towers are packed with quartz or acid-proof brick, small towers with coke. Two towers may be coupled in series. With all towers, the gas enters near the bottom; spray-water is admitted on the top.

[1] Friese, *Chemische Industrie*, 1895, XVIII, 137.
[2] Hofman, "Metallurgy of Copper."
[3] Bräuning, *Zt. Berg. Hüttenw. Sal. Wesen i. Pr.*, 1870, XXV, 157.

CHAPTER XII

METALLURGICAL PRODUCTS

345. **In General.**—Metallurgical products are classed in various ways: (1) According to the necessity of further treatment as finished, unfinished and refuse products; (2) according to the process by means of which they have been formed as roasting, smelting, sublimation, distillation, lixiviation, etc., products; and (3) according to the leading metal they contain such as iron, copper, lead, zinc, tin, etc., products. As these different points of view furnish no clear lines of demarcation, it seems more convenient and simple to discuss them briefly under (4) the heads of solid, liquid and gaseous.

346. **Solid Products.**—The leading solid products are metals, alloys, mattes, speises, slags, flue dust, wall accretions, hearth accretions and furnace refuse. Metals (§4–24), alloys (§25–58), slags (§190–213), flue dust (§329 and foll.) have already been treated.

347. **Matte.**—This is a heterogeneous mixture of metallic sulphides, produced in smelting sulphide ores, which often contains some metal and metallic oxide. Chemical analysis shows a great diversity in composition. The leading components are S, Cu, Fe, Pb, Ni; of secondary importance are Zn, Co, Mn, Bi, Ag, Au, Pt, Ca, Ba; besides S there is frequently present some As and Sb. As to the constitution of matte, the freezing-point curves so far investigated show in solidified matte that there are present chemical compounds, eutectic mixtures and solid solutions. A number of the sulphides give up some of their sulphur when they are heated well above their melting-points; with the exception of Au the simple chemical compounds are stable at their melting-points. Complex chemical compounds are not common; the leading ones are: $(Ni_3S_2+Ni)FeS$; $(Cu_2S)_2FeS$; $(Cu_2S)_3(FeS)_2$; $(Cu_2S)_2(FeS)_5$. Eutectics are formed by $FeS–Cu_2S$; $PbS–Cu_2S$; $FeS–PbS$; $Fe_7S_8–PbS$; $FeS–ZnS$; $PbS–Ag_2S$; $PbS–ZnS$; $Ag_2S–ZnS$; $Ni_3S_2–Cu_2S$; $Cu–Cu_2S$; $Fe–FeS$; $Ni–Ni_3S_2$. Solid solutions are formed by $Ni_3S_2–FeS$; $Cu_2S–FeS$; $Cu_2S–Ag_2S$; $FeS–Fe$; $Cu_2S–Cu$; $PbS–Pb$. The presence of Fe_3O_4 has been noticed in some copper mattes.[1] Mattes produced from ores containing some As or Sb retain small amounts of these elements; whether they form sulpharsenides or antimonides[2] or are present in some other form is not known.

Matte is brittle and the fracture ranges from coarse-grained through fine-grained to conchoidal; its color is bronze-like, often bluish, again dark to light gray; the luster is bright; the specific gravity[3] varies from 4.7 to 5.55; Zn and

[1] Keller, *Eng. Min. J.*, 1895, LX, 465; *Tr. A. I. M. E.*, 1893, XXII, 580; Neill, *ibid*, 675. Walter, *Eng. Min. J.*, 1913, XCV, 213.

[2] Jaeger, *Eighth Internat. Congr. Appl. Chem.*, 1912, II, 139.

[3] Table in H. Lang, "Matte Smelting," Scientific Publishing Co., New York, 1896, p. 36.

Mn make it light; Pb, other metals, Ba, As and Sb make it heavy. Some mattes disintegrate quickly when exposed to air at ordinary temperature, others are more stable. Hot matte in contact with water explodes with great violence.[1]

348. **Speise.**—This also is a heterogeneous mixture composed of metallic arsenides (and antimonides) produced in the smelting of arsenical (and antimonical) ores; arsenides prevail over antimonides. Chemical analysis, also here, shows a great diversity of composition. The leading components are As(Sb), Fe, Ni, Co, Cu; of secondary importance are Pb, Ag, Au, Zn, Bi. The freezing-point curves so far investigated show that chemical compounds are formed by Fe–As, Ni–As, Co–As, Cu–As; eutectic mixtures by Fe–As, Ni–As, Co–As, Cu–As, Pb–As, Ag–As(?) and Pt–As(?); solid solutions by Ni–As and Zn–As; no affinity whatever exists between Bi–As. A speise appears to contain more chemical compounds than a matte; this may be presupposed if one may judge by the greater difficulties encountered in roasting a speise than a matte. Finely divided metal and metallic oxide are usually absent. Speise is brittle; some varieties have a granular fracture, and others clearly developed cleavage planes; the color usually is a yellowish to grayish white; the luster is bright; the specific gravity is greater than that of matte. Most speise remains unattacked for a long time when exposed to the air at ordinary temperature. Speise, just like matte, is an unfinished product, and is treated for its valuable constituents. The general behavior of arsenides and antimonides is discussed in §§66 and 67.

349. **Wall Accretions.**—These are finely to coarsely crystalline mixtures found in the upper cooler parts of a shaft furnace. They may consist of parts of the original charge, or may be the result of chemical changes in the charge. They may be composed of metal and metallic compounds which have been vaporized in the lower hotter zone and condensed in the upper cooler portion of a furnace where they gather either in their original forms or after they have undergone some chemical changes. The accretions either simply adhere to the furnace walls in which case they may be easily barred off, or they attack the furnace lining and this frequently to such a degree that the furnace has to be relined. In the iron blast-furnace are found pure C and cadmia (Zn+ZnO); in non-ferrous furnaces are found volatile sulphides, arsenides, as well as metals and their secondary products. The composition of a wall accretion depends upon the character of the charge and the running of the furnace. Accretions are usually treated in the same furnace in which they have been formed; cadmia forms the leading exception to this general statement.

350. **Hearth Accretions (Sows).**—These form an unwelcome mixture of parts of the charge and of the products of the process which melts at a temperature higher than the one obtaining in the furnace, and which sinking through the melted mass at the tuyère-level collects on the bottom of the hearth. Its formation is caused by a faulty mixture, by a wrong distribution of the charge,

[1] Hutchison, *Eng. Min. J.*, 1909, LXXXVII, 172.
Smith, *op. cit.*, 1911, XCI, 754.

or by accidents in the normal working, etc. In the iron blast-furnace the sow consists mainly of non-carburized iron; in non-ferrous furnaces it is usually a mixture of metallic iron, other metals, speise, matte, slag, and fuel. A sow may be a waste or intermediary product depending upon the values locked up in it. In any case the metallurgical treatment is a tedious, expensive piece of work.

351. **Furnace Cleanings and Refuse.**—The mixture raked out of a furnace after the fluid parts have been tapped usually goes by this appelation. It is returned to the furnace after this has been blown in again.

352. **Liquid Products.**—The solutions obtained in wet or some electrolytic processes contain salts which may be recovered by crystallization as is the case with Fe, Cu, Zn, Ni, Co, etc.; or they may serve as binders in briquetting finely divided ores (*e.g.*, $FeSO_4$+aq) and as a sulphatizing addition in roasting; or they may be run to waste either directly or after having been neutralized to make them harmless.

353. **Gaseous Products.**—The composition of gases formed in pyrometallurgical operations varies greatly with the processes. Thus, the tunnel-head gases from the iron blast-furnace, practically free from O, contain more CO and CO_2 and thus form an important gaseous fuel, while the gases from non-ferrous blast-furnaces are waste products, as they contain more CO_2 than CO. The gases from the copper blast-furnaces at Mansfeld, Germany, form the leading exception. The gases from reverberatory furnaces contain a large amount of unconsumed air; they are waste products and pass off into the open unless they are charged with noxious gases which have to be neutralized and may be utilized (§341). Gases from muffle-furnaces may be waste products or not depending upon the character of the ore that is being treated. Most furnace gases are made to pass through some sort of dust- or fume-collecting apparatus in order either to recover the dust or fume, or to purify the gas. In many cases both recovery and purification have to be carried on at the same time.

CHAPTER XIII

ECONOMIC CONSIDERATIONS

354. **The Corporation and Organization of Work.**[1]—In our days it is the exception that a metallurgical plant is owned and worked by an individual or by a partnership consisting of a few men. The amounts of money usually involved are too large; they call for a capital as well as a limited liability, and this requires the formation of a legalized business combination, the corporation. The capital necessary for metallurgical work has to cover the money required for establishing and for running the plant.

The capital of establishment does not represent simply the original plant investment, but includes also the money expended for starting and financing the corporation. Should the works be closed down, their market- or cash-value is only a small part of the money originally expended. For this reason a sinking fund is required, besides the usual interest charge (6 per cent.), which in a given number of years (10±) shall be allowed to wind up affairs without losing the money originally spent.

In a metallurgical plant this amortization is of special importance on account of the short lives of many furnaces, and the progress that is being continually made in mechanical appliances and chemical processes causing older apparatus not yet worn out to be replaced by newer ones, and modes of operating to be changed. The amount of money which must be put aside annually, at a certain rate of interest regularly compounded, may be figured according to Dilworth[2] by the formula $X=S\frac{r}{(1+2)^n-1}$, in which S = total amount to be retired, r = interest rate, n = life of sinking fund or amortization period.

The cost values of different plants vary with the character of the process and the magnitude of the unit. In Table 252 are assembled a few general data.[3]

The working or quick capital, sometimes termed liquid assets, represents the money necessary to operate the plant. It is usually larger than the capital of establishment, as funds are necessary for purchasing of ore, flux, fuel and other materials. It also includes the value of the intermediary products which have been paid for when bought as ore, but which are still in process of extraction;

[1] Conygnton, Th., "A Manual of Corporate Organization." Ronald Press, New York, 1908. Same author, "A Manual of Corporate Management," Ronald Press, New York, 1909.

[2] *Tr. A. I. M. E.*, 1910, XLI, 533; see also Channing, *Eng. Min. J.*, 1910, LXXXIX, 213.
Haas, *op. cit.*, 1910, LXXXIX, 403.
Smith, *op. cit.*, 1910, XC, 812.

[3] See also Ingalls, *Eng. Min. J.*, 1910, XC, 14.

the finished products which are in transit or have not been sold, or for which, even when sold, the usual thirty days' time for payment has to be awaited, and lastly for the payment of wages and salaries.

TABLE 252.—COSTS OF SOME METALLURGICAL PLANTS

Character of plant	Capacity in 24 hours	Cost
Iron blast-furnace	300 tons of pig iron	$ 650,000
Acid Bessemer with 4 cupolas and hot metal reservoir.	2,000 tons of steel	900,000
Acid open hearth, ten 50-ton furnaces	1,000 tons of steel	1,500,000
Basic open hearth, ten 50-ton furnaces	1,000 tons of steel	1,650,000
Rolling mill	Starting with ingots 20 in. square, weighing about 5,000 lb., consisting of 36-in blooming mill and 28-in. structural mill.	1,250,000 to 1,500,000
Copper smelting and converting	Partial pyritic smelting of 1,000 tons of ore to 100 tons of 45 per cent. matte.	1,250,000
Lead smelting	500 tons of mixed lead ore	250,000
Parkes desilverizing	100 tons of lead bullion	250,000
Moebius electrolytic parting	30,000 oz. doré silver	20,000
Electrolytic copper refining, multiple process	100 tons of copper, casting anodes and refining cathodes.	500,000
Zinc smelting	100 tons of blende, not making sulphuric acid.	375,000

The amount of funds necessary for ores will vary greatly if mine and metallurgical plant belong to the same company and are so situated that regular deliveries can be counted upon, or if the reverse is the case. In the milling of precious-metal ores and the smelting of many copper ores of the United States there is a close proximity of mine and reduction works; with lead and iron ores this is not usual. A mine-plant, as *e.g.*, the smeltery of the Anaconda Copper Mining Co., will not carry in stock as large an amount of ore per ton of product as will a silver-lead custom-plant in Colorado or Utah which receives its ore from long distances, and has to have on hand a stock for about four weeks. A lead plant will carry relatively less ore than an iron smelting plant in the Central Western states treating ore from Lake Superior; the latter must stock up with ore to last during the closed season of navigation, that is, for 5 or 6 months. This is not the case in Alabama where ore, flux and fuel occur in close proximity. The working capital necessary for a plant will vary therefore not only with the kind and quantity of ore treated, but also with the location of the works. Conservative companies lay by funds in times of prosperity in order to have abundant working capital for unfavorable periods. If their affairs have to be wound up, nearly the full value of the working capital can be realized.

The corporation is managed[1] and controlled by the board of the directors which is elected by the stockholders. The necessary officers of the board, are the president, secretary and treasurer who are chosen by the board of directors. The president is the executive head of the board; the secretary has charge of the corporate records, and the treasurer of the corporate funds. These three offices may be held by one, two or three persons depending upon the laws of the state and the by-laws of the corporation. The board which meets at given periods of time, or the president under its authority or the authority of the by-laws appoints the general manager of the works to take general charge of the technical and financial affairs of the plant; he has his headquarters at the plant, and is responsible to the board alone.

The work of a metallurgical establishment is conveniently assembled under the two divisions, the operating and the administrating departments.

The *operating department* is in the charge of a superintendent. In a small plant the offices of general manager and superintendent may be combined in one person. The superintendent has to do the planning and supervising of everything pertaining to the technical workings of the plant. In small works the superintendent is usually also the metallurgist who knows the theory and practice of the processes that are employed. The metallurgist will have foremen to look after the different kinds of work in the plant, and to take charge of the skilled (inside) and unskilled (outside) labor; he will have a chemist, or assayer, to make the routine chemical determinations and carry on some investigations when they are necessary, and lastly he will have a mechanical division which will include mechanics and perhaps the constructing engineer and draftsman.

In larger plants the superintendent will have the technical supervision of the several departments, each of which will be in direct charge of an assistant superintendent.

The *administrative department* includes the divisions of buying, selling, freighting and accounting. The materials bought are of two classes: ore, flux and fuel, form one, and stores, chemicals and all sorts of odds and ends the other. The purchase of ore, flux and fuel, which involves a judicious outlay of a considerable amount of money and thereby governs to a large extent the economic success of the plant, will be in the hands of the general manager or president. The buying of stores, chemicals, etc., may be looked after by the chief clerk or the superintendent.

In the purchase of ores the leading considerations are: (1) Kind and quantity of metal as well as physical and chemical form in which it is present; (2) kind, quantity, physical and chemical forms of associated metal; (3) character of gangue; and (4) size of ore.

With an iron ore the metal is always an oxide, but it will make a difference whether the mineral is magnetite, hematite, limonite or siderite. The percentage of iron is, of course, a leading consideration; the ore may contain too much P to come within the bessemer limit; the presence or absence of S, Ti, Mn and

[1] Lang, "Organization of Smelting Enterprises," *Min. Sc. Press* 1913, CVI, 585, 622.

other constituents will have an important influence upon its value. The gangue may be siliceous, argillaceous or calcareous; the ore may be hard and in lumps, or soft and in a very fine state of division.

Ores are purchased according to schedules based upon the actual cost of treatment, the desirability of the ore for a given process, and other special considerations. In the purchase of iron, lead and zinc ores, (less so with copper ores) schedules are made up annually which govern in a general way the prices that have to be paid. With fuel, the price will be governed in part by the requirement it has to fulfil to meet special cases. Thus an iron-blast furnace man will have to look closely into the S- and P-content of his coke, while the lead and copper smelter will confine his attention to strength, porosity, fixed carbon, volatile matte, percentage of ash and its acid or basic character. Fluxes are received and sampled like ores, only in a less exacting manner.

The *selling of products* is usually attended to by the president through a broker, commission house or special selling agency which is in direct communication with the market centers.

A special knowledge of freight-tariffs is of vital importance to obtain the supply of raw material and delivery of product at the lowest rate possible. Thus, *e.g.*, in regard to certain classes of low-grade ores which under normal conditions could not be mined at a profit, freight rates and smelting charges may have to be adjusted to leave the miner a profit, and thus encourage him to continue extracting ore with the hope that the grade will improve and then give him a greater return. Delicate adjustments of this character are often in the hands of special officers.

The *accounting department*,[1] in charge of a chief clerk, keeps and distributes costs which form part of the general statement of profit and loss. It renders bills for sales of product and collects amounts due and handles the funds entrusted. Another duty of this department is the presentation of the financial status of the company as embodied in the balance sheet. The whole forms a complete record of the business transaction of the metallurgical plant, prepared in such a way as to permit a detailed analysis of each single operation or transaction.

355. Cost Accounts.[2]—The purposes of cost accounts are to determine accurately the cost of ore-treatment in order to fix intelligently the treatment-charge, to furnish the means of judging of the efficiency of the several steps in a process and of the entire process, and to supply the data for analysis of the management of the plant as a whole.

The general principle followed in ascertaining costs is to make, according to a general schedule, a statement of cost of each department, process or step in a process; to assemble these into a total, and from the total to find the cost of the unit of product, such as ounce of precious metal, pound or ton of base metal.

[1] Cole, W. M., "Accounts," Houghton, Mifflin & Co., Boston, 1908.
Hatfield, H. R., "Modern Accounting," Appleton & Co., New York, 1909.

[2] "Cost Data," in Barr, J. A., "Testing for Metallurgical Processes," *Min. and Sc. Press*, San Francisco, 1910.

The total cost of treatment may be considered as being composed of the *prime*, the *general*, and the *administration* costs.

Prime (or flat, actual, direct) cost is made up of labor, fuel, material, supplies and repairs. Under material is classed any special substance that is regularly required for a specific operation; *e.g.*, the zinc necessary in the Parkes desilverizing operation; or the sulphuric acid in the vitriolization department of an electrolytic copper refinery. The heading supplies include tools and other incidentals that are obtained from the general store-room of the plant.

General cost includes the share the special operation has to bear of the expenses caused by interest, motive power, cartage, lighting, foremen, watchmen, miscellaneous labor, sampling, chemical laboratory, etc.

Administration cost includes salaries, expenses of office, of law, of advertising, traveling, purchasing, shipping, selling, taxes, rents, and some miscellaneous items. Each department of the plant has its share of the general burden apportioned to it.

Some works have only two divisions, prime and general costs; others have more than three heads for the classification of the whole subject of cost.

The items given under the several heads are not a complete list, nor is there any general fixed rule for the manner in which the items are to be classified. As long as a schedule is uniform and so arranged that it gives the inspecting person a clear insight into the details of cost, it fulfils its purpose.

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